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Pumped-Slurry Backfilling of Inaccessible Mine Workings for Subsidence Control

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

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PUMPED-SLURRY BACKFILLING OF INACCESSIBLE MINE WORKINGS FOR SUBSIDENCE CONTROL

by

Ralph H. Whaite[1] and Alice S. Allen[2]

ABSTRACT

In undermined urban areas, new solutions to subsidence problems are being sought. The Bureau of Mines is investigating a hydraulic backfilling technique whereby fill material is pumped as a slurry through a closed system and widely distributed in inaccessible mine workings from a single borehole. A full-scale demonstration was completed in 1973 in Scranton, Pa., which is underlain by abandoned anthracite mines in several coalbeds superposed one above another.

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. In the gravity-feed method formerly used to backfill inaccessible mine workings, quantities of fill injected per borehole averaged about 300 cubic yards. The gravity-feed method required many closely spaced injection holes and provided incomplete filling.

The new method, designed for inundated mines, proved successful also in mine workings about mine-water pool level. Distribution of slurry through a buried pipeline minimized disturbance to the community.

INTRODUCTION

The Bureau of Mines current investigations of hydraulic backfilling contribute to the national goal of minimizing adverse environmental effects of mining. Demonstrations of a new technique for hydraulic backfilling are part of a search for feasible and economic methods of protecting undermined land in urban areas from subsidence of the ground surface. Additional environmental benefits are the utilization of mine wastes, removal of waste banks that contribute to air and water pollution, and extinguishment of waste-bank fires.

[1]Mining engineer (now with Environmental Affairs Field Office, Wilkes-Barre, Pa.).
[2]Geologist.

In the United States, nearly 100,000 underground mines are in existence, of which an estimated 90,000 are inactive 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 percentage of the undermined land that has been affected by subsidence is not known accurately. In a recent study of land utilized by the mining industry, the surface area that had subsided or been disturbed by underground mining from 1930 through 1971 was estimated at 105,000 acres. Of this acreage, 84 percent resulted from mining coal (28).[3] The pressure to develop undermined land is increasing in several metropolitan areas in response to accelerating demands for more living space.

The past history of hydraulic backfilling of mine workings reflects greater involvement with active mining operations than with protection of the ground surface over abandoned mines. Although subsidence control is frequently one of the objectives in backfilling active mine workings (though to a lesser extent in metal mining than in coal mining), other purposes that contribute to the success of the mining operations have been of equal or greater significance.

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 (7, pp. 99-100). Hydraulic backfilling was developed during the late 1800's and in the 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 the First World War with the decline of the anthracite industry. In domestic bituminous coal mining operations, backfilling has not been a common practice (9). Applications to metal mining, however, in the United States and elsewhere, have provided solutions to a variety of ground control problems resulting in greater recovery and reduced mining costs (22).

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 common method in the United States (32, 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 large thicknesses of coal are being removed from beneath densely populated areas as in France and Poland.

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

In Germany and Great Britain, however, hydraulic backfilling was largely superceded by pneumatic backfilling (23, p. 13), and more recently the use of backfilling by either method has declined (10, pp. 14-21). In Great Britain, backfilling is considered an impediment to the type of mining necessary to make the coal industry economically viable, requiring multi-shift working of rapidly advancing, highly mechanized faces (16, pp. 122-123, 150). The cost of backfilling is also a factor that limits its use in coal mining operations.

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 protecting the communities. From time to time, studies of subsidence problems have been made for the city of Scranton (17, 21, 36) and for the Commonwealth of Pennsylvania (29, 31). All the studies recommended programs of hydraulic backfilling.

The Bureau of Mines interest in hydraulic backfilling is as old as the Bureau itself. The First Annual Report of the Director (18, pp. 42-43) describes ongoing study of mine filling (hydraulic backfilling) to reduce the annual waste of some 80,000,000 tons of anthracite coal, 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 (14). Other early Bureau publications reported on sources of backfill materials and applications of hydraulic backfilling to various mining problems. At the end of the Second World War, 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 (6). 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 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. A 1962 amendment (Public Law 87-818) to the 1955 Anthracite Mine Drainage Act (Public Law 162) authorized the Secretary of 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 the 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. To date, eight subsidence control projects have been completed under the Appalachian program and three additional projects are in progress, all in the anthracite region of Pennsylvania.

The long established method of hydraulic backfilling used in accessible mines where men can enter and safely perform controlled injection operations continues to provide satisfactory results. Existing blind flushing methods, however, provide only partial filling of workings in inaccessible mines. The subject of this report is a recent development in hydraulic backfilling that was designed for flooded or otherwise inaccessible mines. In the recently developed method, a dilute slurry is pumped through a closed system and widely distributed in underground openings through a single borehole. The Bureau of Mines undertook demonstrations of the new technique to assess the completeness with which inaccessible mine workings can be filled and to determine the costs. The new technique proved successful not only below the mine pool level but in the small area of workings that were above water level as well. Both fine sand and crushed mined waste were found to be suitable materials for use in the process.

During the past year, the Bureau of Mines arranged for a model study of hydraulic backfilling of mine cavities to be made by the Bureau of Reclamation at the hydraulic laboratory in Denver, Colo. The report on the model study is reprinted in this report as an appendix.

Further experimentation with the pumped-slurry injection process includes (1) an extension of the demonstration project in Scranton using coarser material crushed to minus 5/8-inch size, which will complete the removal of two waste banks, (2) a pilot project to backfill shallow beds above pool level in which caving is already well advanced, and (3) injection of burned mine refuse to extinguish a mine fire.

The purpose of this report is to record the experience gained to date by the Bureau of Mines in demonstrating the use of the pumped-slurry method in backfilling abandoned, inaccessible mine workings. Because of pertinence to the evaluation of subsidence potential and to the planning of backfilling operations, subsurface conditions at the demonstration sites are described in terms of geologic setting, hydrology, and past mining operations. Some background information on hydraulic backfilling is included in the hope of making the report meaningful to readers with a variety of backgrounds, including those with responsibility for urban planning. Specific data on injection operations are included in tables and diagrams to provide technical information on the pumped-slurry process in terms of elapsed times, injection rates, quantities, and disposition of fill material. Elements of cost are identified. A preliminary evaluation of the pumped-slurry method is based on limited experience to date.

ACKNOWLEDGMENTS

The contributions of many individuals to this report and to the demonstration projects on which it is based, are gratefully acknowledged. Donald L. Donner, mining engineer with the Denver Mining Research Center, Denver, Colo., was the Bureau Project Officer on the demonstration project at Rock Springs, Wyo. Firsthand observations of the Rock Springs injection and the exploratory phase of the Scranton project were contributed by Gary Colaizzi, mining engineer. Valuable subsurface data and interpretations were provided by Frank Morgando, engineering geologist of the Wyoming Highway Department.

Charles S. Kuebler, Chief of the Environmental Affairs Field Office at Wilkes-Barre, Pa., was in charge of the Bureau of Mines activities at the Scranton demonstration project. Support activities including daily monitoring were provided by William W. Everett, supervising mining engineer. Carl D. Sauer, mining engineer of the Environmental Affairs Field Office, was responsible for interpretations of the mine maps, selection of borehole drilling sites, and computations of the lateral and vertical extent of emplaced fill material. Operational data were compiled and systematically reported by Gerald F. Durkin, Bureau representative at the Scranton site for the duration of the project. Paul H. Struthers, chemist, of the Division of Environment, assisted in compilation of slurry injection data for the project in Scranton.

Jerrald R. Hollowell, ground-water geologist with the Susquehanna River Basin Commission and formerly with the U.S. Geological Survey, furnished information on the ground-water conditions at the demonstration project site in Scranton, Pa.

Injection operations at both the Rock Springs, Wyo., and Scranton, Pa., demonstration projects were performed by the Dowell Div. of the Dow Chemical Co. of Houston, Tex. The Rock Springs demonstration was carried out in cooperation with the Bureau of Mines, the Department of Housing and Urban Development, and the city of Rock Springs. Injection at the Scranton project was done under contract to the Bureau of Mines. Dowell Div. personnel who contributed to the success of this nonroutine operation were L. D. Boughton, project manager; D. E. Campbell, senior design engineer; A. E. Steinkirchner, who supervised the plant site and provided daily logs of the injection operations; and A. J. Myers, who was responsible for the sonar surveys.

Michael J. Naples, Jr., partner in the Empire Contracting Co. of Old Forge, Pa., which performed the crushing and drilling at the Scranton project, contributed helpful suggestions based on his wide experience in backfilling operations in the anthracite region.

HYDRAULIC BACKFILLING METHODS

Two methods of hydraulic placement of backfill material in underground mined-out spaces have 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 by jets of water.

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, large 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.

Blind Flushing

Many abandoned mine openings are inaccessible because of flooding or extensive caving. Such openings must be flushed blindly. In the past, the gravity-feed method has been used as in controlled flushing, but the injected granular material 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 1. Depending on conditions underground, such as the dip of the bed, its height, and the proximity of pillars or caved beds, the volume of material that can be injected in 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 can not 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.

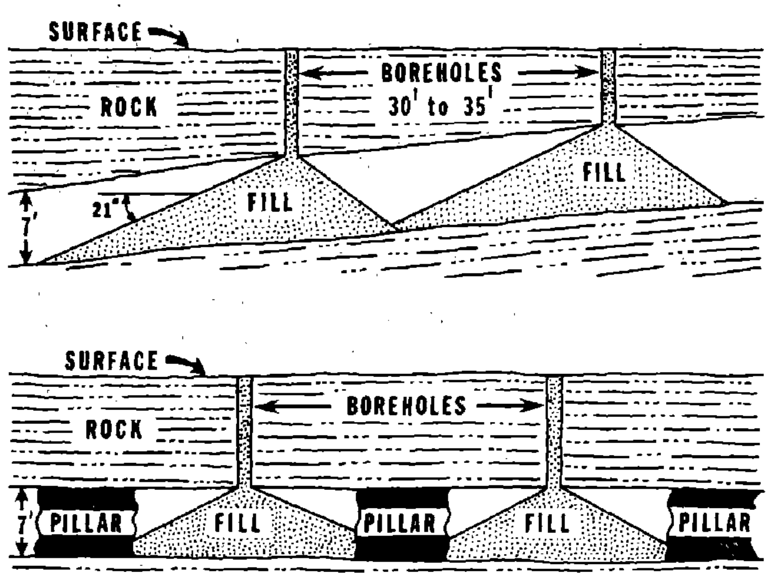

FIGURE 1. - Typical sections showing boreholes and configuration of fill material placed in mine voids by the gravity blind flushing method.

Pumped-Slurry Injection

A new technique for the blind flushing of inaccessible mine workings has recently been developed (19, 34). This technique differs from the open gravity-feed methods previously described in that energy is used to achieve a dynamic suspension of solid particles and the system is completely closed from the point of slurry mixing to the bottom of the injection hole.

In the pumped slurry 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. Figure 2 is a photograph taken during a model demonstration of the process by

FIGURE 2. - Close view of outer slope of mound of solids being deposited in a glass tank. The mound is being extended to the left as solid particles settle in deeper water.

(Courtesy of the Dowell Div. of the Dow Chemical Co.)

the Dowell Div. of the Dow Chemical Co.[4] 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 3.

In a table-size mine model simulating the arrangement of rooms and pillars, deposition of fill material is shown in figure 4. 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 into 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.

[4] Reference to specific company names is made for identification only and does not imply endorsement by the Bureau of Mines.

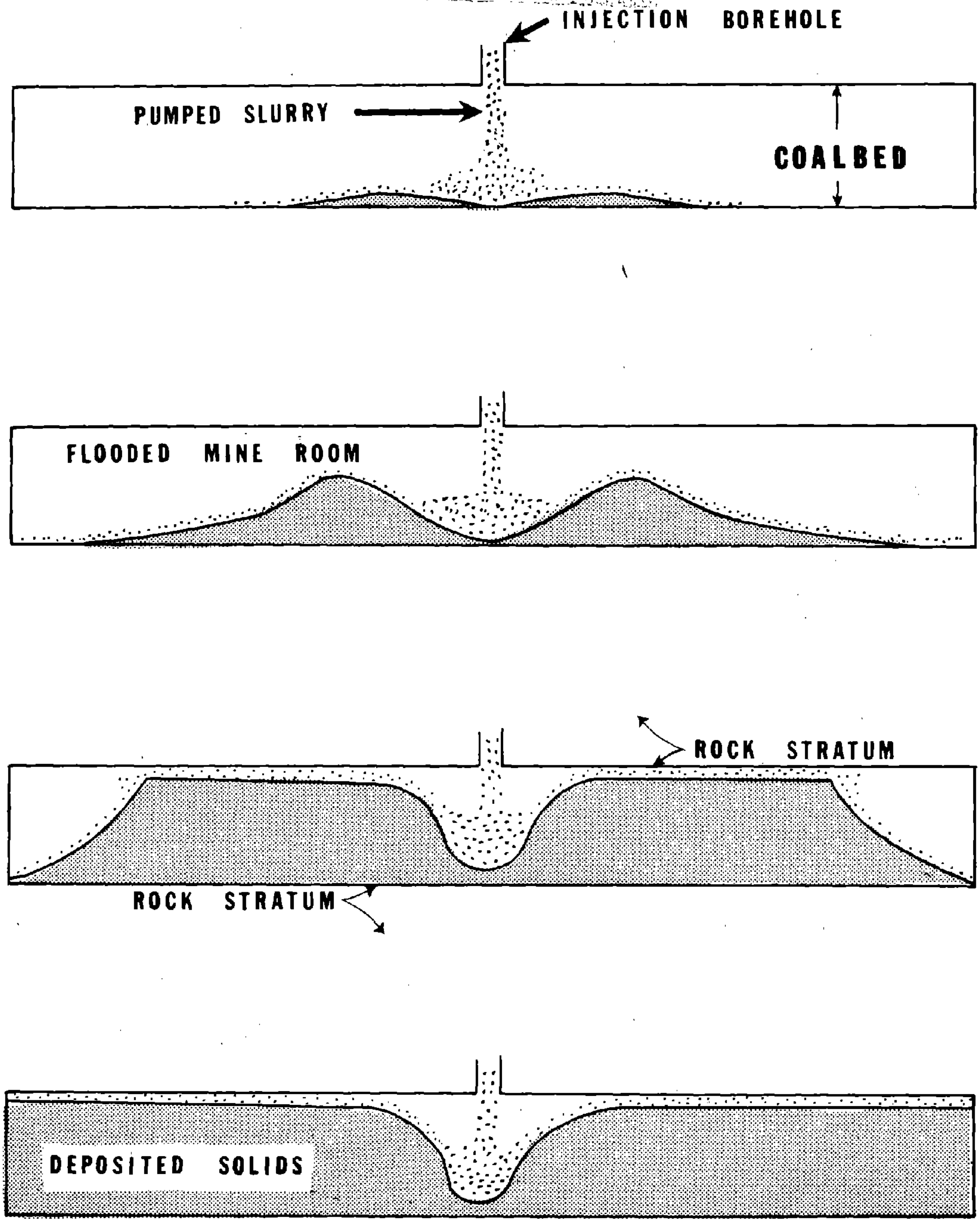

FIGURE 3. - Sectional views through a flooded mine room at the point of slurry injection showing movement of particles and growth of deposit.

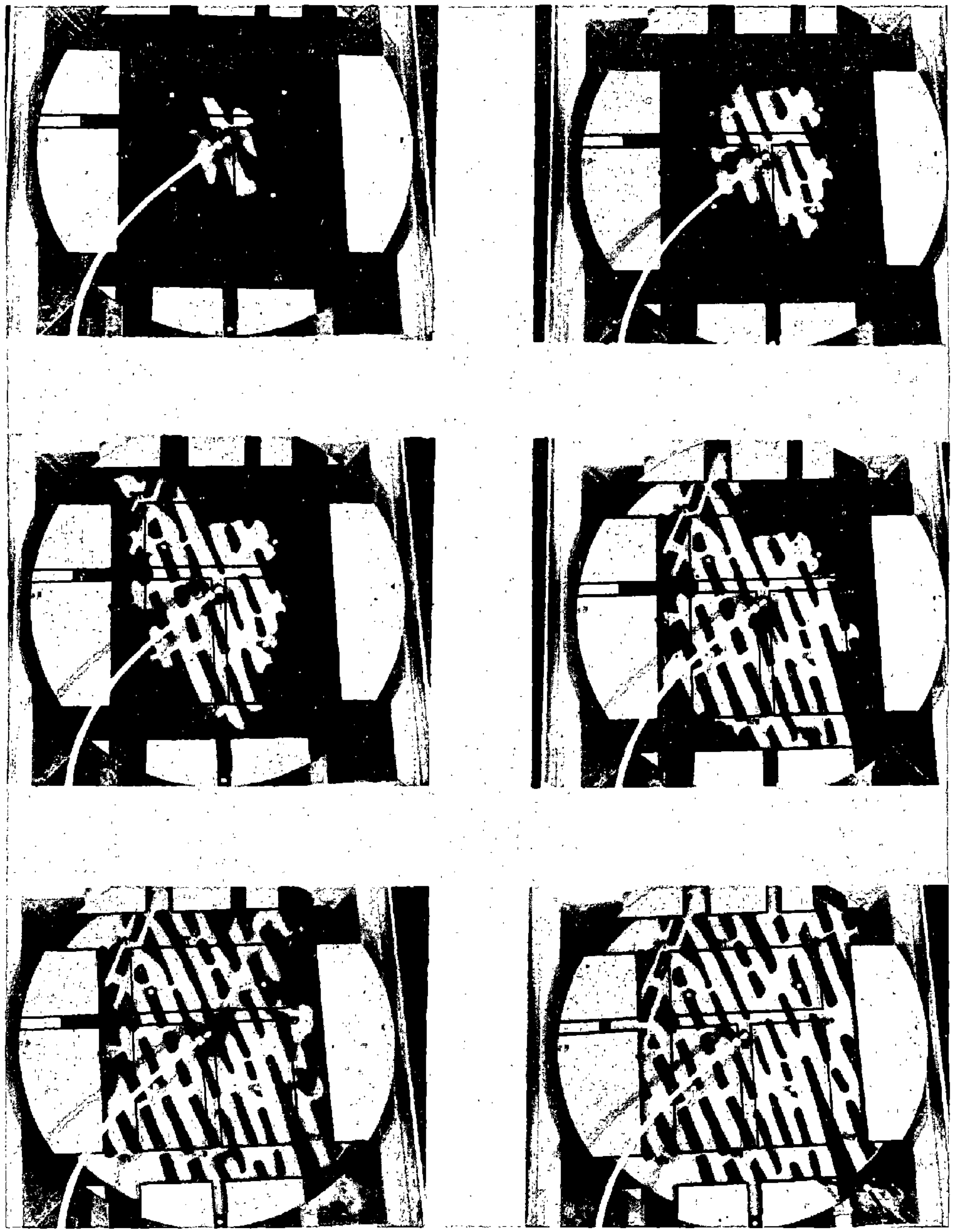

FIGURE 4. - Top view of mine model with transparent roof, showing stages of radial distribution of fill material by pumped-slurry process.

(Courtesy of the Dowell Div. of the Dow Chemical Co.)

DEMONSTRATION AT ROCK SPRINGS, WYO.

Subsidence Problem

The pumped-slurry injection process was first demonstrated in a test area in Rock Springs, Wyo., adjacent to an area in which subsidence problems had developed. 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 (13). Severe subsidence damage had been experienced since 1967 within a 2-acre area in the eastern part 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" had appeared in the surface of the ground from time to time.

Subsurface Conditions

Geologically, the city of Rock Springs is located on the west edge of the Rock Springs Uplift (20, 33). Bedrock in the area is the Rock Springs Formation of Cretaceous age--a repetitive sequence of coalbeds, carbonaceous shale, siltstone, claystone, and sandstone. The strike of the bedrock formations is N 36° E, and the dip is 5° to the northwest at the site of the subsidence. Downdip 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.

Depth to bedrock ranged from 6 to 52 feet in boreholes that were drilled in the test 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. In the Rock Springs subsidence area, however, the evidence indicated that collapse over mine openings was the principal cause of the subsidence (25). 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.

Mining has been conducted in the Rock Springs coalfield for about 100 years, and annual production exceeded 2,000,000 tons for almost one-half that time (13). Presently, only one small underground mine is operating in the area. Market conditions, rather than available minable reserves, have dictated the limited production. Mining activity under the City of Rock Springs ceased about 35 years ago.

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 due to minor faulting and changes in mining techniques over the years.

Water presented a considerable 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 recent drilling indicated that the mpas 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.

Selection of Subsidence Control Method

To backfill the entire undermined area of the city would be difficult and expensive. Support in the form of backfill was recommended for designated areas in which the strata above the mines are less than 300 feet thick and less than 50 percent of the coal had been left as pillars. Areas where the pillars had been removed were not considered for backfilling on the assumption that such areas had subsided shortly after mining and before buildings had been constructed (13).

Controlled flushing, the most effective method of backfilling, was not considered for the Rock Springs area because of the flooded and caved conditions in the old workings. Dewatering was ruled out because of the expense and the possibility that movement of water and admission of air might disturb the existing equilibrium.

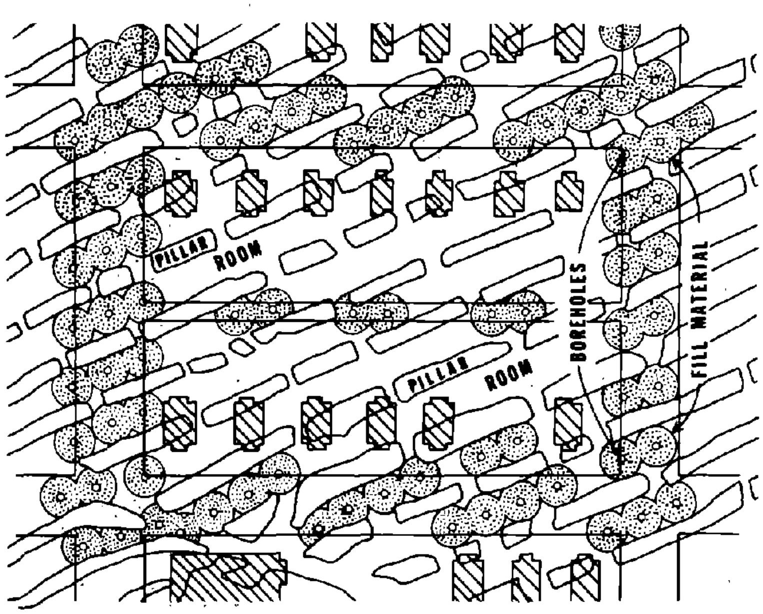

FIGURE 5. - Typical residential block in Rock Springs, Wyo., showing the pattern of boreholes in streets, alleys, and vacant lots proposed for blind flushing. Circular areas around boreholes represent backfill material to be placed in the mine voids.

The only method of blind flushing then in use was under consideration in 1969 for designated parts of the city with a realistic appraisal of the difficulties anticipated. The closely spaced injection holes required by the method (averaging 45 per city block) and the lack of direct support to the houses are illustrated in figure 5.

In 1970, a proposal to try the pumped-slurry technique of blind flushing was considered favorably because of the claim that nearly complete filling of the mine voids could be accomplished by injection through a single borehole. A project to demonstrate the feasibility of the new method was planned and carried out in a residential section of Rock Springs adjoining the area of active subsidence. The area selected was believed to have a high subsidence potential, but subsidence had not yet become apparent. The flooded condition of the mines was favorable to the method. Exploratory drilling in conjunction with sonar scanning indicated that the site 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. A source of sand suitable for use as fill was available nearby, and an ample supply of water for the slurry mix could be pumped from the mine pool.

Injection Operation

The test objective of the Rock Springs 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, available from a nearby deposit of wind-blown origin. 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 (8). Two submersible water pumps, each with 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.

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

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 pounds 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 6, the heavy line encloses the distribution of fill as determined by the

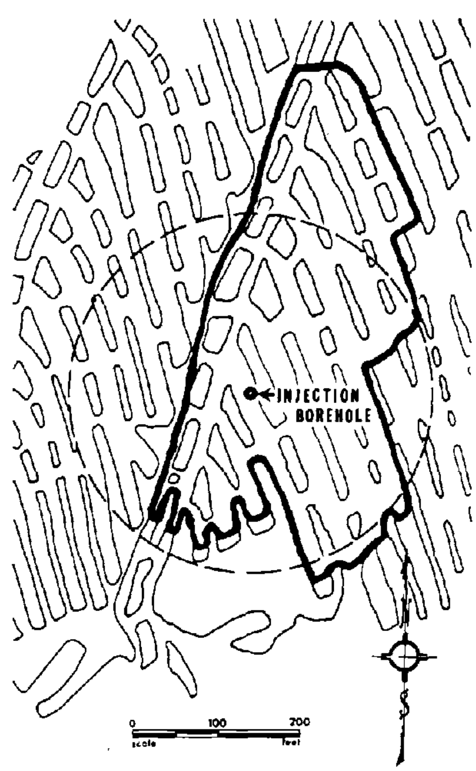

FIGURE 6. - Map of mine workings at the site of the demonstration project in Rock Springs, Wyo., showing pattern of sand deposition in the mine voids. Dashed circle shows area of planned backfill.

evaluation team on the basis of all available subsurface data. The circle represents the area with 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 man-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 drillholes, sand was found interbedded with the caved rubble; in others, the sand was in the 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 with distance 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 (26).

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 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.

DEMONSTRATION PROJECT IN GREEN RIDGE SECTION OF SCRANTON, PA.

Selection of Project Site

Scranton was selected as the locale for the full-scale demonstration project 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 that was carried on 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 in Scranton, and most of the central part of the city overlies 6 mined beds.

The location and amount of subsidence at the surface and the time lag between mining and subsidence are difficult to predict because of the many factors involved. The stability of a piece of property is determined by the mining operations that have been carried on below and their interaction with the natural geologic conditions at that site.

Most of the anthracite mining was by the room-and-pillar system. Extraction was accomplished in one, two, or three distinct phases known as first, second, and third mining (37). In first mining, 40 percent or more of the coal was left as pillars to support the overburden. This percentage increases with depth. Second mining consisted of splitting pillars, either lengthwise or crosswise, and taking a slice off the entire length. The resultant size of the remaining pillars was generally smaller than is believed necessary to effectively support the overlying strata. In third mining, locally known as "robbing," the pillars were extracted as completely as could be done while insuring safe exit of the miners. Third mining induced complete caving of the overlying strata. Hydraulic backfilling was employed selectively by some mining companies to provide support that would permit extraction of pillars (35).

In Scranton, the mines have been abandoned, and the continuing but sporadic problems of subsidence have been inherited by present owners of surface property. Through cooperative efforts of the State and Federal governments, four subsidence-control projects have been completed in Scranton since 1965, and others are in progress.

The demonstration project site is in an area of potential subsidence owing to undermining in five coalbeds. Subsidence has not yet become apparent at the ground surface in this area, and caving below ground was not believed to be sufficient to block the effective movement of slurry.

The locations of the anthracite region and the demonstration project are shown in figure 7. Scranton is located in Lackawanna County, on the Lackawanna River. The specific site selected for the demonstration project was in the Green Ridge residential area, about 1-1/2 miles northeast of the downtown section, as shown in figure 8. It is approximately 3/4 mile east of the river, near the base of the slope that forms the eastern side of the river valley. The area slopes gently toward the river.

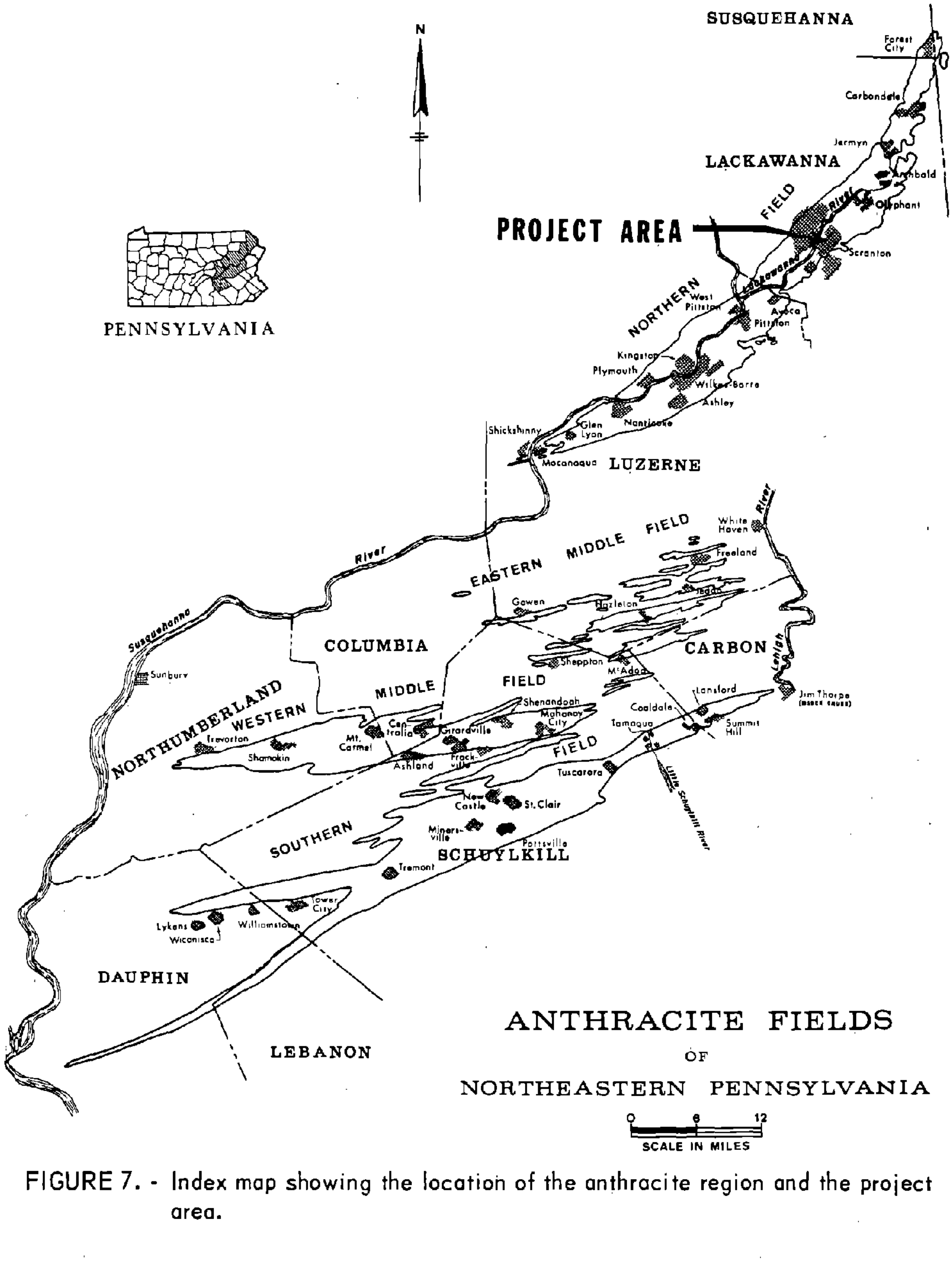

FIGURE 7. - Index map showing the location of the anthracite region and the project area.

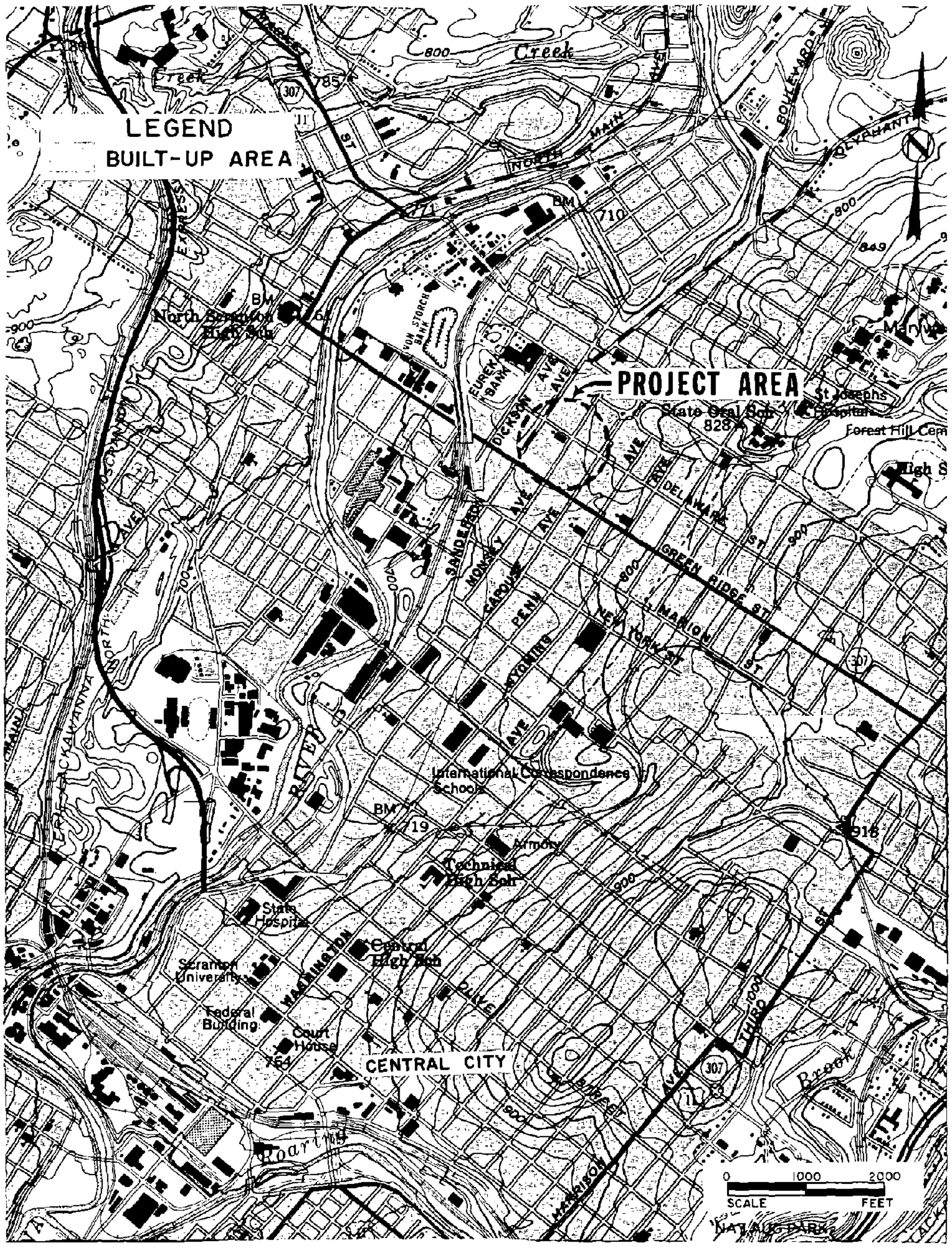

FIGURE 8. - Map of the Green Ridge section of Scranton showing the location of the hydraulic backfilling demonstration project in relation to streets and the Eureka refuse bank.

The demonstration project area is in a well established residential neighborhood of houses surrounded by lawns as shown in figure 9. One hundred and forty private dwellings and two churches wholly or partly overlie the 30-acre area that was backfilled, which represents a population of about 600 people. The property was conservatively valued at more than $3,000,000, not including streets and utilities. The site is adjacent to the Eureka refuse bank, which was made available as the source of crushed mine rock, thus eliminating the cost of purchase or transportation of fill material for backfilling. Figure 10 is a map of the Green Ridge demonstration project area that shows the locations of individual houses and streets.

Subsurface Conditions

Geologic Setting

Immediately beneath the surface in the project area are deposits of sand and gravel that are part of the alluvial fill of the Lackawanna River Valley (11, pl. 1). Boreholes drilled in the project area encountered the top of rock at depths ranging from 13 to 44 feet. The average thickness of the alluvium is 30 feet.

The coal-bearing bedrock sequence underlies the alluvium. The coal measures, roughly 400 to 500 feet thick in the area, (12, pl. 10), are mainly sandstone, siltstone, and shale. Structurally, the coal measures form a northeast-trending canoe-shaped basin, fairly flat in the central part and rising more steeply along the valley sides. The basin is a synclinorium that includes subordinate folds, some parallel to the main basin axis and others oblique to it. The project area is located 3/4 mile east of the main basin axis; the dip of the beds is 3° to 4° to the northwest. The axis of an oblique-trending minor fold (the WSW-trending "Green Ridge Anticlinal") passes within 1,000 feet to the north of the project area (30).

Beneath the Green Ridge area are seven coalbeds, as shown in figure 11, that lie at depths ranging from 22 to 440 feet below the surface. In descending order, the coalbeds are--

Big (or Fourteen-Foot),

New County,

Clark,

No. 1 Dunmore,

No. 2 Dunmore,

No. 3 Dunmore,

and No. 4 Dunmore.

FIGURE 9. - Oblique aerial photograph of the Green Ridge section of Scranton, Pa., with the Eureka and Von Storch refuse banks on the near and far sides, respectively, of the railroad tracks. The dashed outline encloses the backfilled area.

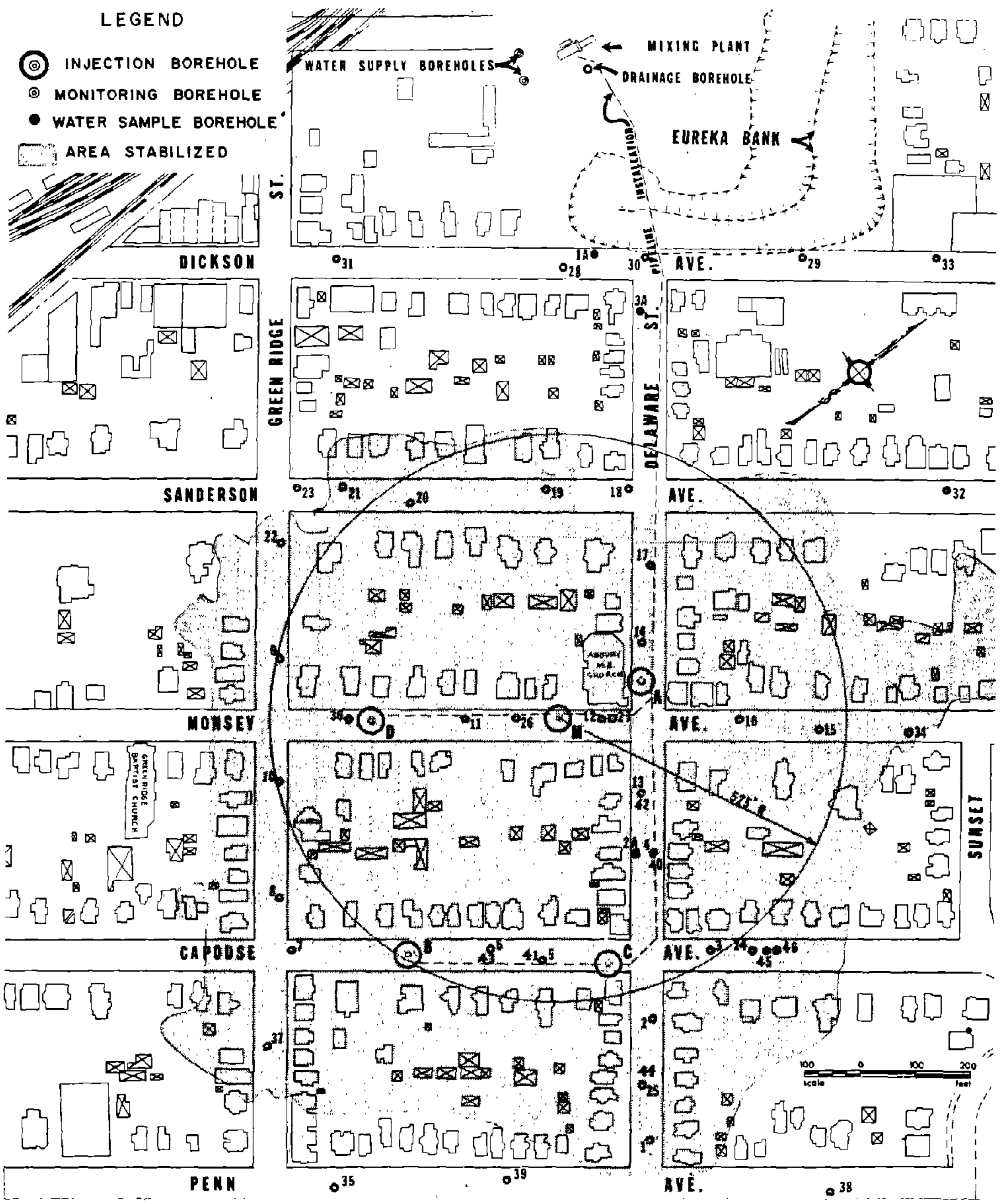

FIGURE 10. - Surface map of Green Ridge demonstration project area.

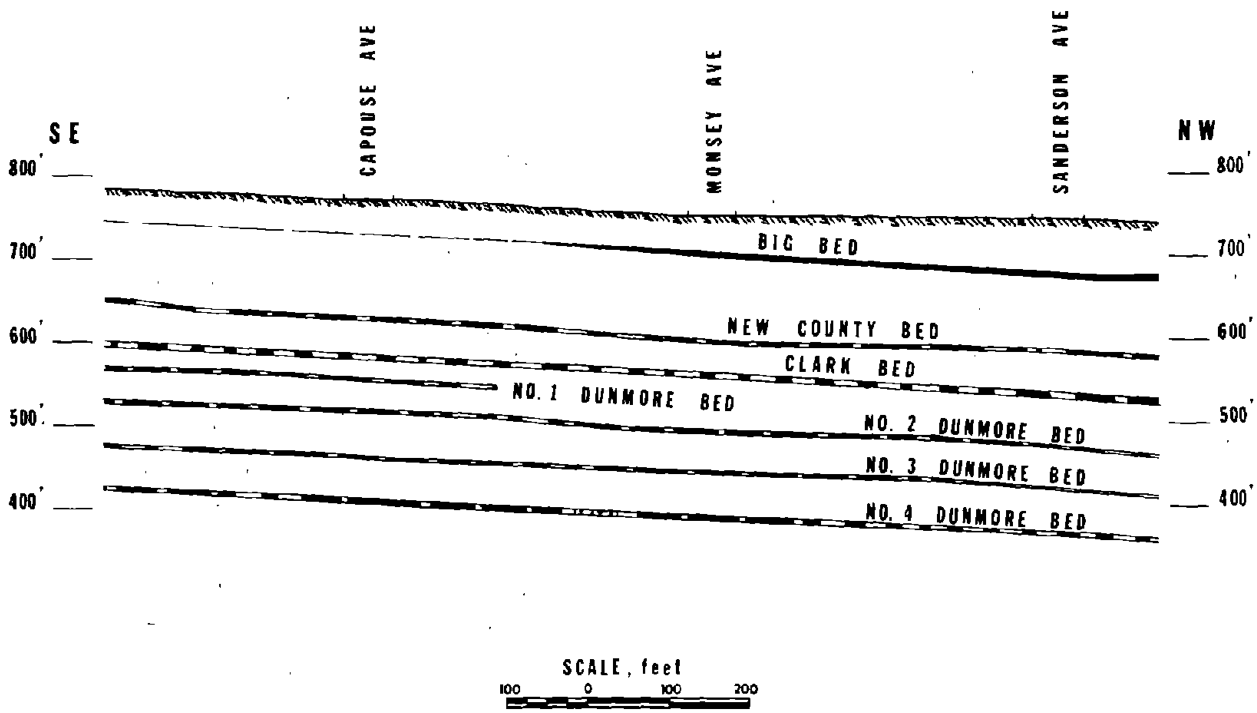

FIGURE 11. - Cross section through Green Ridge demonstration project area along mine boundary parallel to, and 250 feet southwest of Delaware Street.

The No. 1 Dunmore bed is discontinuous and occurs only near the southeastern edge of the project area. In the same area, the Big bed becomes very thin so that a total of six coalbeds are of minable thickness.

The uppermost coalbed, the Big bed, is the thickest, ranging up to 14 feet thick and averaging nearly 10 feet. In the sourthern part of the project area, the Big bed is missing owing to erosion before the alluvial fill was deposited. The Big bed has not been mined in the immediate area because the rock cover is thin, ranging from 4 to 33 feet. The depth of the Big bed beneath the surface ranges from 22 to 63 feet and alluvial material occupies from 13 to 44 feet of this interval. In the Northern anthracite field, the dangers of mining beneath saturated alluvium, where the rock cover between alluvium and the coal is thin, are well known (1). Failure of rock cover causes an inrush of water and sediment into the mine and subsidence at the ground surface. Bureau engineers have advocated a rock cover 50 feet thick to allow for the possible presence of potholes (37).

The other coalbeds in the project area are discussed in the section "Results of Mining."

The water table in the project area has been modified by mining operations and will be discussed in the section "Post-Mining History."

Results of Mining

In the immediate area of the demonstration project, five of the six underlying coalbeds have been mined. The primary source of information on the mining was the file of mine maps maintained at the Bureau of Mines Environmental Affairs Field Office in Wilkes-Barre, Pa. Supplementary information appeared in studies sponsored by the city of Scranton in 1911 (17), in 1936 (36), and in 1960 (21). The most recent subsurface information was provided by the drilling of boreholes in 1971 in support of the current demonstration project.

The project area includes mine workings that originally belonged to two collieries. Figures 12 and 13 are mine maps of the two beds that were backfilled during the demonstration project. The mine maps show the remnants of the barrier pillars in each bed that once separated the mines, and the difference in orientation of rooms and pillars of the two mines. The former barrier pillars were parallel to, and 250 feet southwest of, Delaware Street. Later, the two mines were operated as one mine, and rooms were driven through the columnized barrier pillars in both the New County and Clark beds.

New County Coalbed

The New County coalbed is the uppermost of the two mined beds that were backfilled in the current project. Depths to the New County bed range from 108 to 153 feet below the ground surface within the project area. The average thickness is 6 feet, and about 64 percent of the coal was extracted. In the mine southwest of the old barrier, mining was reportedly prohibited in the New County bed in old leases. By 1911, extraction of one-third was permitted, leaving two-thirds as pillars (17, p. 47). The mine map shows that these pillars were later split, resulting in pillars as shown on figure 13. In this case, the second mining resulted in support comparable with that of the first mining in the adjoining mine. Third mining was shown only in the area occupied by the refuse bank, outside the area of planned injection.

Along three of the east-west haulageways that cross the area diagonally, the mine map showed the presence of permanent brattices in all openings (crosscuts) in a given line of pillars.

Within the project area, the depth to the New County bed falls within the depth range of the fluctuating mine pool level. The altitude of the base of the New County bed ranges from 583 to 655 feet above sea level within the area of backfilling, and the mine pool level has fluctuated between 608 and 630 feet above sea level since January 1970. In January 1972, about half the mine workings in the New County bed were above water whereas most of them were inundated in June 1972 when heavy rains accompanied Tropical Storm Agnes.

Clark Coalbed

The Clark is the lower of the two mined coalbeds that were backfilled in the demonstration project. The depth to the Clark bed ranges from 162 to 207 feet beneath the ground surface in the project area. The average

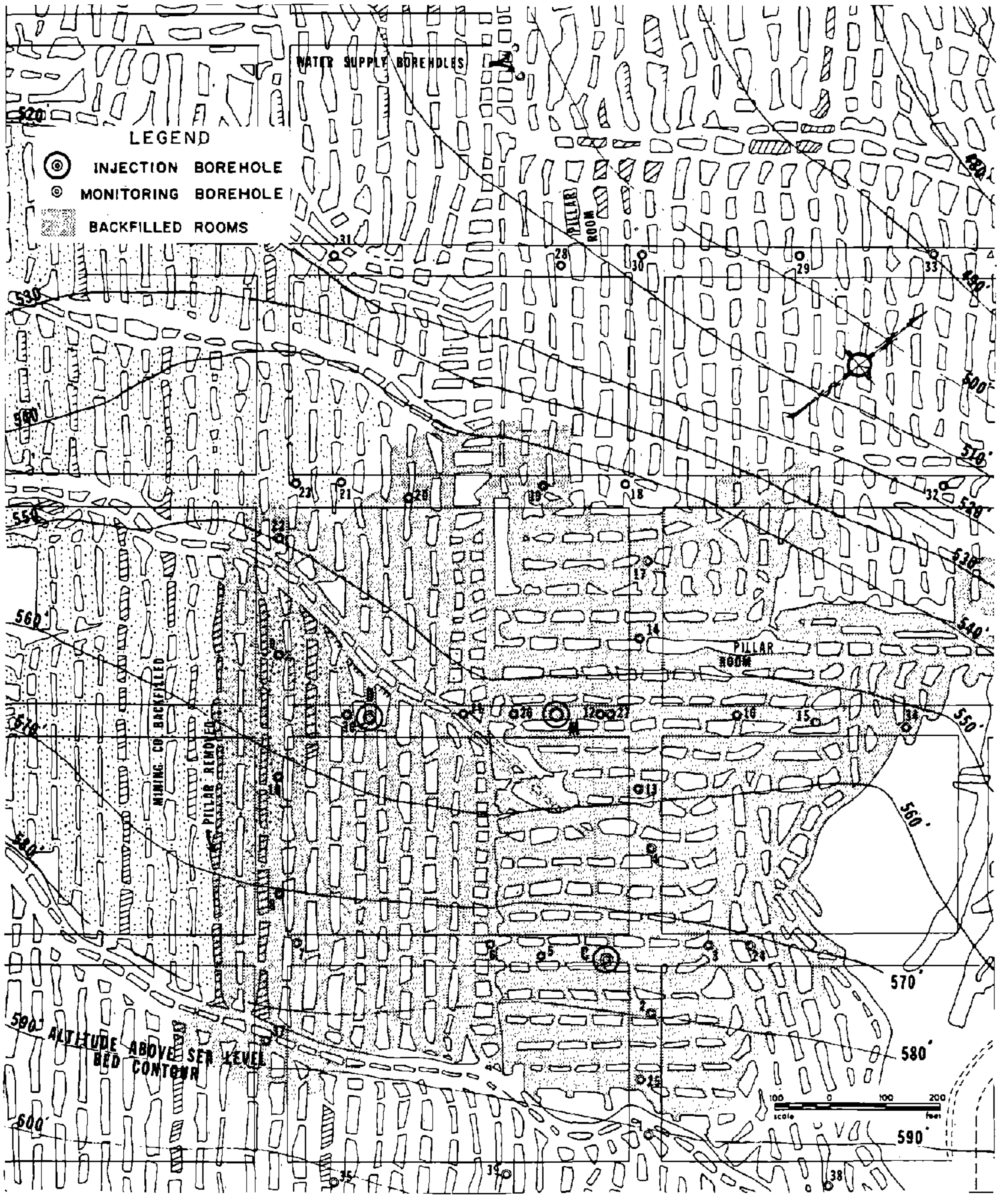

FIGURE 12. - Mine map of Clark bed in demonstration project area showing extent of emplaced backfill.

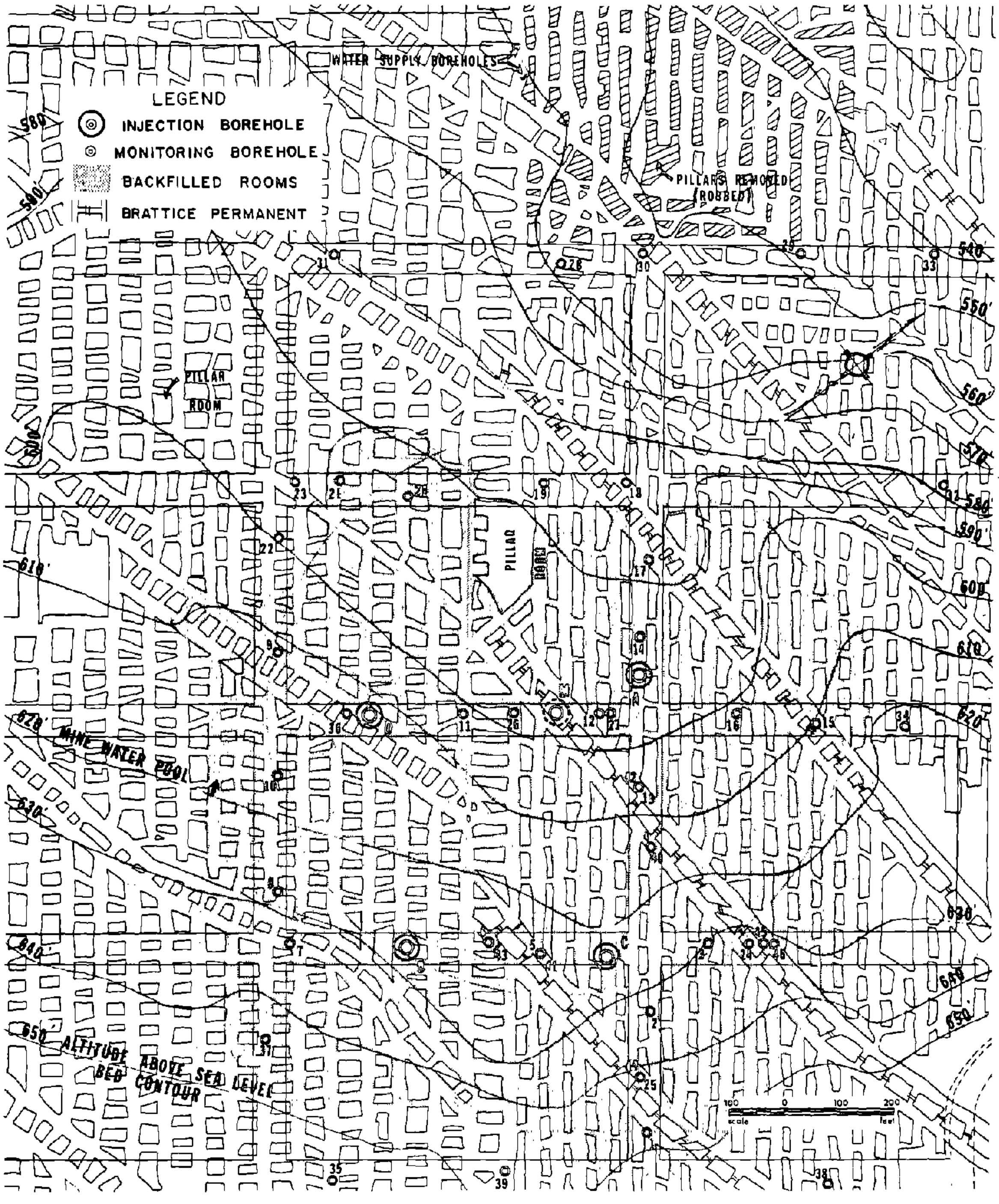

FIGURE 13. - Mine map of New County bed in demonstration project area showing extent of backfill.

thickness is 8 feet, and approximately 69 percent of the coal was extracted. The interval between the Clark and New County beds ranges from 41 to 62 feet in thickness. All workings in the Clark bed in the project area are flooded.

Mining in the Clark bed, in the mine northeast of the old boundary, was done in the 1880's, which represents the oldest mining in the project area. The pillars have fairly regular alinement, but they vary in width (from 10 to 28 feet) and in shape (fig. 12). Southwest of the old boundary, the mining pattern is regular.

In 1911, Griffith and Conner made an underground inspection of the conditions then existing in mines beneath the city of Scranton. They noted a bench in the middle part of the Clark bed that was considerably softer than the remainder of the bed (17, pp. 46-48). In many pillars in the Clark bed, the coal in the middle bench had chipped or flaked off. The chipping and flaking were interpreted as the result of exposure to air, and in some cases, as early stages of squeeze, which is often the first indication of subsidence in a mine.

During the 1930's, a few pillars along the southwest perimenter of the project area were removed. The mine map shows some old backfilling by the mine operator in this area, but one of the sonar surveys made in the course of the present investigation revealed that a passage in this area is still open. Pillars were removed also beneath the refuse bank.

Dunmore Coalbeds

The four Dunmore beds--No. 1 Dunmore, No. 2 Dunmore, No. 3 Dunmore, and No. 4 Dunmore at average depths of 230, 270, 320, and 370 feet, respectively (fig. 11)--were mined between 1900 and 1929. In most of the project area, No. 1 Dunmore bed is missing and No. 2 Dunmore bed is about 60 feet below the Clark bed. No. 1 Dunmore bed, which was mined in only a small segment along the eastern perimeter, is about 25 feet below the Clark bed and 35 feet above the No. 2 Dunmore. The mine map showed that the pillars in the No. 1 and No. 2 Dunmore beds were columnized, which improves the stability of mined beds that are separated by a small thickness of rock strata.

The Dunmore coalbeds are thin, averaging between 3 and 4 feet thick. In the course of mining, several feet of rock from the bottom or the roof were removed in all the gangways to make room height for cars and mules (17). Waste rock was stowed in the mined-out rooms.

Post-Mining History

In formulating plans for backfilling abandoned mines, information on changes that have taken place since the mines were abandoned is needed in addition to the geometry of the mine workings as recorded on mine maps. Such changes include caving of the mine roof, failure of pillars, and the effects of changes in the ground water regimen. Both the incidence of subsidence and backfilling operations for controlling subsidence are affected by deterioration of the mine workings and by changes in water levels. Subsurface information from drilling of boreholes, sonar surveys, and geohydrologic investigations were utilized in assessing the present conditions of the mine workings at the Green Ridge project site.

Deterioration of Mine Workings

Subsurface information developed by drilling revealed little evidence of caving except in the vicinity of the refuse bank where third mining operations had removed pillars in several coalbeds. These caved beds are beyond the limit of backfilling. At the horizon of each coalbed penetrated by the drill, the logs recorded the height of open space and the thickness of broken rock or solid coal that was encountered. The conditions in many of the boreholes are shown graphically in figures 14-16. Of 41 boreholes drilled for various purposes to the Clark bed (the lower bed that was backfilled), 21 boreholes encountered open space with no broken rock. Open space ranging from 2 to 6 feet high and underlain by broken rock was reported in 11 other boreholes. In two boreholes, 1 foot of open space above broken rock was reported. In the upper (New County) bed, 13 of 53 boreholes encountered open space with no broken rock, and 26 boreholes encountered open space from 2 to 6 feet high underlain by broken rock.

Sonar caliper surveys were made in 25 monitor holes at the levels at which they intersected the New County and Clark beds. The sonar surveys were made to test the usefulness of the sonar method in determining whether mine workings are open or obstructed by caving. The use of this device to survey underground mine workings was a departure from its intended uses for surveying underground cavities in connection with solution mining and gas storage, and for recording deviation of drilled shafts.

The sonar caliper was originally designed for, and is best adapted to, use in liquid-filled cavities. A tool was adapted for use in dry cavities to survey the part of the mine workings in the New County bed that was above the mine pool level. This tool requires a perfectly reflective surface, and its beam has a 22° spread as compared with the 4° beam used in a liquid. The results of the surveys made in the dry workings were difficult to interpret.

The sonar surveys were made by lowering the sonar caliper tool down each borehole and suspending it at the depth of the mine roof, after the hole had been checked for alinement, possible blockages, and water elevation. The downhole tool is approximately 72 inches long and 3-5/8 inches in diameter, permitting it to pass through a clean 4-inch-inside-diameter pipe. The tool emits an intermittent sound pulse (every 2-1/4° as the tool rotates at 2 rpm) that is directionally focused into a narrow beam. The sound wave travels through the fluid--in this case mine water or air--until it strikes the cavity wall and is reflected back (27). The air tool was calibrated at the surface, and the water tool was calibrated in the hole in which it was to be used. At each elevation at which the tool is positioned, it rotates through a 360° arc; magnetic north is identified by a special pulse marker (the directional pulse was lost in some holes owing to the position of the casing pipe). The tool is then lowered by increments of 1 foot (or other suitable interval), stopped, and rotated to develop an oriented three-dimensional model of the cavity.

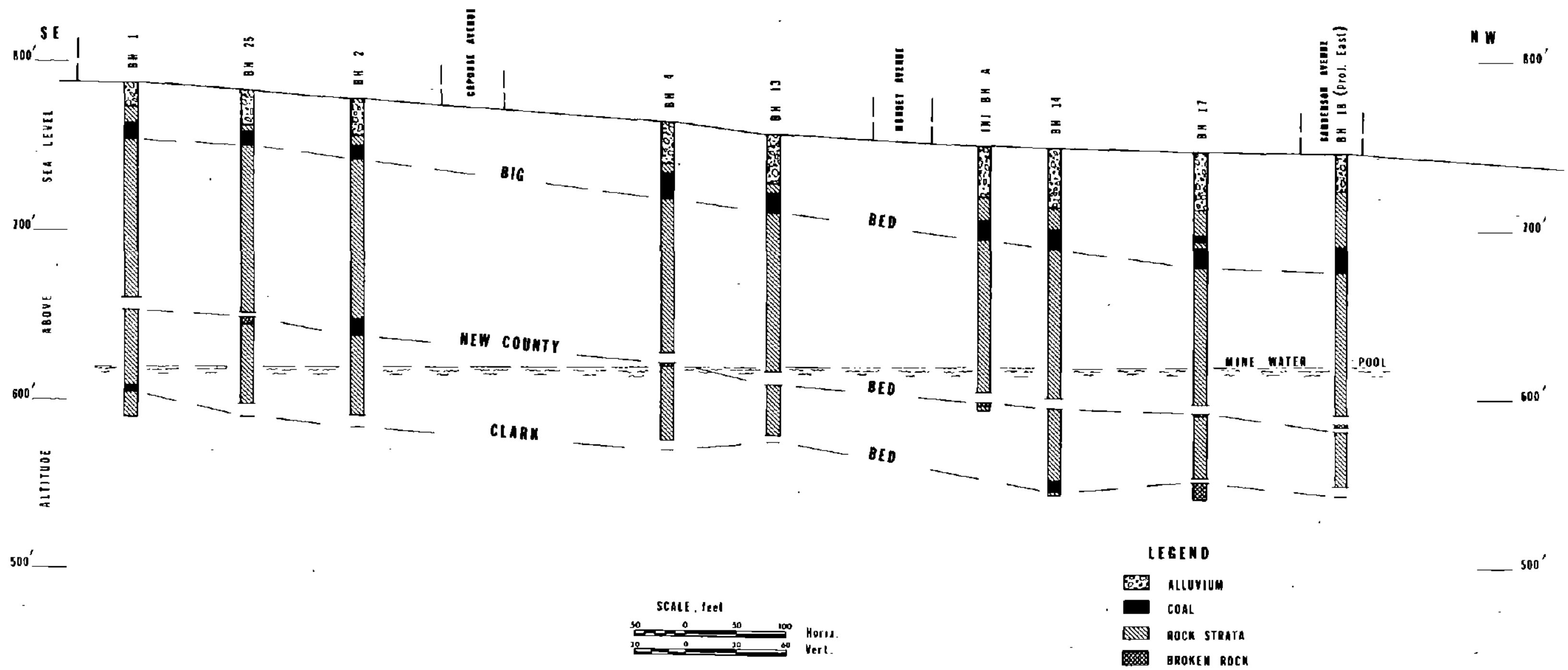

FIGURE 14. - Section along Delaware Street showing boreholes.

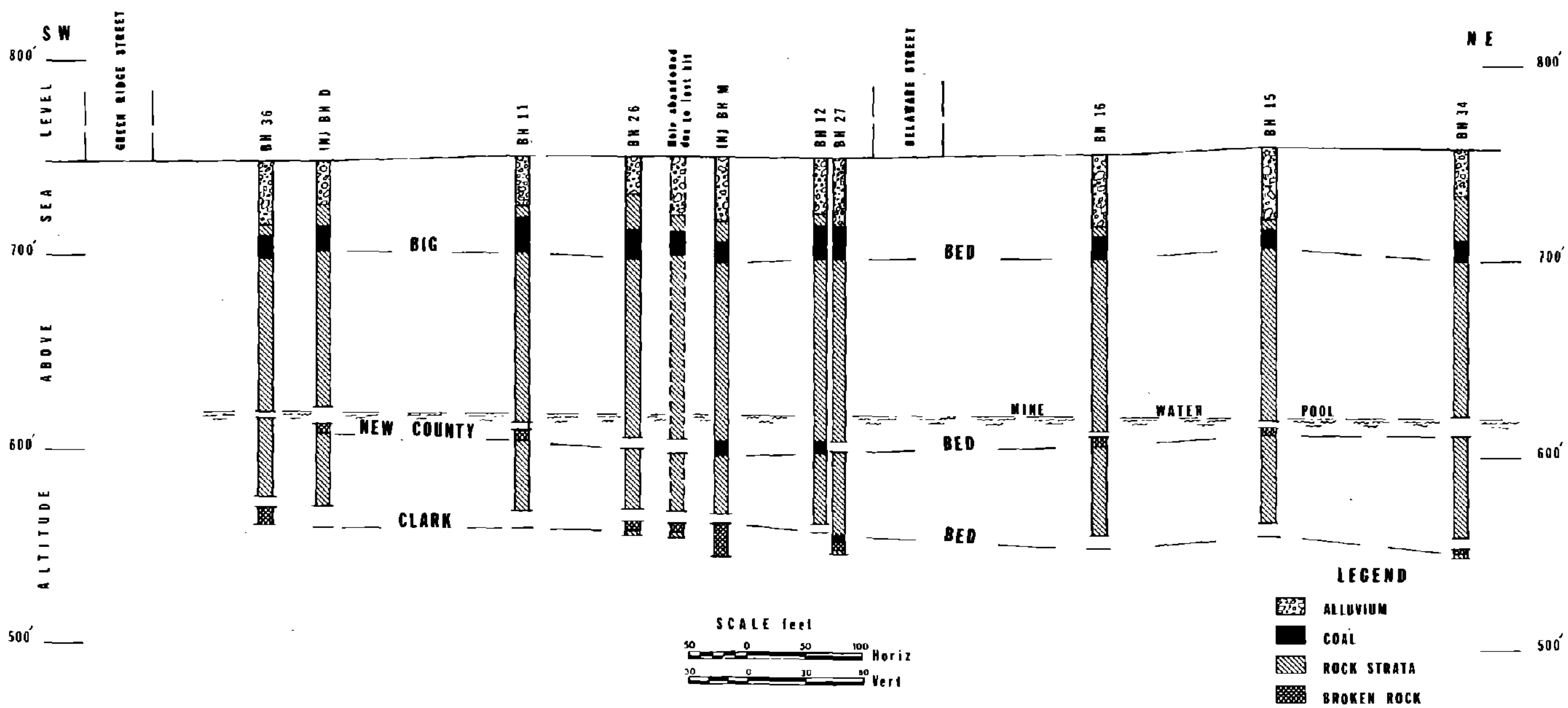

FIGURE 15. - Section along Monsey Avenue showing boreholes.

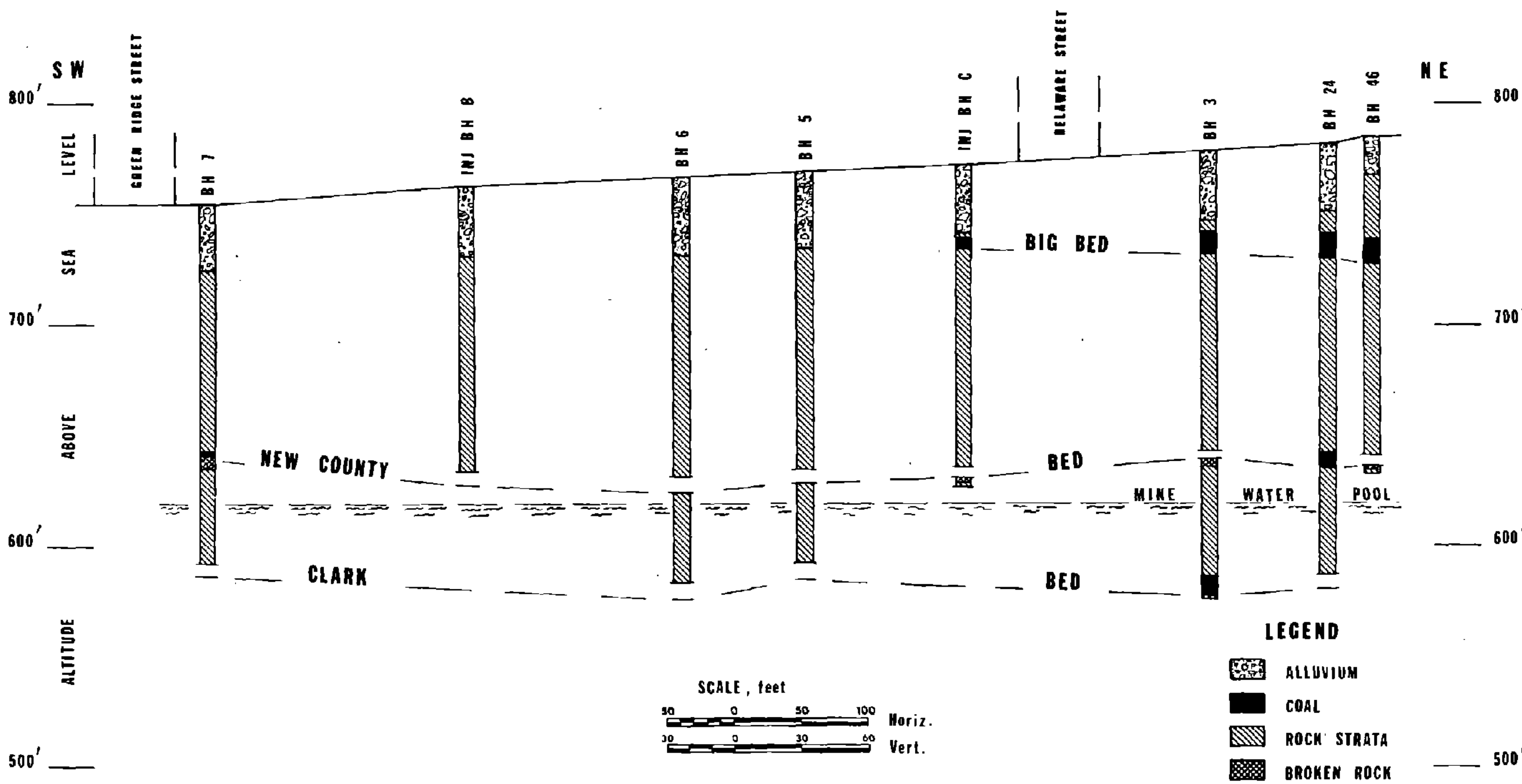

FIGURE 16. - Section along Capouse Avenue showing boreholes.

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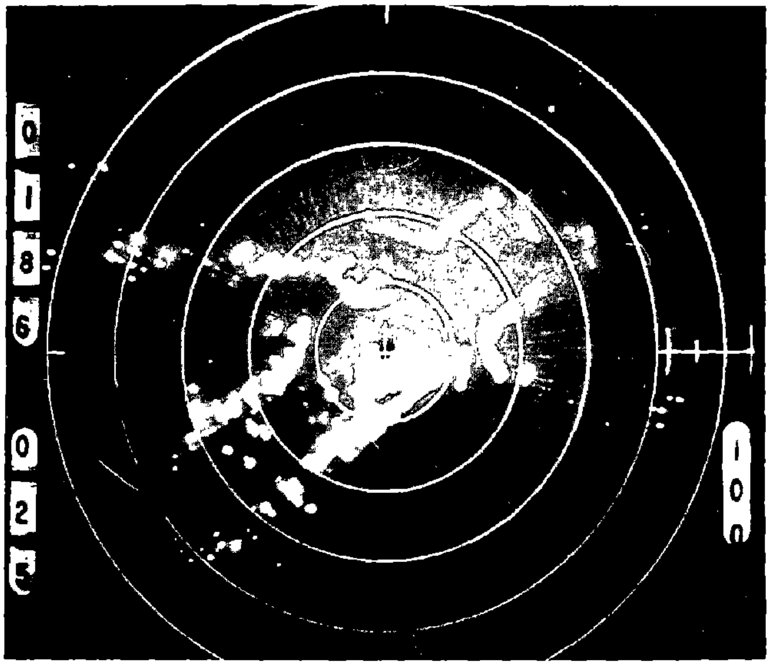

FIGURE 17. - Photograph of sonar caliper record made in mine opening in Clark bed in monitor hole 25, depth 186 feet. Concentric circles are spaced at 20-foot intervals.

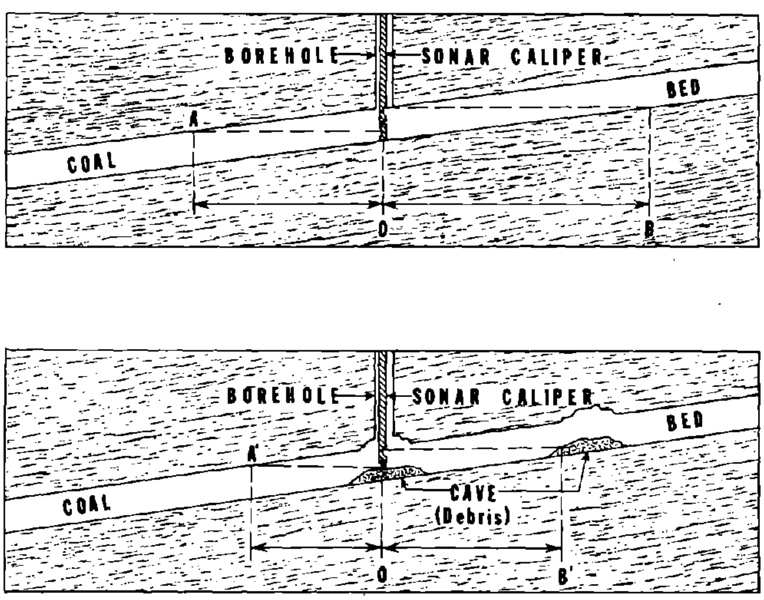

FIGURE 18. - Diagram showing line-of-site distances measurable by sonar caliper lowered through a borehole into a mined coalbed, with limitations imposed by the angle of dip and the location of debris accumulations on the mine floor.

The reflected sound pulses are amplified and transmitted to the surface through an electrically conducting wireline. The signals are displayed on an oscillograph tube calibrated for measuring the distance from the tool to the cavity walls. A permanent record of each cross section of the cavity was made by a camera attached to the oscillograph that takes time exposures through 1 revolution of the sonar tool. Figure 17 shows the record made in monitor hole 25 at a depth of 186 feet, which is 1 foot below the roof of the Clark bed as recorded by the driller. The configuration is consistent with the mine geometry shown in figure 10.

The objective of the sonar surveys was to gain information on the extent of open space in the mine voids between boreholes. The sonar tool measured only line-of-site distances in a horizontal plane; it cannot "see" around corners or through debris (15, p. 493). In sonar surveying of mine workings, geometric constraints on the distances that can be measured are imposed not only by the position of pillars, but also by the angle of dip, the uniformity of dip, the thickness of the mined bed, and the presence and location of debris as illustrated in figure 18. Assuming a uniform dip of 4° and the absence of debris accumulations, the maximum measurable distance parallel to the direction of dip in an 8-foot bed is 114 feet; in a 6-foot bed, 86 feet. Greater distances could be measured along mine openings that are not parallel to the dip, and lesser distances would be recorded where debris obstructed the line of sight.

At the time the sonar surveys were made, the vertical positions of the roof and floor of the mine voids were measured and compared with their positions as recorded by the drillers. The sonar positions of the roof accorded within half a foot with those reported in the drillers' logs. The positions of the floor as measured by sonar, however, were higher by an average of 2 feet than those in the drillers' logs. Mounds of cuttings may have accumulated below the boreholes similar to those shown in figure 19. If

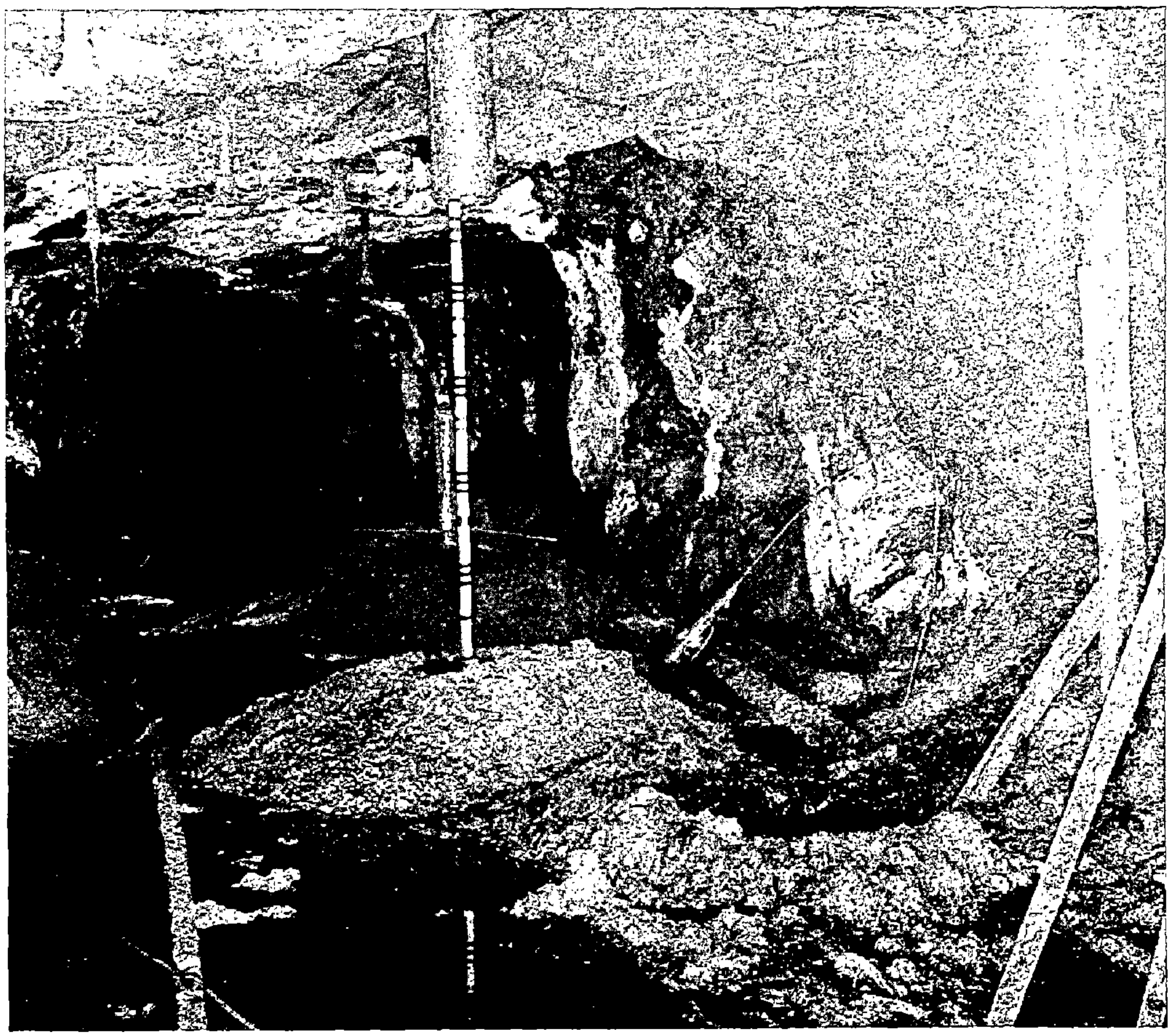

FIGURE 19. - Underground view in an accessible mine void, showing mounds of cuttings on the mine floor below boreholes.

so, sonar measurements in a down dip direction would be curtailed as shown in the diagram (fig. 18).

Sonar measurements made in the lower (Clark) bed recorded open space for distances as great as 100 feet, with the average about 70 feet. Of the four holes in which no open space was recorded, numbers 1, 3, and 14 were located in coal pillars in the Clark bed (as shown on the sections, figures 14 and 16). In the fourth, hole 9, the drillers' log reported 5 feet of broken rock and 1 foot of open space, which was consistent with its location near the sourthern boundary of the project (fig. 12), where a few pillars had been selectively removed and some backfill had been placed by the mining company. Thus, the sonar surveys in the Clark bed corroborated the conditions reported by the drillers and revealed no signs of caving within the distances that could be measured by the sonar method.

The sonar surveys were performed during the winter when the mine pool level was low, at an altitude of about 615. At this time, nearly half the mine workings in the upper (New County) bed were above water in the project area (fig. 12). The sonar surveys made with the air tool did not provide useful results. The sonar readings in the inundated part of the New County bed indicated a maximum open space of 85 feet with the average about 42 feet. As in the case of the Clark bed, holes in which no open space was recorded were those that terminated in coal pillars or broken rock. These results were interpreted as providing no positive evidence of caving in the New County bed within the distances that were measurable by the sonar method.

Changes in Underground Water Conditions

In Scranton, the naturally high water table in the Lackawanna River Valley was lowered progressively as anthracite mining was carried down to the lower coalbeds. During years of active mining, the entire city was undermined by contiguous mining operations separated only by barrier pillars. The usual cone of depression created by pumping to drain a single underground mine became a trough of depression extending longitudinally through the synclinorium of the northern anthracite field. Barrier pillars are parts of the coalbed that were left unmined along property lines, or between mines, or between parts of mines. Their principal functions in the anthracite region were to control ventilation and to act as dams to prevent water that had accumulated in a mine from overflowing into adjacent mines (4).

With the decline of the anthracite industry, the cessation of pumping water from abandoned mines greatly increased the pumping requirements in the remaining mines. Many barrier pillars had become ineffective as dams, having been mined through, breached at various altitudes, or affected by subsidence cracks. During the period 1944 to 1951, 23.2 tons of water were pumped to the surface for each ton of anthracite produced in the Northern Field (3, p. 36). By 1960 the ratio by weight of water to coal being produced in the Scranton area was estimated at 75:1 (21, p. 14). As the water levels rose above the highest breaches, individual mine water pools coalesced. Since all pumping from the mines has ceased, the mine water in the Scranton area is now contained in two enormous mine water pools. The source of the water used in the Green Ridge hydraulic backfilling operations is the mine water pool that extends from Ash Street in Scranton to Throop Borough north of Scranton, a distance of about 3 miles.

The height to which the mine water can rise in the Scranton area is controlled by the geologic structure of the Northern anthracite basin and its relation to the present topography. The long, narrow bedrock trough coincides fairly closely with the overlying Lackawanna River Valley in the north and the Susquehanna River Valley to the south. The bedrock synclinorium is subdivided into two basins by a structural saddle south of Scranton. The lowermost coalbed lies at an altitude of 20 feet below sea level in the deepest part of the basin to the north, 512 feet above sea level at the structural saddle, and 1500 feet below sea level in the deepest part of the basin south of the saddle (2). The rising mine water north of the saddle has built up to an altitude higher than the bedrock saddle, and water overflows onto the ground surface south of the saddle through a borehole and backfilled strip pits both adjacent to the Lackawanna River.

The Lackawanna River has a gradient of about 14 feet per mile in this area. The surface of the underlying mine water pool has a very low gradient, a little over 4 feet per mile. According to Jerrald R. Hollowell, who conducted ground-water investigations of Lackawanna County for the U.S. Geological Survey, the two bodies of water merge in the vicinity of the structural saddle. Therefore, in longitudinal section, there is a hiatus between the mine water pool below and the shallow ground water of the Lackawanna River alluvium. In this unusual hydrologic situation, the river and the valley fill are losing water to the mine water pool. The surface water and shallow ground water seep downward through cracks in the rock strata that overlie the mined coalbeds. As much as a third of the total runoff in the Lackawanna River drainage area joins the water in the mine pools (5).

In the immediate area of the Green Ridge demonstration project, the natural ground water level before ming started would have been a little higher than the elevation of the Lackawanna River 3/4 mile away, which is close to 700 feet. Lowering of the water table to permit mining operations reached altitudes close to 300 feet in the 1920's when the Dunmore coalbeds were being mined. Water did not accumulate in the two mines within the project area until 1960. During the interval, they drained into an adjoining mine where the pool level was maintained at a lower altitude (250 feet) by pumping water up through a shaft to discharge on the surface (2, 21). By 1960, all pumping from the mines ceased. Within the project area, the mine pool probably reached a stable condition by May 1962.

The position of the fluctuating mine pool was pertinent to the project operations because of the relative proportions of area in the upper coalbed that were above and below pool level during the time of the demonstration project. Because the pool level was low during the sonar surveys, the usefulness of sonar to detect open space in the mine workings in the upper bed was restricted to about half of the project area where the voids were below water level. At the time of injection into the upper bed, however, at least two-thirds of the area was inundated, which provided optimum conditions for backfilling the major portion of the upper bed yet permitted opportunity to test the method in part of the area that was above pool level.

Project Planning and Specifications

Early project planning included an inventory of property values on the surface to be protected and consultation of mine maps to judge the suitability of subsurface conditions for backfilling. Exploratory boreholes were drilled to confirm subsurface conditions. Availability of adequate supplies of power, fill material, and water were confirmed. The water source was sampled and analyzed to determine requirements for corrosion-resistant equipment.

Operational plans for injection projects are based on the number of coalbeds to be backfilled and the volume of void space in each bed as determined by the thickness of coal that has been mined and the percent extraction. If more than one mined coalbed is to be backfilled, the lowest bed should be filled first. Moreover, the quantities planned for injection into an upper bed should be proportional to and directly above the void space to be filled in a lower bed so that the weight added by placing material in an upper bed would be supported by the backfilled portion of the lower bed. Original plans for the Green Ridge project were based on a total quantity of 300,000 cubic yards of fill material. The lower (Clark) bed averages 8 feet in thickness, and extraction was 69 percent; the upper (New County) bed averages 6 feet thick, with 64 percent extracted. Thus,

$$(8 \times .69 \times \pi r^2) + (6 \times .64 \times \pi r^2) = 300{,}000 \times 27 \text{ cubic feet.}$$

Solving for r, the area of ground that could be stabilized with 300,000 cubic yards of fill was 20 acres, and the radius was 525 feet. The proportionate allocations of fill material were 177,000 cubic yards to the lower bed and 123,000 cubic yards to the upper bed.

Planning for the number and location of injection boreholes was based on study of the mine maps supplemented by sonar surveys. Factors considered included dip of the coalbed, arrangement of pillars, location of manmade barriers or caved areas in the mine, and the position and fluctuation of the water level.

Design of the mixing plant and distribution system involves determination of optimum water requirements, slurry velocity, ratio of solids to water in the slurry, and pipeline diameter. One objective of the demonstration project at Green Ridge was to gain experience on which these determinations could be based. As a result of the demonstration project, the following range of conditions were identified for injecting crushed mine refuse (specific gravity about 2.2) at rates of 200 to 400 tons per hour through a borehole to achieve approximately 100 percent filling of surrounding mine voids to distances of 500 feet or more from the injection hole:

Water volume in the range of 4,000 to 7,500 gpm.

Slurry velocity in the distribution pipeline and injection borehole not less than about 15 fps.

Water-to-solids ratios of 5.5:1 to 21:1 by volume and 2.5:1 to 9:1 by weight.

Slurry distribution pipe of inside diameter between 10 and 14 inches.

Casing of injection borehole of same size as distribution pipeline.

A slurry pump capable of delivering slurry at 50 psig at the top of the farthest distant injection borehole.

Further refinements of these requirements can be made as more experience is gained. The relationships between injection rates and slurry concentrations are summarized in table 1.

In planning the number of shifts per day to be worked, the amount of disturbance that the operation will cause in the neighborhood must be balanced against the loss of efficiency that attends interruptions to injection. Both at the start of injection after a period of shutdown and at the end of slurry injection, the distribution line needs to be purged with clear water. At when resistance to injection is building up, interruptions to pumping are generally followed by increased resistance, and extended periods of injecting clear water may be necessary to open up an underground channel. The brief demonstration at Rock Springs, Wyo., was scheduled for three shifts per day. The Green Ridge demonstration project started on a one-shift (8-hour) day and changed to two shifts when the scope of the project was enlarged.

The basis of payment to the contractor was the quantity of solids injected as measured by a belt scale placed on the conveyor that fed the solids to the mixing tank.

Specifications for drilling of boreholes and installation of pipeline included provision for restoring the streets to normal condition by removing pipe and filling and capping all boreholes and wells.

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

	Water pumped				
	4,000 gpm (16.69 tons/min)	5,000 gpm (20.86 tons/min)	6,000 gpm (25.04 tons/min)	7,000 gpm (29.21 tons/min)	8,000 gpm (33.38 tons/min)
SOLIDS INJECTED: 200 TONS/HR, 3.33 TONS/MIN, 364 GPM					
Slurry water-to-solids ratios:					
Weight..............	5.0	6.3	7.5	8.8	10.0
Volume..............	11.0	13.7	16.5	19.2	22.0
SOLIDS INJECTED: 300 TONS/HR, 5.00 TONS/MIN, 546 GPM					
Slurry water-to-solids ratios:					
Weight..............	3.3	4.2	5.0	5.8	6.7
Volume..............	7.3	9.2	11.0	12.8	14.7
SOLIDS INJECTED: 400 TONS/HR, 6.66 TONS/MIN, 728 GPM					
Slurry water-to-solids ratios:					
Weight..............	2.5	3.1	3.8	4.4	5.0
Volume..............	5.5	6.9	8.2	9.6	11.0

Crushed breaker refuse

Bulk density 74 pcf (loose) 46.1 pct voids; 53.9 pct solids
2.2 (specific gravity) × 62.4 lb = 137.3 pcf (solid)
137 ÷ 7.48 gal/ft^3 = 18.3 lb/gal (solid)

Water

4,000 gpm × 8.345 lb/gal = 33,380 lb/min = 16.69 tons/min
5,000 gpm × 8.345 lb/gal = 41,725 lb/min = 20.86 tons/min
6,000 gpm × 8.345 lb/gal = 50,070 lb/min = 25.04 tons/min
7,000 gpm × 8.345 lb/gal = 58,415 lb/min = 29.21 tons/min
8,000 gpm × 8.345 lb/gal = 66,760 lb/min = 33.38 tons/min

Injection rate of solids

$$\frac{\text{tons/min} \times 2000}{18.3 \text{ lb/gal}} = \text{gal/min}$$

200 tons/hr = 3.33 tons/min = 364 gal/min
300 tons/hr = 5.00 tons/min = 546 gal/min
400 tons/hr = 6.66 tons/min = 728 gal/min

Injection rate of slurry by volume (2 examples)

Water	+ Solids	= Slurry
5,000 gpm	3.33 tons/min	5,364 gpm
7,000 gpm	6.66 tons/min	7,728 gpm

System Components

Water

The source of water, which is needed in large quantities to accomplish a hydraulic backfilling operation, was the local mine water at the project site. 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. At the Green Ridge project area, the mine pool is a vast body of subsurface water about 24 square miles in area with an estimated volume of 30 billion gallons. The effect on average pool altitude of either water withdrawal or slurry injection is considered to be insignificant.

A disadvantage of using mine water is the possible corrosion of pumping equipment due to the dissolved mineral content. In the anthracite region, acidity and high iron content are characteristic of much of the mine water. Early in the planning stage of the Green Ridge project, the chemical quality of local water sources was investigated. Table 2 presents chemical analyses of four mine water samples.

TABLE 2. - Chemical analyses of mine water in and near injection project area

	Pine Brook mine shaft[1]	Dickson Ave. borehole 1A[2]	Delaware St. borehole 2A[2]	Delaware St. borehole 3A[2]
pH......................	3.46	6.97	6.68	7.12
Concentration, mg/l:				
Sodium................	460	46	92	23
Calcium...............	129	74	47	67
Magnesium.............	120	40	24	33
Manganese.............	12-120	2-20	2-20	2-20
Aluminum..............	4-35	1-10	1-10	1-10
Chloride..............	690	70	160	70
Sulfate...............	745	133	62	63
Bicarbonate...........	0	221	110	270
Total solids..........	2,299	598	509	540
Iron, ferrous.........	0	.25	6.2	.8
Iron, ferric..........	14	.1	.6	.1
Total acidity, Co_2....	54	20	20	25
Total acidity, H_2SO_4..	120	45	45	56

[1]Approximately 1 mile southwest of project area.
[2]Locations of boreholes shown on fig. 10, average depth 163 feet.

The results of the chemical analyses indicated that the local mine water was less corrosive than much of the mine water that had been pumped to the surface from active mines in the region over a decade ago. Because mine water was to be pumped over a period of months, however, with the possibility of drawing in water of varying composition, stainless steel pumps were used in the water wells. The pH of the water was checked daily during the injection operation, and provision was made to raise the submersible pumps should the pH drop below 4.5. During the injection period, the pH ranged between 5.5 and 5.0.

The mine water source was developed by drilling two water supply wells at the plant site. Their location combined convenience of plant site layout with maximum feasible distance from the closest injection hole to guard against suspended fine material from the injected slurry reaching the water pumps.

In order to locate two water wells, three probe holes were drilled with a 7-7/8-inch bit. Probe hole No. 1 encountered a solid coal pillar and was abandoned. At the horizon of the upper (New County) coalbed, probe hole No. 2 encountered 12 feet of broken rock; probe hole No. 3 found a 2-foot opening and 5 feet of broken rock. At the lower (Clark) bed horizon, 9 feet of broken rock was reported in both holes. The water level in the mine stood at altitude 616 (about 95 feet below the surface) at the time the probe holes were drilled.

The two probe holes that intercepted mine openings were enlarged and cased. Casing of 24-inches diameter (wall thickness 0.281 inch) extended to a depth of approximately 70 feet; 20-inch-diameter casing (wall thickness 0.250 inch) extended to a depth of 160 feet. The casing at mine level in the New County coalbed was slotted by cutting torch to admit water; the slots cut were 1-1/2 by 3 inches.

Two submersible water pumps constructed of stainless steel were installed, one in each of the two water wells, 148 feet below the surface (about 50 feet below average mine water pool level). The pumps were rated at 4,200 gpm each, at a dynamic head of 310 feet. The pumps were driven at 1,800 rpm by 400-hp motors using 3-phase, 60-cycle, 2,300-volt power. Failure of one pump after pumping approximately 875,830,000 gallons required replacement of the motor.

The water supply pipeline from the water wells to the mixing tank, a distance of about 150 feet, was welded pipe having 12-3/4-inch outside diameter and wall thickness of 1/4 inch (3/8 inch in one segment).

The total volume of water used and the rate of flow were continuously measured and recorded. The total quantity of water that was used, 2,263,219,000 gallons, far exceeded the quantity used in actual slurry injection. Clear water was flushed through the system at the start and end of each day and at times when the injection of slurry encountered resistance in the mine openings.

Solids

Source of Material

The solid component of the slurry used in the backfilling project was crushed waste rock that had been left from the mining and processing of anthracite coal. Throughout the anthracite region, the landscape is marked by large piles of mine waste material locally termed "refuse banks" (24). The main constituents of the banks are rock fragments, "bone," and small pieces of coal. The rock fragments are chiefly sandstone, siltstone, claystone, and shale from the strata that occur above, within, and below the coalbeds. "Bone" is the term used for hard carbonaceous shale containing approximately 40 to 60 percent noncombustible materials. The coal content of a refuse bank is determined by the efficiency of coal separation methods at the time the refuse accumulated or at the last time the bank may have been reworked to recover fine coal.

Much of the waste rock was derived from the processing of coal in preparation plants locally termed "breakers." Some waste rock that was produced in the mine or in excavating access tunnels was brought directly from the mines to the refuse banks.

For the Green Ridge backfilling project, the source of backfill material was the Eureka refuse bank, located at the project site along the northwest edge as shown in figures 7 and 8. The close proximity of the Eureka bank eliminated both the cost and the public nuisance associated with trucking large quantities of rock through city streets. The material in the bank was made available by the owners without cost to the Government.

The Eureka refuse bank has been used as a source of material for backfilling mine voids under other areas in Scranton. Between 1964 and 1971, about 900,000 cubic yards were removed from the bank for this purpose. At

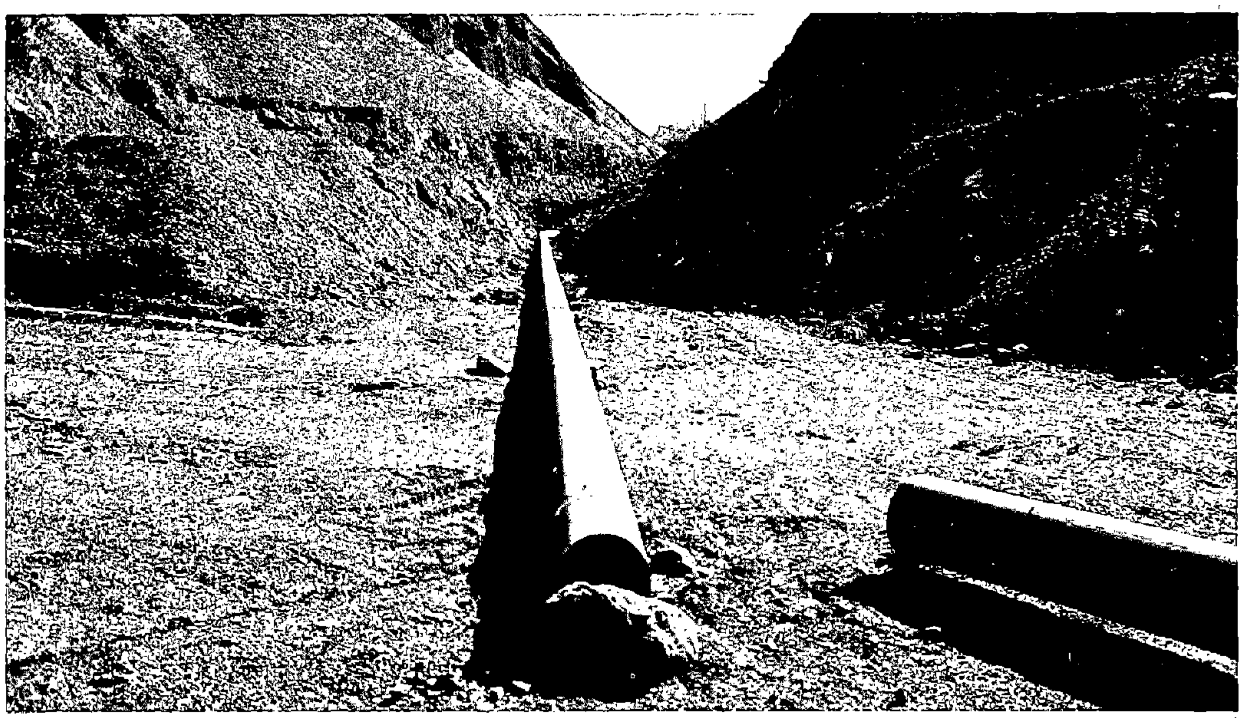

FIGURE 20. - Excavation through Eureka bank for slurry distribution pipeline.

the start of the Green Ridge demonstration project, the volume of the refuse bank was estimated at 455,000 cubic yards, and the height of the bank was 123 feet. The uncrushed compacted material on the bank had an average bulk density of 95 pcf. A factor of 20 percent volume increase was used in estimating the quantity of crushed solids to be produced from the original bank material.

Composition

The Eureka bank was composed mainly of breaker refuse with some mine rock. The bank had been reworked twice for coal recovery; estimates of the remaining coal content ranged from 3 to 10 percent.

Variation in the size of rock fragments on the bank can be seen in figure 20. Particle sizes ranged from boulders of about 1 cubic foot down to silt-size grains. The grain-size gradation of unprocessed refuse from the Eureka bank is shown graphically in figure 21 by the curve on the left;

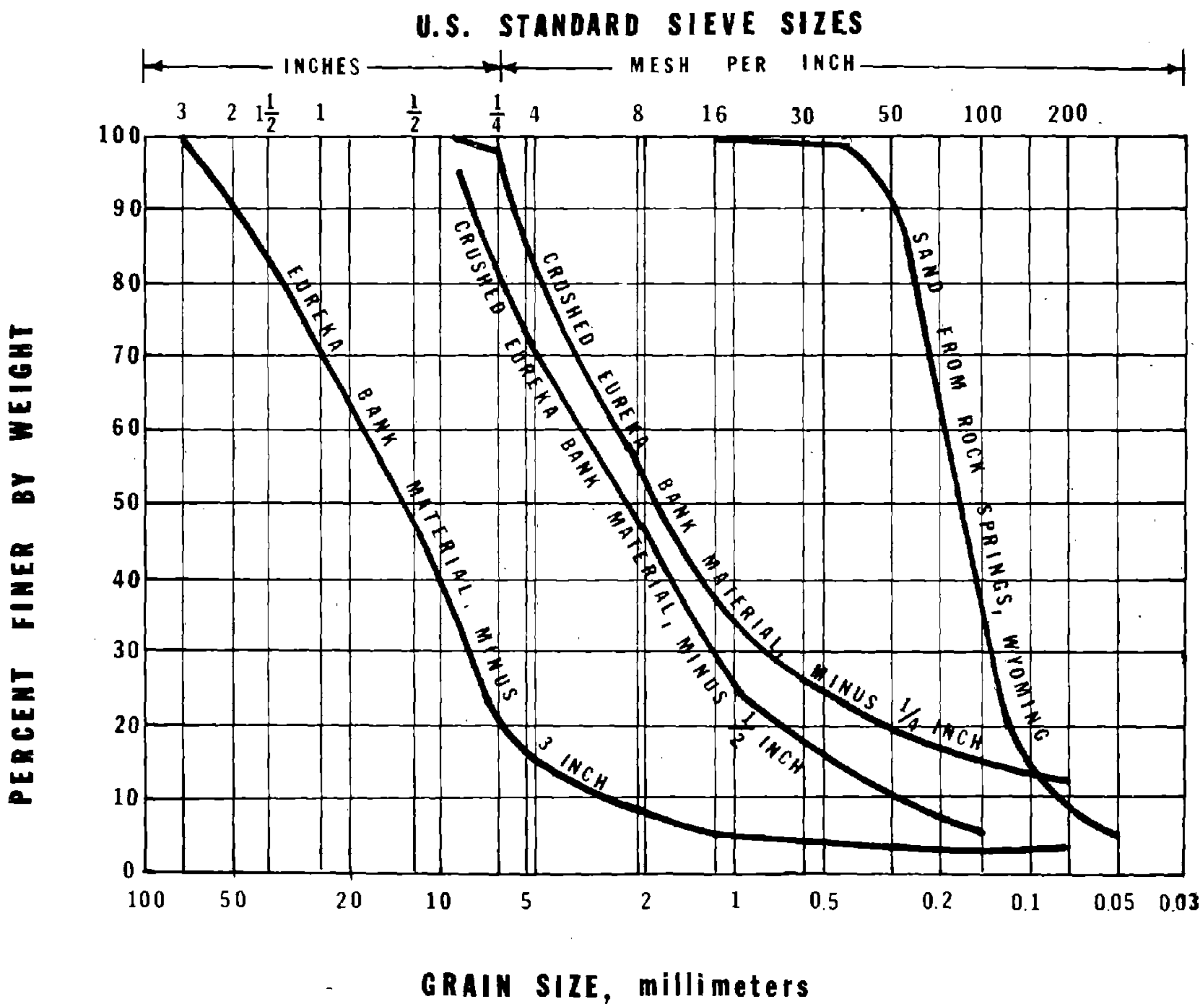

FIGURE 21. - Analyses of Eureka bank material showing cumulative distribution of particle sizes.

fragments larger than 3 inches in diameter had been removed from these samples.

Much of the material in the refuse bank had become well compacted. Excavation of a trench through the bank for pipeline installation proved difficult, and cut sections stood at angles near vertical or even overhanging (fig. 20).

The mineral composition of Eureka bank material was determined by X-ray diffraction analysis to be predominately quartz with minor amounts of a non-crystalline material (probably carbonaceous matter) and illite; some kaolinite and chlorite were identified. An optical emission spectrographic analysis showed silicon as the major element, with minor amounts of iron, aluminum and calcium. The average specific gravity of samples from the Eureka bank was 2.2.

Engineering Test Data

Laboratory tests for moisture content, density, and compressibility were made on samples from the Eureka refuse bank, crushed to minus 1/4-inch size. The average moisture content was 5.5 percent. Bulk density of samples of dry material ranged from 76 to 80 pcf.[5] The bulk density of samples hydraulically placed in a water-filled receptacle ranged from 92 to 97 pcf. The bulk density of samples similarly placed and allowed to drain ranged from 96 to 106 pcf.

Laboratory tests of compressibility indicated that crushed mine refuse is subject to a greater degree of compressibility than fine sand. The loose packing characteristic of crushed mine rock permits greater volume reduction under loading than sand in which the particles tend to be rounded and better sorted. At the Green Ridge project site, the average weight of overburden is about 150 pcf. The maximum depths of the two coalbeds that were backfilled are approximately 200 feet and 150 feet. Therefore, the loading imposed by overburden at the levels of the coalbeds is estimated to be 208 psi at the lower (Clark) coalbed and 156 psi at the upper (New County) coalbed. Compressibility tests were made on samples of Eureka refuse crushed to minus 1/4-inch size, poured into a length of pipe, and subjected to loading at increments of 1,000 pounds. Compressibility under loads of approximately 208 psi and 156 psi was found to be 4.5 percent and 3.7 percent, respectively.

Assuming that the mine voids were filled to the roof, the thickness of emplaced fill in the 8-foot-thick lower bed would be reduced by 4.3 inches if the overburden were to rest directly on the fill. Similarly, fill in the upper bed, 6 feet thick, would be compressed by 2.6 inches. Thus the combined contribution to possible future subsidence, if the overburden were to be supported entirely by emplaced fill deposits, could be 7 inches at most. This assumes complete failure of pillars at both mine levels, and does not allow

[5]Tests made during the progress of the injection operation on the crushed refuse actually used indicated an average onsite density of 74 pcf.

for a probable volume increase in the overburden strata if they were substantially broken in the course of subsidence working its way upward toward the surface. It is unlikely, however, that the total weight of overburden will ever rest on the fill material. Pillars occupy about one-third of the voids, and the presence of fill material in the mine chambers inhibits coal breakage and provides lateral support for pillars regardless of whether the fill reaches the roof.

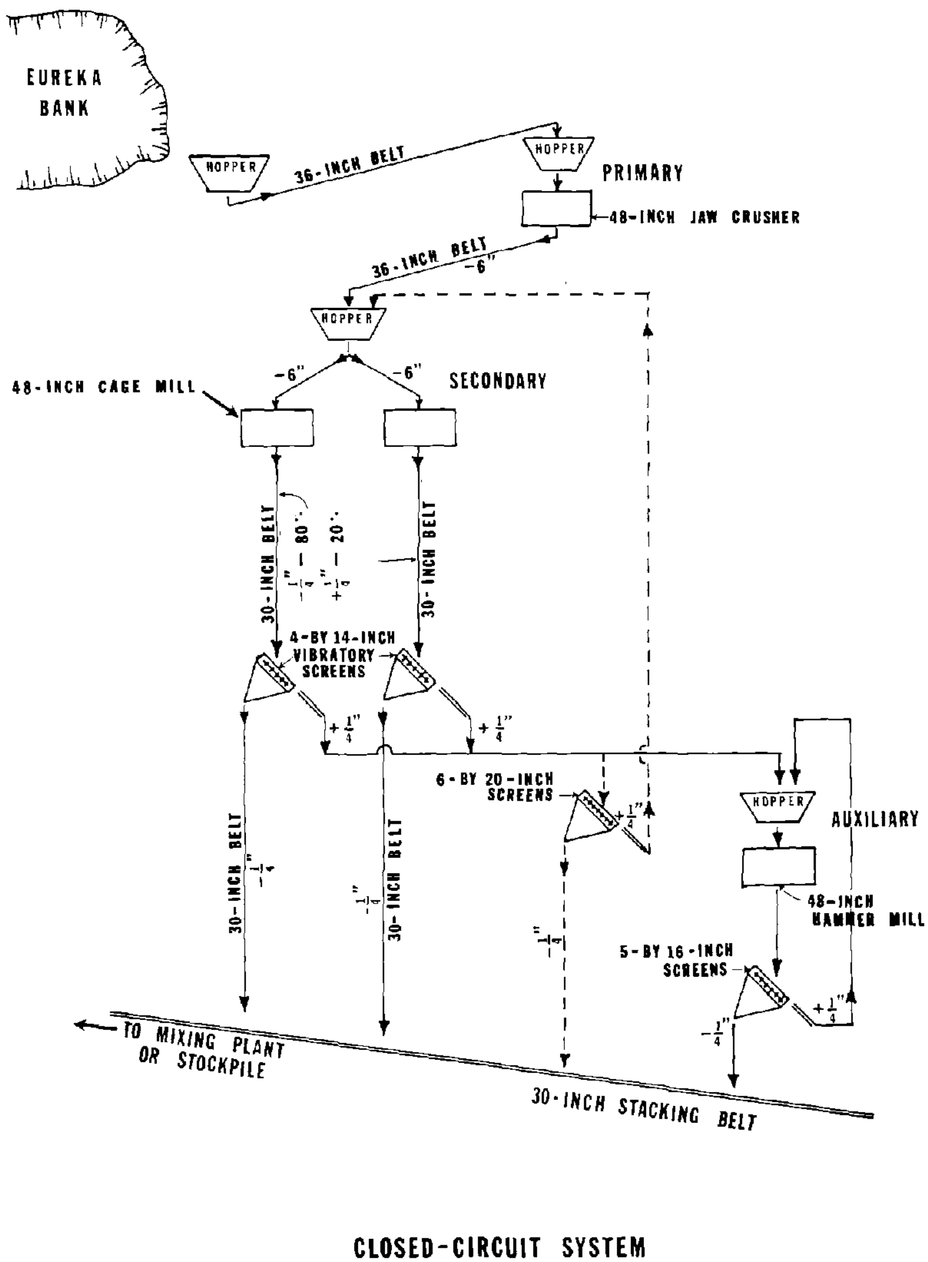

FIGURE 22. - Flow chart of crushing operation, Green Ridge demonstration project.

Crushing

The original contract for crushing the Eureka bank material required crushing of up to 300,000 tons of breaker refuse to minus 1/4-inch size and delivery of crushed breaker refuse material to the slurry mixing plant. The crushing and sizing equipment were to be operated with a minimum polluting effect on the environment. The required rate of production was an amount sufficient to permit uninterrupted operation of the injection contract according to a schedule of 2,000 tons per 8-hour day, 5 days per week, until a total of 300,000 tons was injected. The price for crushing included picking up the raw material from the refuse bank and conveying it to the crusher, as well as conveying the crushed material from the crusher or stockpile to the hopper of the slurry mixing plant.

Before the injection phase had started, the daily crushing output of minus 1/4-inch material was expected to be somewhat less than the daily injection capacity. Therefore, a stockpile of crushed material was produced before the start of injection. Available storage space at the project site limited the size of the stockpile to about 60,000 tons. During part of the injection period, the crushing plant was operated on weekends as well as during the week. About 2 months after the start of injection, an auxiliary crusher was added to increase the capacity of the crushing plant.

The crushing plant equipment included front-end loaders, primary and secondary crushers, conveyor belts, and heated vibrating screens. Figure 22 is a flow chart depicting the operations of the original and auxiliary crushing plants.

The original specification of minus 1/4-inch size of crushed refuse for use in the demonstration project was based primarily on the "fluid loss" or permeability characteristics of the emplaced fill. Low permeability of the solids being deposited on the mine floor around the injection borehole was required to insure that incoming water would transport solids over the top of the fill rather than flow through the deposit.

The specifications provided for experimentation with a limited quantity of larger-size crushed material. After 4 months of successful injection of minus 1/4-inch solids, a trial period of injecting minus 1/2-inch solids demonstrated the feasibility of using the coarser material. The large percentage of fines in both minus 1/4-inch and minus 1/2-inch materials provided favorable permeability. The use of minus 1/2-inch crushed refuse reduced the time required and costs of crushing. The contracts were modified to change over to use of minus 1/2-inch material, and the total quantity to be injected was changed from 300,000 tons to 450,000 tons. The planned rates of both crushing and injection were increased from 2,000 tons per 8-hour day to 5,000 tons per 16-hour day.

Slurry Mixing Plant

The mixing plant is the heart of the pressure injection system. Here the water and the solids are combined to form a slurry that is maintained throughout the injection operation. From the mixing plant, the slurry is pumped through the pipeline leading to the injection boreholes. The mixing plant equipment for the Green Ridge project included--

A 20-yard hopper, conveyor belt, and weighing device;

A 12-inch-diameter pipeline from the water supply wells;

A mixing tank, capacity 2,000 gallons;

Two suction lines from the tank to the slurry pumps;

Two diesel-powered slurry pumpers;

and an instrument-control module.

The plant layout is shown diagrammatically in figure 23.

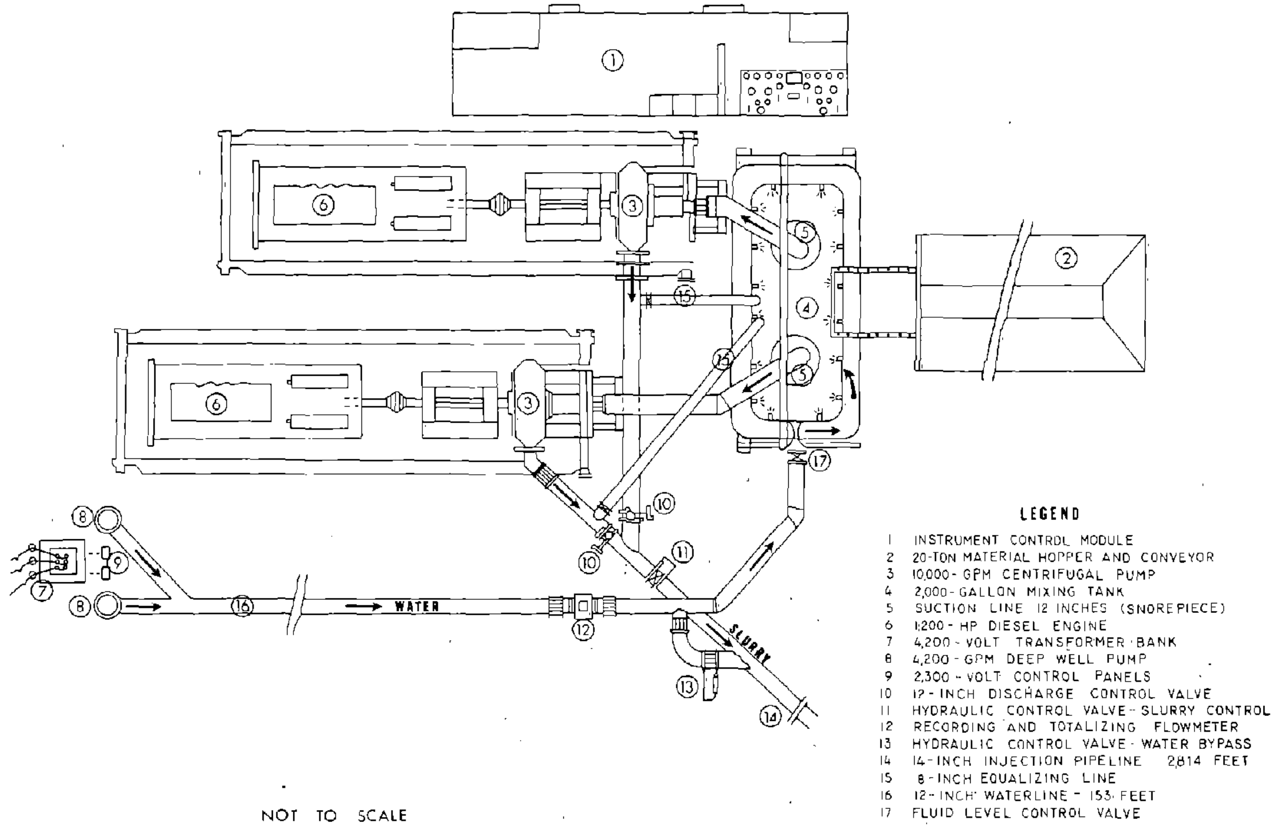

FIGURE 23. - Diagram showing slurry mixing-plant components.

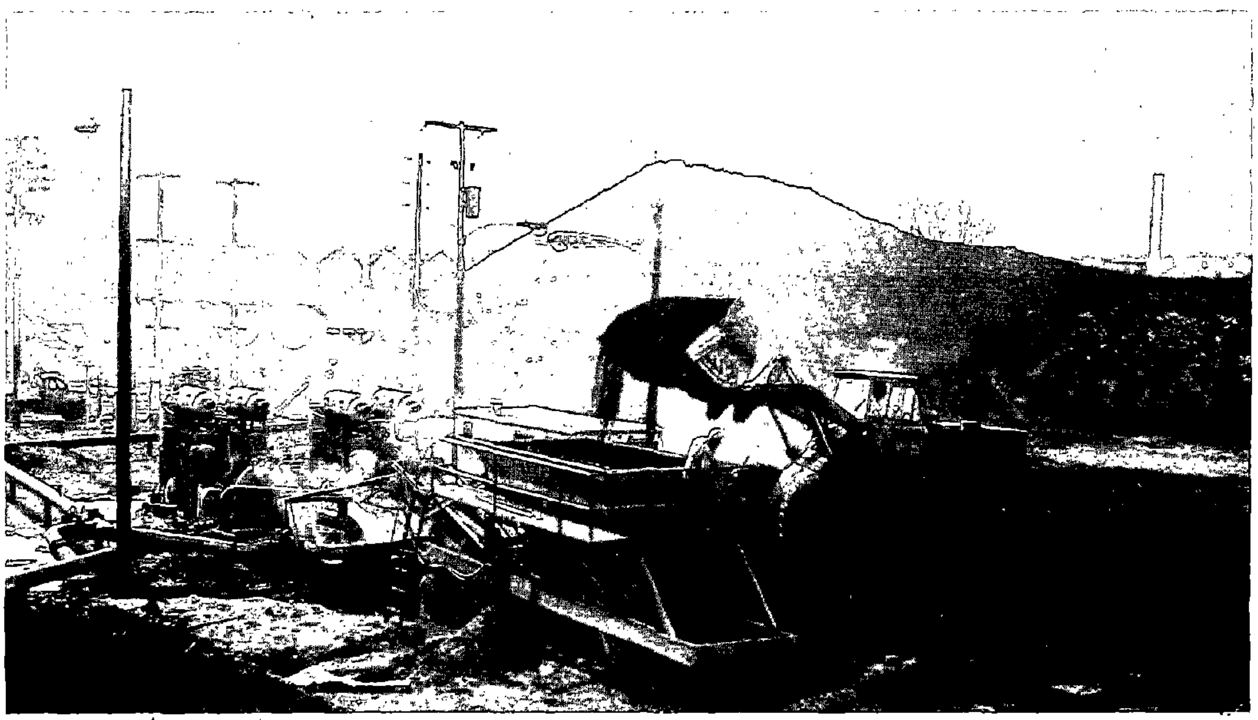

FIGURE 24. - Hopper at mixing plant receiving crushed refuse.

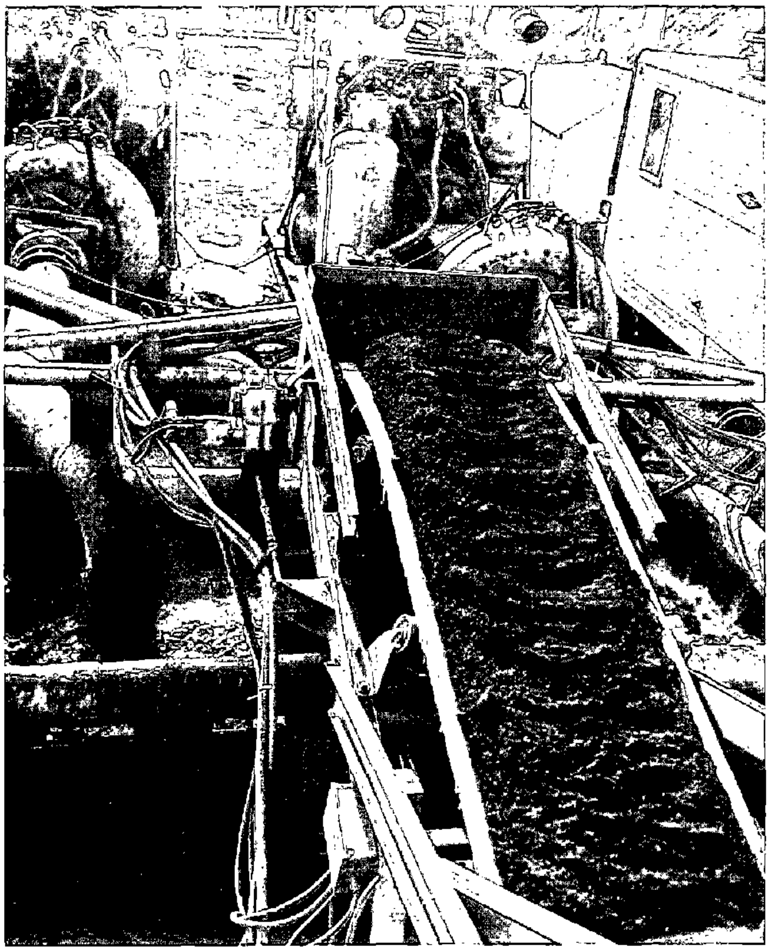

FIGURE 25. - Variable-speed belt conveyor, transporting solids to mixing tank. Weight meter attached below belt.

The hopper, rated at 800 tph (fig. 24), received solids from the stockpile of crushed material or direct from the crushing plant via conveyor belt. From the hopper, the solids were transported by a short but large-capacity conveyor belt (shown in operation in figure 25) into the mixing tank. Underneath this belt was a metering device that continually weighed and totalized the weight (in tons) of solids on the belt being delivered to the mixing tank. This meter provided the basis of payment for both the crushing and injection operations, and it was calibrated weekly.

The mixing tank, shown in figure 26, was a steel tub measuring 16 feet long, 8 feet wide, and 2.4 feet deep. Above its rim was mounted a manifold connecting with the incoming waterline. Water entered the tank through a series of rectangular nozzles extending from the manifold, producing sufficient turbulence to maintain the solids in suspension (fig. 27). During the course of the project, some leaks developed at the square corners of the manifold outlet connections. Two 12-inch suction lines with snorepieces, shown in figure 28, picked up the suspended material and conveyed it to the impellers in the slurry pumps. Soon after the injection operation started, a bypass was installed in the pipeline to permit clear water to be delivered directly to the injection line without passing through either one of the slurry pumps. The view in figure 29 shows the spatial relation of the operations of the conveyor belt, mixing tank, suction lines, and slurry pumps. The level of water in the mixing tank was controlled by a throttling valve on the water supply line and by the speed of the slurry pump. The rate of solids delivery was controlled by varying the speed of the conveyor belt.

The slurry was pumped into the distribution line by one of two identical pumper units, each completely integrated with the system for immediate switchover in case of malfunction. Each unit was used on alternate days and their total operating times were 860 and 836.5 hours. The slurry pumps, of

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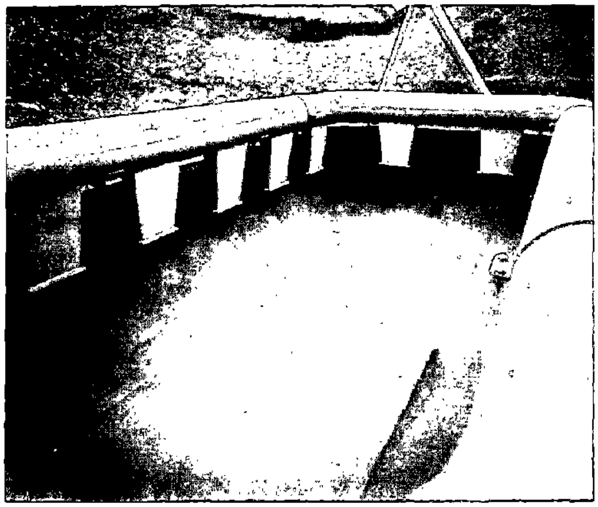

FIGURE 26. - Mixing tank and manifold.

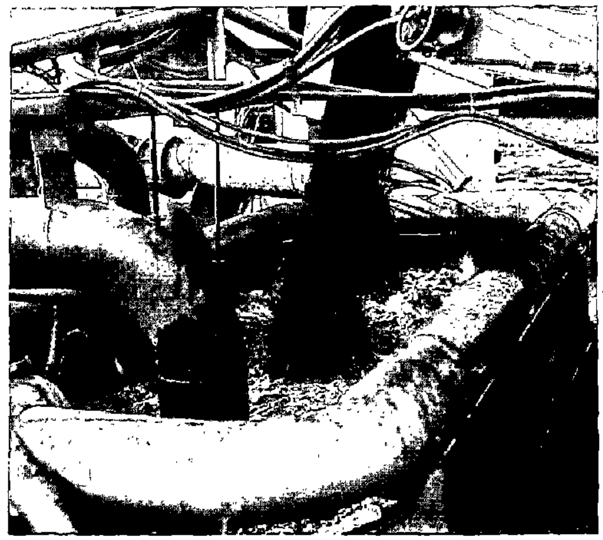

FIGURE 27. - Slurry mixing tank in operation.

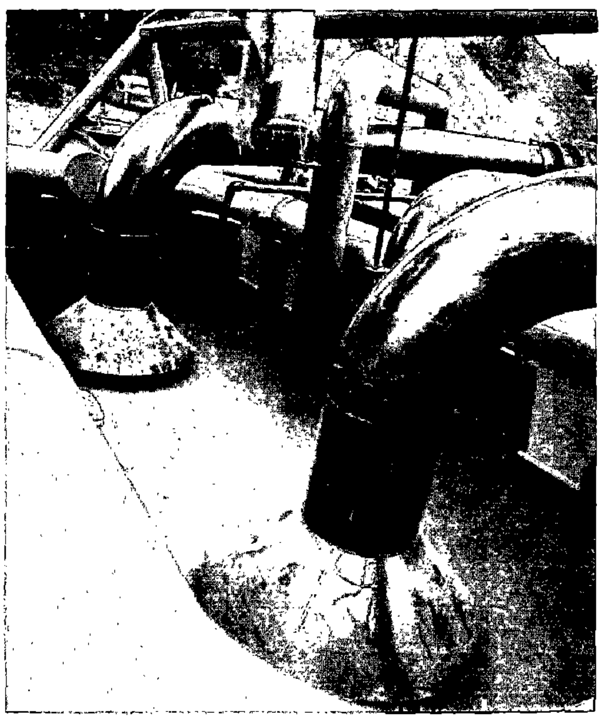

FIGURE 28. - Snorepieces to pick up slurry from mixing tank.

high-chrome cast iron, were diesel powered, centrifugal, dredge service pumps, each one capable of pumping slurry at a rate of 8,400 gpm at a head of 175 feet. Pump wear was minor. The impeller housing clearance had to be taken up twice during the project.

During periods when solids were being mixed with water to form a slurry, the average water flow rate ranged from 5,000 to 7,000 gpm; when clear water was being pumped through the bypass, the rate was close to 8,000 gpm. The

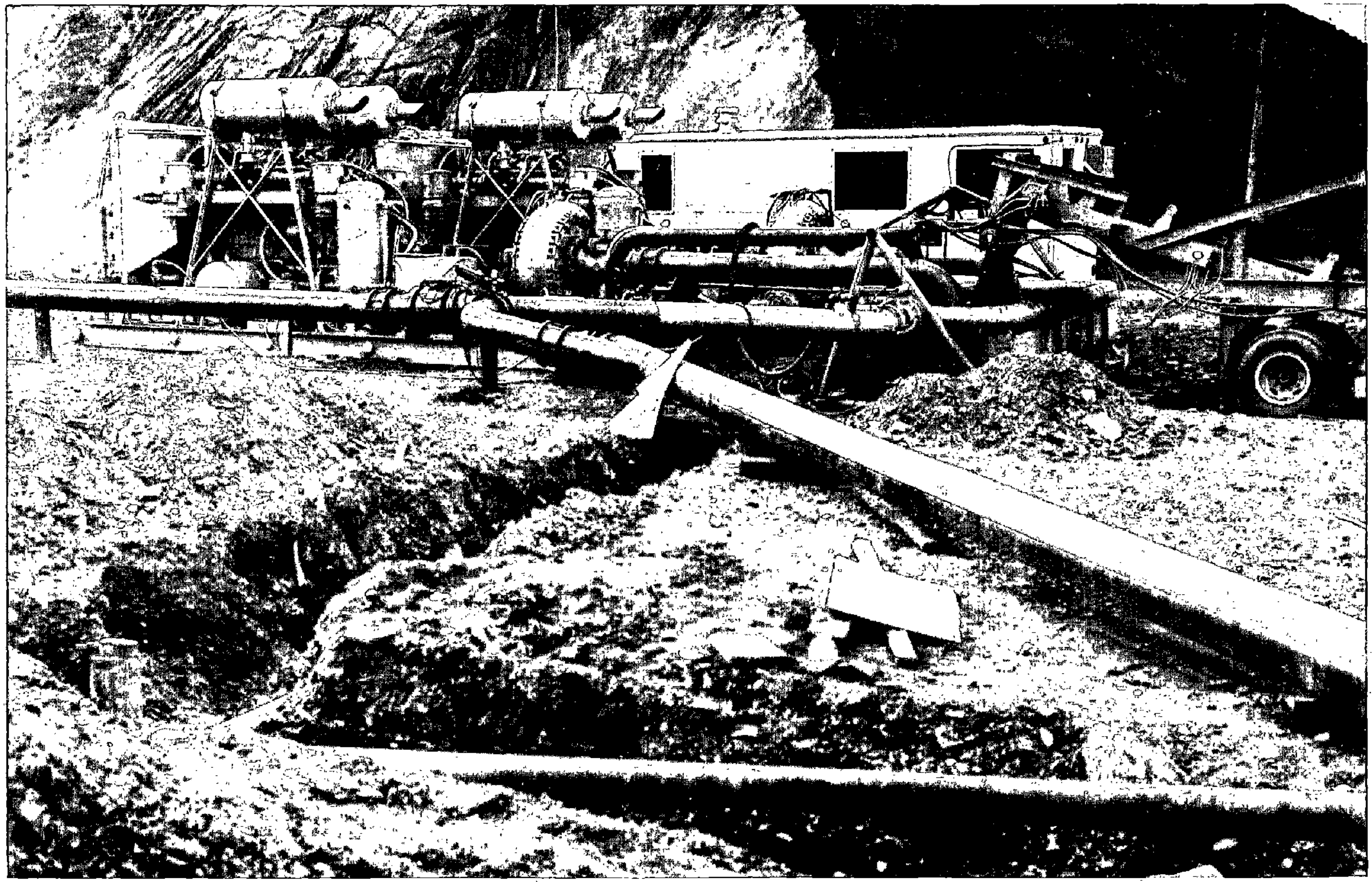

FIGURE 29. - View of mixing plant in operation. Water enters through pipe from left to mixing tank manifold; solids enter tank from conveyor at right; suction line transmits slurry to pumps in center; slurry enters distribution pipeline that passes under waterline to right foreground. Bypass in center provides direct connection from waterline to distribution pipeline. Drainage hole in left foreground. Trailer in background contains recording instruments and control panel.

diesel engines, with maximum rated capacity of 1,200 hp on an intermittent basis and 940 hp with constant use, were usually operated at about 750 hp. Power was increased during periods when back pressure was produced by resistance to injection of slurry as the underground flow channels became constricted.

A photograph of the pumpers (fig. 30) shows the slurry pump and the power assembly, which includes speed reducer, air starter, engine, fuel tank, and radiator, with exhaust mufflers placed above. When installed in the residential area of the demonstration project, the mufflers were modified to further reduce the noise level. The diesel engines also provided auxiliary power for control of the conveyor leading to the mixing tank and for operating the squeeze valves in the slurry distribution pipeline.

All operations involved in the mixing and injection processes were controlled from an instrument panel located in a trailer placed adjacent to the mixing tank (fig. 31). While in operation, the tank and surrounding equipment were under constant surveillance through a window in the trailer. In addition, continuous readings in the rate of waterflow, the rate at which solids were introduced, and the discharge pressure from the slurry pump were displayed on recorders adjacent to the instrument panel. Automated signals were installed to activate a warning light and to sound an alarm in the control module if the pressure at the slurry pump should reach

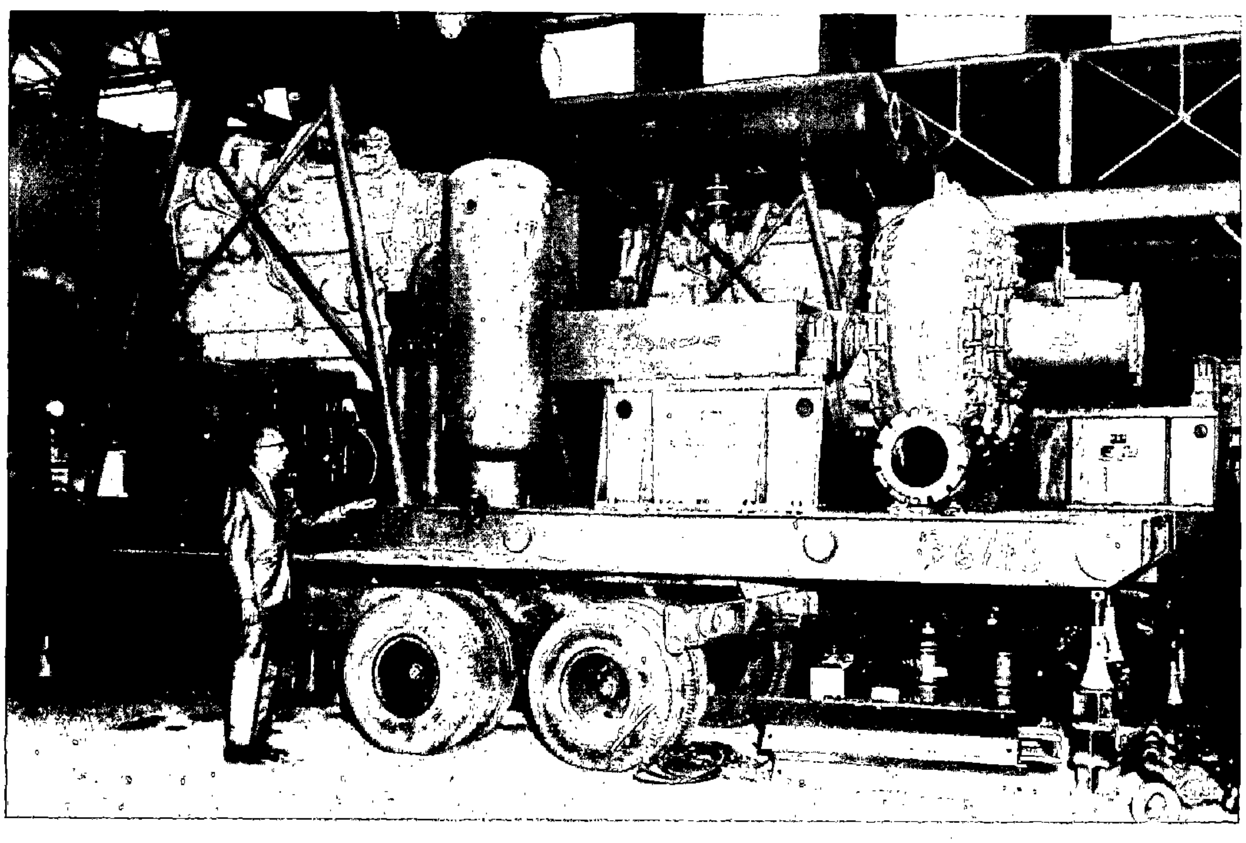

FIGURE 30. - Pumping units mounted for transport.

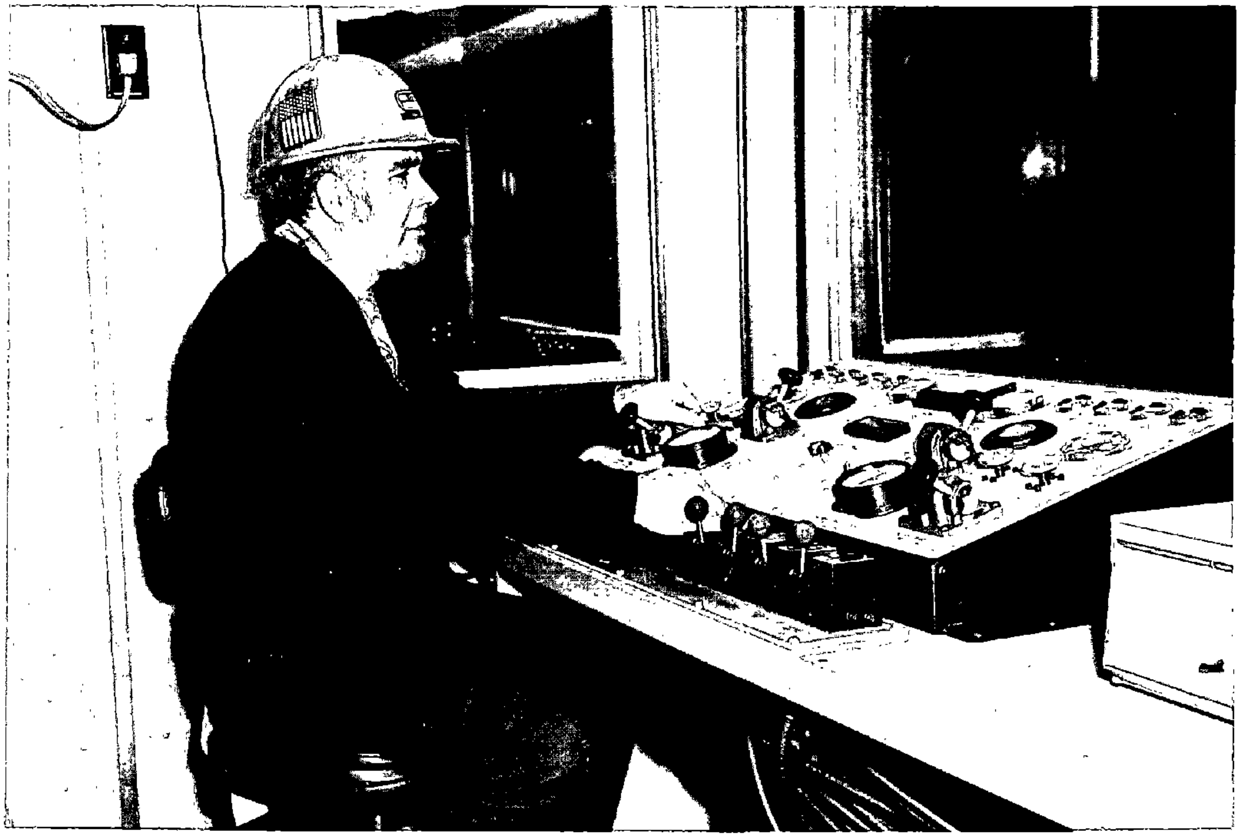

FIGURE 31. - Control panel for operation of mixing plant.

100 psig and if the slurry velocity should drop to 12 fps, indicating an abnormally low volume of flow.

A drainage borehole at the plant site provided for possible overflow from the mixing tank. The borehole was located at the bottom of a large sump enclosed by an embankment, and it terminated in the New County bed.

Slurry Distribution System

A pipeline system conveyed the slurry overland from the mixing plant to each injection borehole. Five injection boreholes provided the vertical connections between the pipeline and the mined-out space in the subsurface coalbeds. Originally one injection borehole was provided for the lower coalbed and four injection boreholes for the upper coalbed. When the project was enlarged, two of the upper bed injection boreholes were deepened to provide additional access to the lower bed.

Pipeline

The plan of the distribution pipeline layout is shown in figure 32. To reach Delaware Street from the mixing plant site, a trench was excavated through the Eureka refuse bank (fig. 20). The distribution line followed

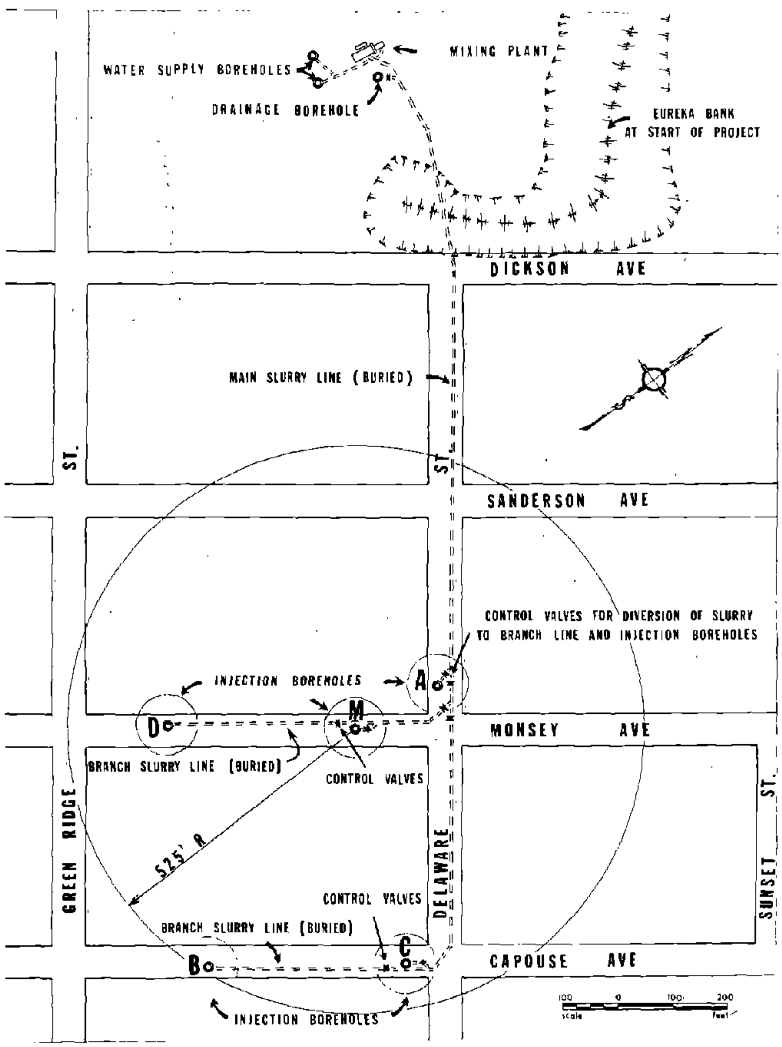

FIGURE 32. - Slurry distribution pipeline layout.

Delaware Street for three blocks on an uphill grade to Capouse Avenue. Injection borehole A, to the New County coalbed, is located on Delaware Street, northwest of its intersection with Monsey Avenue. To reach the other four injection boreholes, pipeline was laid along Monsey and Capouse Avenues. At the street intersections, each right-angle turn in the pipeline was accomplished by two 45° bends. Laterals were provided for bypassing injection boreholes M and C during times when boreholes D or B were being used for injection. The pipeline system included 2,814 feet of pipe, of which 2,364 feet were buried about 2 feet below street level as shown in figure 33.

The drainage borehole was used to dispose of water left in the pipeline at the close of each injection period. Purging and draining the system each night prevented clogging of the pipeline and freezing during winter.

During the first few days of the demonstration project, each shutdown of the deep well pumps was accompanied by severe water hammer (dynamic pressure added to the normal static pressure of the return flows). At the time of shutdown, the supply of solids had been cut off, and water was being circulated to purge the slurry pipeline. For purging, the bypass was used that connected the waterline directly to the distribution pipeline and injection borehole, rather than through the mixing tank and slurry pump. When the water wells were shut down, flow in the pipeline reversed direction, moving back toward the deep well pumps where the elevation was about 25 feet lower than the injection borehole collar. The check valves at the well pumps had a small opening that impeded return flow, and water hammer resulted. The condition was alleviated somewhat by first shutting down one pump and gradually slowing down the other. The problem was not completely solved until a drainage borehole was drilled adjacent to the low point in the pipeline, and

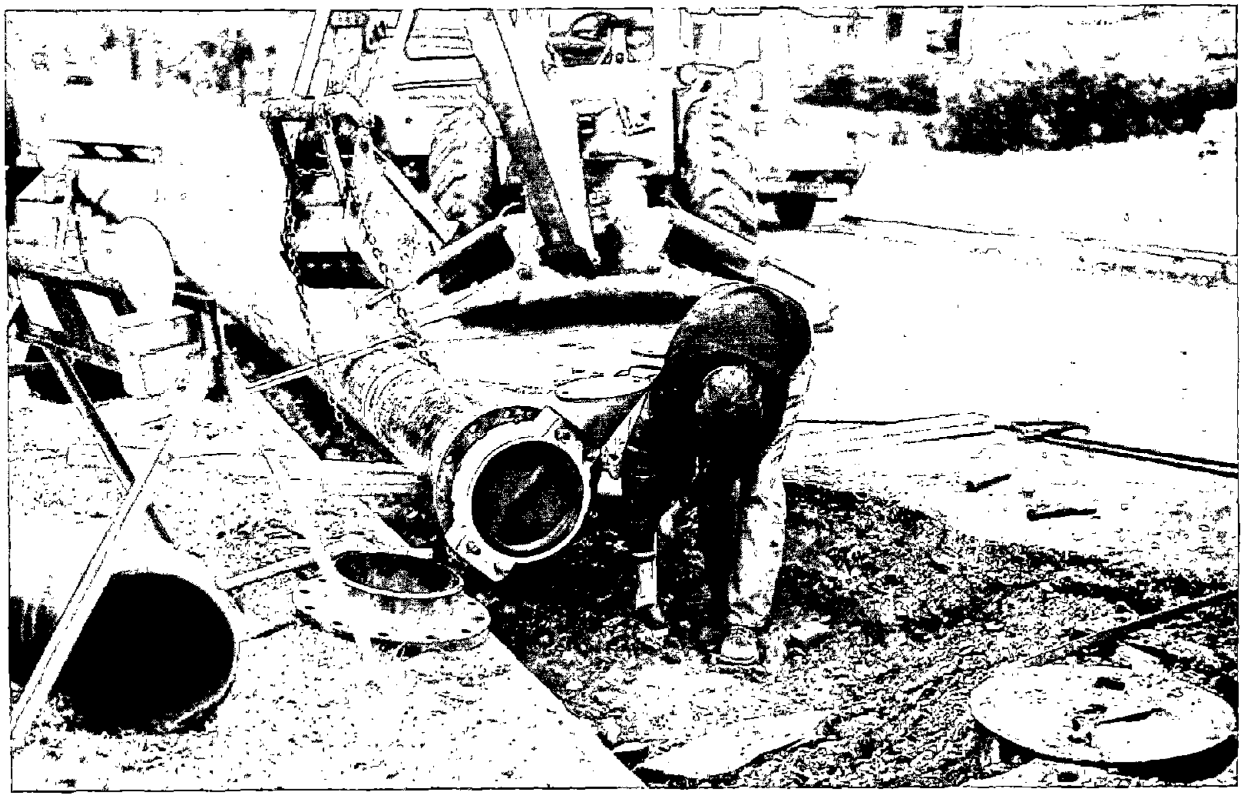

FIGURE 33. - Placing slurry distribution pipeline below street level; inserting butterfly valve.

the return flow was diverted away from the pumps through the drainage borehole into subsurface mine openings.

Slurry pressures were monitored by gages at two locations in the distribution pipeline system. A recording gage close to the slurry pump indicated the pressures imparted by the slurry pump (total dynamic head) to move the total quantity of slurry in the pipeline at the required velocity through the existing static head and to overcome the friction head in the entire system. For the various injection boreholes, the length of the discharge line ranged from a minimum at 1,215 feet to a maximum of 2,165 feet horizontal distance and from 25 to 37 feet of vertical lift. During most of the injection period when little or no underground resistance was encountered, the pressure at the slurry pump ranged between 30 and 70 psig, averaging about 60 psig. When back pressure was caused by resistance underground, pressures at the slurry pump ranged between 70 and 95 psig.

Pressures were also measured at the top of each injection borehole. Due to the vertical drop of the slurry column to the mine levels, pressures at the injection boreholes were in the vacuum range (less than atmospheric) during most of the injection period. When resistance underground was encountered, however, pressures above atmospheric pressure were recorded at the top of the injection borehole. Periods of pressure buildup are discussed in the section on the progress of the backfilling operations.

The first elements of the slurry distribution pipeline system to show wear were the valves. Originally, butterfly valves were installed to control discharge from the slurry pumps and to divert the flow of slurry to laterals when injection operations were changed from one borehole to another (fig. 33). During the injection periods while the valves were open, the impact of the slurry stream eroded the part of the rubber coating that faced upstream. Figure 34A shows the amount of wear on one butterfly valve after only 55,000 tons of slurry had been injected. On another valve shown in figure 34B, which was in use during injection of 189,000 tons, maximum erosion was 1-3/8 inches. In early February after 146,760 tons of solids had been pumped through the pipeline, the two slurry discharge-control valves were replaced with air-operated pinch valves. These valves were closed by squeezing a rubber insert. Figure 35A shows the interior of the valve in open position; figure 35B shows the exterior surface of a worn liner. Liners were replaced in the pinch valves after injection of approximately 100,000 tons of solids.

After the bypass butterfly valves at the laterals located close to the injection boreholes had become useless, changes from one injection borehole to another were accomplished by welding steel blanks as cutoffs and later removing them as slurry lines were reactivated.

The slurry pipeline was composed of welded lengths of abrasive-resistant carbon steel pipe having a yield strength of 54,000 psi and tensile strength of 99,000 psi. The outside diameter was 14 inches, and wall thickness was 0.375 inch. During the project, injection was interrupted 66 times for repair of pipeline leaks. The earliest detected pipeline wear occurred at an elbow connection to the M injection borehole after 43 shifts of injection and the passage of 98,870 tons of solids. A long-radius steel ell was replaced after injection of 119,900 tons of minus 1/4-inch material and 11,475 tons of minus 1/2-inch material. By the fifth month of injection, leaks at elbow turns developed frequently. Repairs were facilitated by the placement aboveground of elbow turns connecting horizontal pipe to vertical boreholes wherever feasible. Minor obstructions such as weld-bead created turbulence resulting in erosion.

Leaks in the straight segments of the distribution pipeline developed later and with less frequency. Perforations in straight pipe developed in the underside of the pipe. Repairs were made in buried pipe by cutting the damaged pipe segment, rotating it to place the former top half at the bottom, patching, and welding.

At the close of the demonstration project, the pipeline between the mixing plant and Monsey Avenue was taken up and examined. Signs of wear extended from the bottom up each side for about 20°. The cross section of pipe shown in figure 36 shows the change in wall thickness at the bottom of the pipe. A micrometer-caliper survey was made of the residual thickness of the pipe wall. The results of the survey are plotted in figure 37, together with the locations of repairs made during the course of the project.

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FIGURE 34. - Wear on butterfly valves. *A*, After injection of 55,000 tons of solids; *B*, after 189,000 tons, maximum erosion 1-3/8 inches.

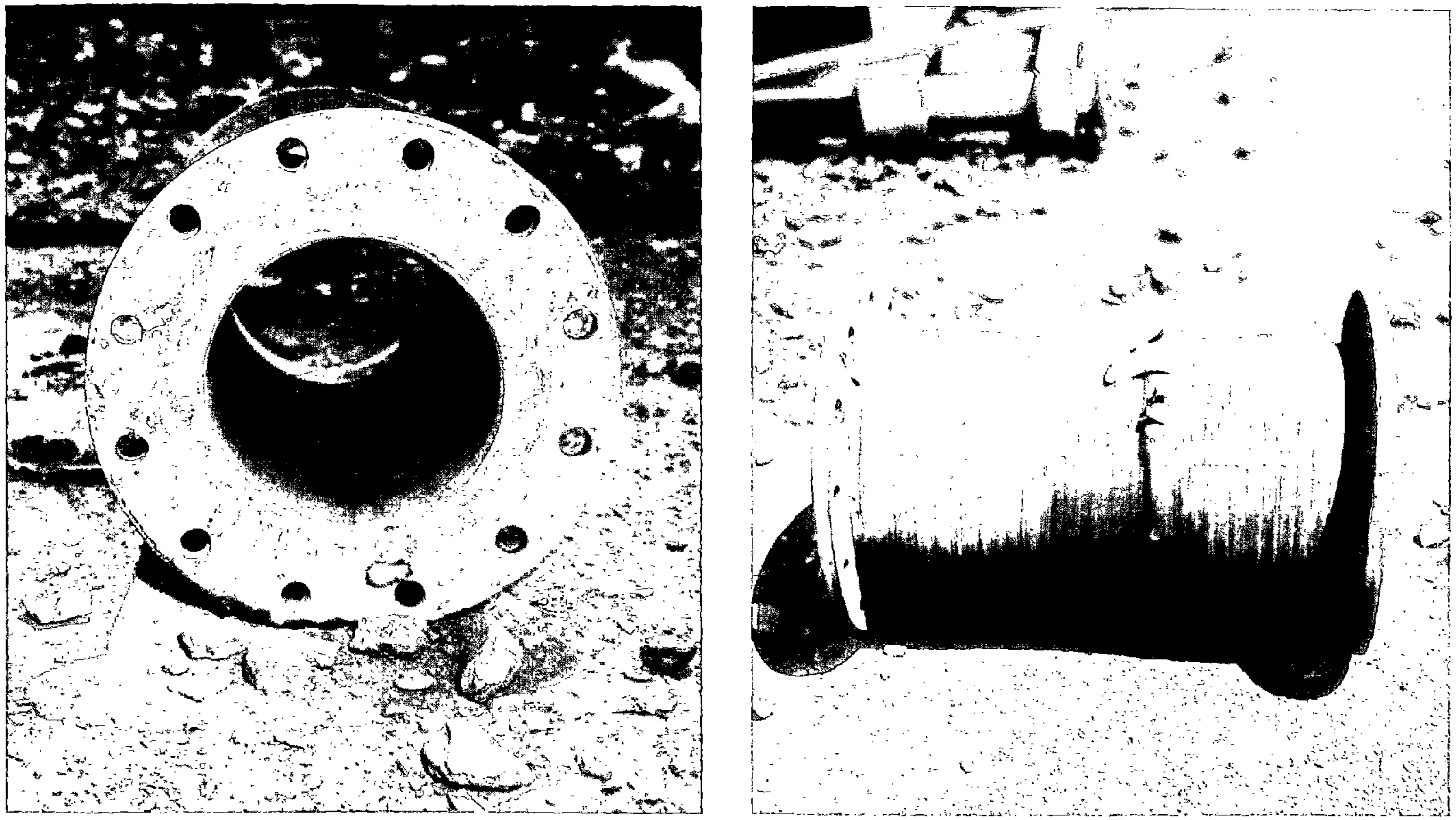

FIGURE 35. - Air-operated pinch valve. *A*, Interior view; *B*, exterior view of worn rubber liner.

FIGURE 36. - Section of worn slurry-distribution pipe.

Injection Boreholes

Three factors were considered in selecting the location for the injection boreholes. The economic advantage of the closed-system method of injection is the large area that can be filled from a single injection borehole. Thus the preferred location is near the center of the area to be backfilled. For the area of the project as originally defined, study of the mine maps indicated that the lower (Clark) coalbed workings could be filled from a single borehole. The location for injection borehole M, for backfilling the Clark bed, was placed at the center of the project area, on Monsey Avenue about 150 feet southwest of its intersection with Delaware Street.

The second consideration in locating injection boreholes is the presence of obstructions within the mine that might interfere with a radial pattern of backfill distribution. The mine map of the upper (New County) coalbed showed three lines of manmade barriers (stoppings for ventilation purposes), which apparently divided the area to be backfilled into four segments (fig. 13). Four injection boreholes were provided for injection to the New County bed to insure coverage of the four segments and also to provide injection boreholes to the part of the bed that would be above the level of the mine pool at the time of injection. One objective of the demonstration project was to find out whether the pressure method of backfilling could be used in dry as well as inundated mines.

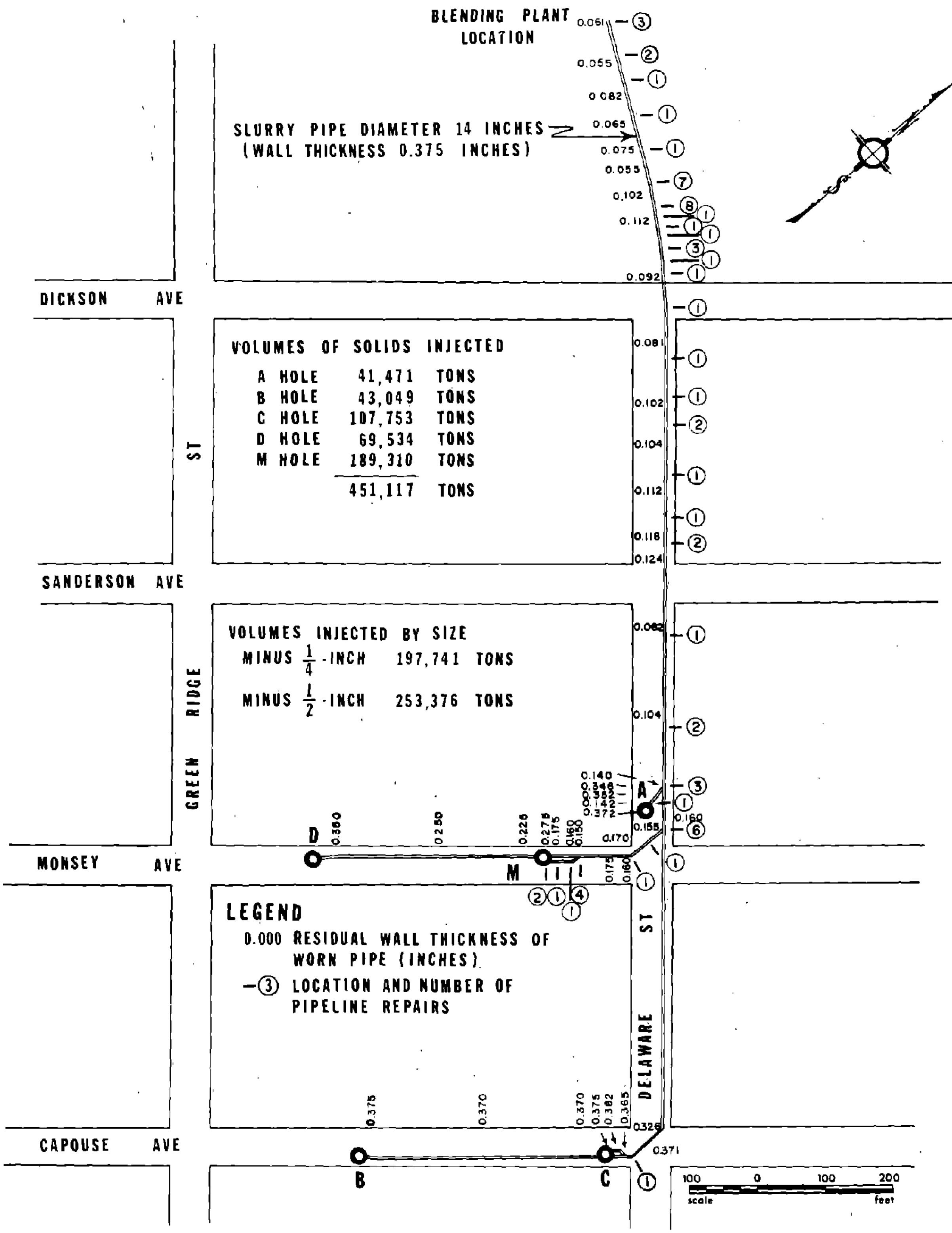

FIGURE 37. - Map of slurry-distribution pipeline, showing variations in thickness of pipe wall after injection of 451,117 tons of solids, locations of pipe repairs, and numbers of repeated repairs at same location.

The third requirement in selecting the location for an injection borehole is to intersect open passageways at the mine level, so that the slurry can move out with minimum obstruction. For this reason, 6-inch probe holes were drilled to verify the position of openings that were indicated on the mine maps. Five probe holes encountered open spaces in the mine workings at the specified coalbed. The thickness of open space and broken rock are shown on table 3. One probe hole, drilled to relocate an injection borehole, encountered solid coal instead of a mine opening. In the probe holes, sonar surveys were made at the level of the mine workings to indicate the lateral extent of open passages. Where more than one bed is to be backfilled, injection boreholes should be located to superimpose fill in an upper bed directly over the backfilled part of the lower bed.

TABLE 3. - Injection borehole data

Injection borehole	Character of mine opening[1]		Depths of casing, ft	
	Open, ft	Broken rock, ft	Standpipe (18-in-diam)	Inner casing (13-3/8-in-OD)
A	5	5	56	140
B	6	0	33	123
C	5	2	37	131
D	6	3	48	123
M	4	15	49	176

[1]Mine openings reported in injection boreholes A, B, C, and D were in upper (New County) coalbed; mine opening reported in borehole M was in lower (Clark) coalbed.

After the locations for injection boreholes had been established by the probe holes and sonar surveys, the injection boreholes were completed by enlarging the probe holes and installing casing. An outer standpipe, or collar, of 18-inch casing extending from the ground surface through the alluvium, was cemented to rock. The inner casing, 13-3/8 inches outside diameter, which conveyed the slurry from the ground surface to the mine opening, was K-55 casing weighing 52.50 pounds per foot with wall thickness of 0.380 inch. The lengths of casings are listed in table 3; the inner casings terminated 5 feet above the mine roof. Attempts to grout the base of the inner casing to rock were only partially successful.

When the decision was made to enlarge the backfilling project to inject 450,000 tons, the quantity to be injected into the Clark coalbed was increased from an estimated 178,000 tons to about 264,000 tons. At that time, 189,310 tons had been injected into the Clark bed through the M injection hole, and continuing back pressures indicated that the practical point of refusal had been reached. Two of the New County injection boreholes, D and C, were then deepened to the Clark bed. In these boreholes, the segment of casing between the two coalbeds was 10 inches in diameter instead of 13-3/8 inches as in the rest of the injection pipeline.

After work was completed in the Clark bed, the casings in D and C were cut opposite the upper bed to permit injection into the New County bed as planned originally.

Monitoring System

For the purpose of tracking the progress underground of fill material as it was being injected into the mine openings, a network of monitor holes was established. Most of the boreholes were drilled well in advance of injection operations, and they served also as exploratory drillholes to obtain the subsurface data needed to make detailed plans for the project. The monitor holes also provided access for lowering the sonar tool to mine level to gain information on the extent of open space within the mine workings.

These boreholes were necessarily located on streets because of the impracticability of operating drilling equipment in people's yards. Boreholes were drilled along five northeast-southwest avenues (Penn, Capouse, Monsey, Sanderson, and Dickson) and two northwest-southeast streets (Delaware and Green Ridge), totaling 46 in number. The area covered by the monitor holes extended somewhat beyond the design perimeter of backfilling in order to define the actual limits of backfill as emplaced (fig. 10). Of the 46 monitor holes drilled, only one failed to intersect an open space in the mine; most of the boreholes intersected openings in both coalbeds.

For each monitor hole, an outer standpipe 8 inches in diameter extended from the ground surface to a depth 2 feet below the top of bedrock and was cemented to rock. The inside casings (6 inches in diameter) were suspended from the surface so that they could be lowered and raised as necessary to provide access to, or to seal off, the upper coalbed. For this reason, that part of the borehole below the maximum casing depth was drilled with a 6-inch bit to provide a rock shelf on which the casing rested when extended. Before injection started, the casings were installed only to a point above the upper coalbed, to permit sonar surveys to be made in both mine levels. The casings were then lowered to seal off the upper bed while slurry was being injected into the lower bed and later raised during injection to the upper bed.

Each monitoring borehole was capped by an 8-1/2-inch-diameter flanged cover with an asbestos gasket, held down by eight 3/8 by 2-inch bolts. The center of each cover was tapped with a 2-inch plug. The borehole caps were depressed about 2 inches below pavement level to avoid interference with traffic. Once injection was well underway, the monitor holes were opened after each operating shift to measure the depth to any backfill material that had been deposited and the water-level depth in the borehole. Measurements were made by removing the plug from the cap and lowering a weight on a calibrated cable. A float was put on the line to measure water levels

The presence of monitor holes close to the injection boreholes was found to provide a point of pressure relief during injection. At times when the slurry encountered high resistance in seeking to open up new outlets through the packed fill, some energy was dissipated in sending slurry or water up through monitor holes (though this could also be considered a safety factor). Close spacing of boreholes drilled for any purpose also increases the chance for communication between coalbeds to the extent that slurry can migrate along the outside of the casing.

Progress of the Backfilling Operation

Overview

The duration of slurry injection at the Green Ridge site was nearly 8 months, from October 26, 1972, to June 21, 1973. The daily work period was one 8-hour shift during the first 5 months and changed on March 19 to two 8-hour shifts. A total of 233 shifts were worked during the overall injection period. The only major interruption to slurry injection (for a period of 19 shifts) was caused by the breakdown of one of the deep well submersible pumps on April 19; injection was resumed on May 3. The average rate of injection was 2,000 tons of solids per 8-hour shift.

A total of 451,117 tons of solids was injected into two mined coalbeds. Of this quantity, 263,518 tons or 59 percent was injected into the lower (Clark) coalbed, and 187,599 tons or 41 percent, into the upper (New County) bed. These proportions were required to provide support for essentially the same area on the ground surface because of differences in the average thickness of the coalbeds (Clark bed, 8 feet; New County bed, 6 feet) and in the percent of coal that was extracted (Clark bed, 69 percent; New County bed, 64 percent). The surface area overlying the backfilled portions of the coalbeds was estimated to be approximately 30 acres.

Five injection boreholes were used in backfilling the two beds. Two of these boreholes served for injecting material into both beds. The sequence in which the injection boreholes were used is shown diagrammatically in figure 38. Ideally, all injection into the lower bed would be completed before starting injection into the upper bed. In the actual course of work, however, injection into the lower bed was interrupted twice for brief periods. The first interruption was for the purpose of gaining information on the feasibility of injecting slurry into dry mine cavities. The other interruption was a result of enlarging the area of the project after the plans were originally made. When injection through the M borehole was completed, two other injection boreholes had to be deepened in order to inject additional material into the lower bed. While the drilling was being done, slurry was injected into the upper bed to avoid interrupting the backfilling operation.

Clark Bed

The mine workings in the lower (Clark) coalbed were backfilled with 263,518 tons of crushed mine refuse injected in discontinuous periods totaling nearly 23 weeks (126 shifts). Three injection boreholes were used designated as follows: M, which was used for the lower bed only; C and D, which were originally intended for the upper bed only but were subsequently deepened to reach the lower bed. The estimated extent of the backfilled area is shown on the map of the mine workings in the Clark bed, figure 12.

The progress of backfilling operations in the Clark bed is summarized in tables 4 and 5. Table 4 contains operational data on injection rates, pressure ranges, and slurry composition for the periods during which each injection borehole was being used. For the same periods, table 5 summarizes the underground progress of the distribution of the fill material.

TABLE 4. - Operating data during injection into Clark bed

Injection borehole	Injection period: Elapsed time (Number of 8-hr shifts)	Injection period: Hours of inject-ing solids[1]	Number of tons injected[2]	Average rates of injection: Water, gpm	Average rates of injection: Solids, tph	Pressure range: At slurry pump, psig	Pressure range: At injection borehole, psig[3]	Slurry composition: Pounds solids per gallon water	Slurry composition: Water:solids ratio, By weight	Slurry composition: Water:solids ratio, By volume
M (1st)...	10/26/72-1/12/73 (53 shifts)	384	107,138	7,097	279	45-70	-4.9 to 12	1.3	6.4	14.0
M (2d)....	2/12-3/29[4] (42 shifts)	265	82,172	7,234	310	50-90	-7.4 to 65	1.4	5.8	12.8
D deepened	4/16-4/19; 5/3-5/9[5] (16 shifts)	109	37,818	6,107	347	70-100	-9.8 to 50	1.9	4.4	9.6
C deepened	5/9-5/18 (15 shifts)	97	36,390	5,687	375	85-95	0 to 10	2.2	3.8	8.3

[1]Excludes times of pumping with clear water and times when pumps were shut down for repairs to pipeline.
[2]The volume occupied by 1 ton of crushed refuse is approximately 1 cubic yard.
[3]Pressure gages at the top of the injection boreholes were compound gages, measuring vacuum (pressures below atmospheric) in inches of mercury and pressures above atmospheric in psi. Gage readings corresponding to the negative psi values given are: -4.9 psig=20 in Hg; -7.34 psig= 15 in Hg; -9.8 psig= 10 in Hg.
[4]On Mar. 19, 1973, the work schedule changed from an 8-hour day to a 16-hour (two-shift) day.
[5]Injection was interrupted from 4/19 to 5/2 owing to breakdown of water pump.

TABLE 5. - Progress of fill distribution in Clark bed

Injection borehole	Period of injection (Number of 8-hr shifts)	Number of tons injected	Distance traveled underground[1]			
			Monitor hole	Date filled	Distance from injection point, ft	Gradient
M (1st)...	10/26/72-1/12/73 (53 shifts)	107,138	12	10/27	80	Level.
			26	11/3	80	Level.
			11	11/3	170	Level.
			16	11/13	330	Level.
			15	11/22	475	Level.
			13	11/23	205	Upgrade.
			17	12/8	340	Downgrade.
			20	12/26	485	Downgrade.
M (2d)....	2/12/73-3/29/73 (42 shifts)	82,172	[2]4	2/13	300	Upgrade.
			6	2/27	445	Upgrade.
			34	3/13	640	Level.
			36	3/22	385	Level.
			24	3/26	555	Upgrade.
D deepened	4/16/73-4/19/73[3] 5/3/73-5/9/73 (16 shifts)	37,818	[2]10	4/18	195	Upgrade.
			9	5/3	205	Level.
			22	5/3	370	Downgrade.
			7	5/7	440	Upgrade.
			37	5/7	630	Upgrade.
			8	5/8	360	Upgrade.
C deepened	5/9/73-5/18/73 (15 shifts)	36,390	[2]2	5/11	130	Upgrade.
			5	5/11	120	Level.
			25	5/18	230	Upgrade.

[1]Evidence of distance traveled underground is depth of fill measured in monitor holes. Distances listed are measured in direct line; actual distances traveled usually greater due to location of pillars. (See figure 12 for locations of boreholes.)

[2]Only those monitor holes are listed that had not previously received fill to the level of the mine roof; additional fill was recorded in some of the previously filled boreholes.

[3]Injection interrupted 4/19-5/2 owing to breakdown of water pump.

The operational data in table 4 were based on quantities of water and solids injected each day and hours of slurry pump operation, which were reported on daily operation logs. Pressure measurements were made twice daily (rather than continuously monitored) and oftener at times of rapidly rising pressures. The range of values reported at the slurry pump and at the injection boreholes is given on table 4. Slurry density fluctuated continuously because the rate at which solids were fed to the tank was irregular (ranging from near zero to 500 tph and occasionally reaching 600 tph), whereas the rate of waterflow was fairly constant except for periods of purging with clear

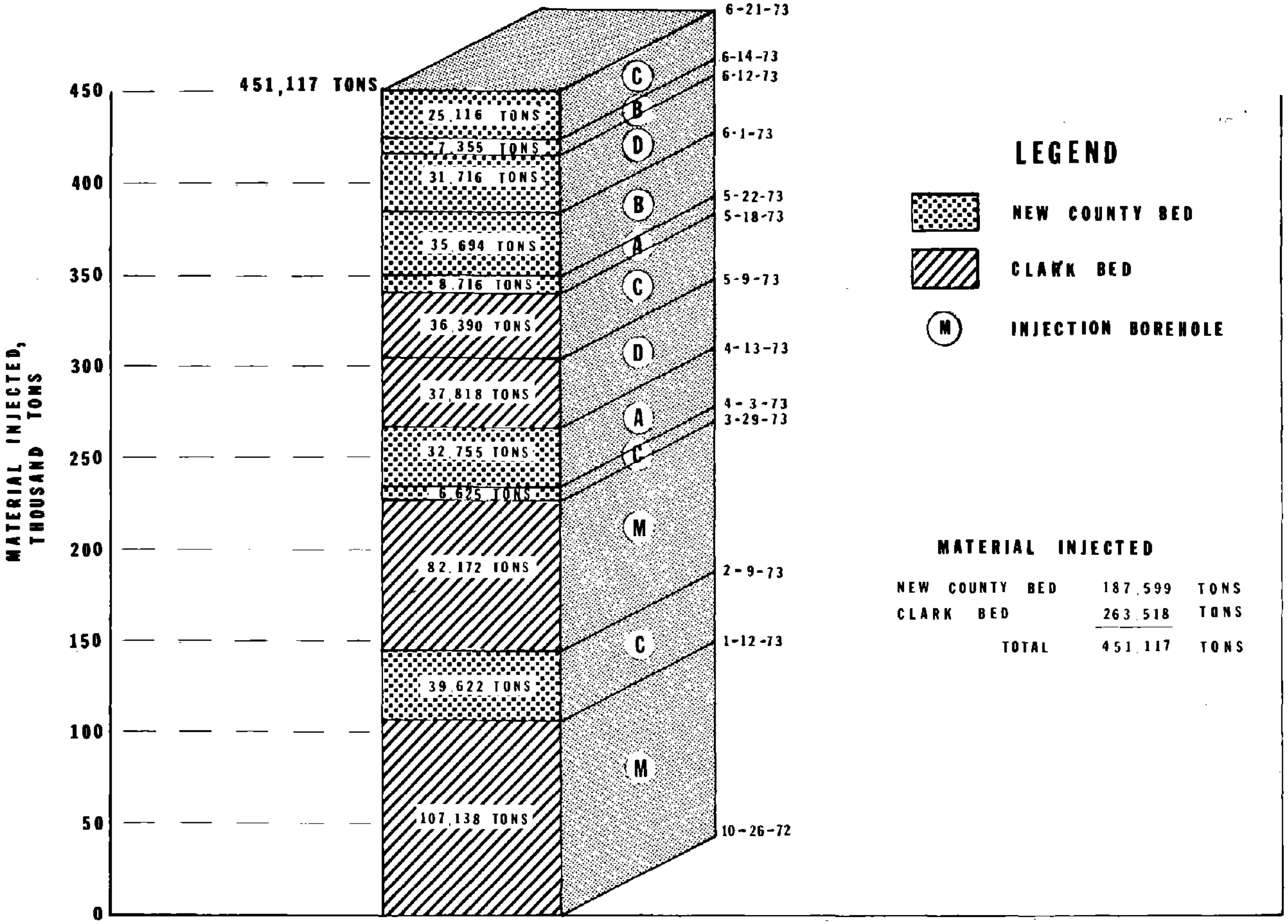

FIGURE 38. - Diagrammatic representation of injection of solids into two mined coalbeds in Green Ridge demonstration project showing sequence in which injection boreholes were used.

water. Actual densities may have been somewhat higher than indicated on table 4, especially during the second period of injection through borehole M. It was not possible to determine and eliminate from the total quantities of water used the amounts of clear water used in purging for only short periods of a few minutes duration.

The evidence for the subsurface extent of the emplaced fill consisted of significant changes in depths of the monitor holes, which were measured daily. In the Clark bed, which was inundated, the first arrival of fill material in mine voids below the monitor holes indicated that the fill material moved forward in sufficient thickness to essentially fill the vertical extent of the mine opening. In table 5, the monitor holes are listed in order of arrival time of the fill material to demonstrate the irregular pattern of progressive distribution. Each monitor hole is listed only once to record the period during which the mine void at that point was first filled. During the course of continued injection, backfill material actually rose to considerable heights in most monitor holes.

Injection Through Borehole M

The M borehole was used for injection over a total time of 4 months (95 shifts). The quantity injected (189,310 tons of solids) was undoubtedly the largest quantity to date of fill to be injected hydraulically through a single borehole directly into mine voids without the use of manually controlled distribution within the mine.

Because crushed mine refuse was injected by the new process for the first time in this demonstration project, the operation started conservatively with a very liquid slurry. About 4-1/2 tons of solids were injected per minute with a waterflow rate of about 7,000 gpm (a water-to-solids ratio of 14 by volume). As the operation proceeded successfully, the solids injection rate was gradually increased. This resulted in an overall corresponding decrease in waterflow rate (because of displacement and the fixed capacity of the mixing tank) and increased density of the slurry (though the average water-to-solids ratio by volume remained above 8).

During the first 8 weeks (34 shifts) of injection through borehole M, during which approximately 70,000 tons of solids was injected, no resistance to the underground movement of slurry was detected. Pressure at the top of the injection borehole was consistently in the vacuum range (below atmospheric). Fill material being distributed underground reached the positions of seven monitor holes; the most distant hole was 475 feet from the point of injection as shown in table 5.

During the remaining 4 weeks of the first period of injection through M, one additional monitor hole was reached at a distance of 485 feet. The first evidences of pressure buildup were reported toward the end of December. The pressure at the top of the injection borehole occasionally rose for periods of a few minutes to 5, 10, and 12 psig, although daily pressure readings remained mostly in the vacuum range. Resistance to horizontal movement in the mine workings caused fill material to rise in monitor holes for vertical distances ranging from 15 to 42 feet above mine level. On December 27 when the cap was removed from monitor hole 13 to measure the level of fill material, both solids and water erupted high above the ground surface. The slurry pump had to be stopped in order to replace the cap. The procedure then adopted was to stop the flow of solids whenever pressure at the injection borehole gage reached 5 psig, and to pump clear water until the pressure reverted to vacuum range. With more experience in pumping under pressure in later phases, the threshold for cutting off solids was gradually increased, reaching 50 psig toward the end of the project.

After being interrupted by 4 weeks of injection into the upper coalbed, injection through borehole M was resumed. During the 7 weeks (41 shifts) of the second period of injection through M, 82,172 tons of material was injected. The maximum distance that fill material moved underground from an injection point was recorded during this period (640 feet to monitor hole 34). The areal extent of backfill was increased to include the locations of six additional monitor holes that had not been reached previously (table 5).

Two operational changes took place during the second period of injection through M. The changeover from the use of minus 1/4-inch crushed refuse to minus 1/2-inch material started February 22 and was completed by March 7. On March 19, the daily injection period was changed from 8 hours to 16 hours. The outstanding characteristic of this period was the marked increase in resistance to injection. Manifestations of increased resistance included--

1. Higher pressures at the slurry pump,

2. Higher pressures at the top of the injection borehole,

3. The rise and fall of the level of fill within monitor holes,

and 4. The presence of trapped water in monitor holes and erratic fluctuations of water level.

1. Pressure at the slurry pump that was encountered in conveying the slurry to the injection borehole during the first injection period ranged up to 70 psig. During the second period of injection through M, the upper limit of pressure at the slurry pump gradually rose to 90 psig as resistance to injection increased.

2. Pressures at the top of the injection borehole fluctuated from vacuum to positive, requiring frequent interruptions of slurry injection to pump clear water. Pressures above atmospheric became progressively higher and more frequent. In the third week of the second period of injection through M, pressures ranged from -7.4 to 10 psig. When clear water injection was started in the morning, pressures of 35 to 52 psig lasted for periods of 2 to 14 minutes before dropping to the vacuum range. Operating pressures during the fourth week ranged from -7.4 to 45 psig with starting pressures as high as 75 psig. During the seventh and final week the pressure rose to 65 psig. Because of the risk that solids would settle out of suspension in the pipeline due to reduction of flow velocity, further use of the M injection borehole was discontinued after 96 shifts.

3. The daily measurements of fill deposits in several monitor holes indicated frequent changes in the heights to which fill material rose within the casing. Table 6 shows changes in heights of fill according to measurements made during the second period of injection through M. The boreholes apparently served as pressure relief holes at times when the incoming slurry flow encountered resistance, indicating that mine openings surrounding the injection point had become blocked temporarily. As the slurry attempted to force a passageway along the top of the emplaced fill, some of the slurry rose in the casings of monitor holes. Then, when a passageway was cleared, the material in the borehole above that channel apparently dropped back by gravity to mine roof level. At monitor hole 26, only 80 feet from the injection borehole M, water and slurry came up to the surface along the outside of the casing. On February 16, the casing was pulled and the borehole was temporarily plugged with fill material, but continuing pressure at mine level caused the fill material to flow out of the hole. On March 2, the hole was permanently plugged with concrete.

4. A conspicuous phenomenon during the last half of the second period of injection through M was the presence of water in monitor holes, which became trapped temporarily above the level of the mine pool. The changes in water level within the boreholes were large in magnitude and they took place rapidly. When possible, measurements were made twice daily, and several changes greater than 100 feet took place between morning and afternoon measurements. In figure 39, changes in water levels in three monitor holes are plotted to show their relation in time. The fact that water-level changes took place at different times in different holes showed that each borehole was hydraulically independent for periods of a few days or hours. The water level changes constituted a more sensitive indicator of the same type of pressure changes that were indicated by changes in the level of fill in monitor holes.

Injection Through Boreholes C and D Extended to Clark Bed

The remaining quantity of fill material apportioned for the Clark bed under the enlarged project plan was injected through boreholes C and D, which had been extended to reach the lower bed. The operational data and the underground distribution of the fill are summarized in tables 4 and 5. Solids injected through D filled in the area south and southwest of the fill that had been emplaced previously, reaching the location of five additional monitor holes. The area to the southeast of the previously backfilled portion was filled by injection through C, reaching three additional monitor holes. Pressures at both injection boreholes were somewhat higher than at other injection boreholes because the casings in the extended portions of C and D were 10 inches in diameter whereas the remainder of the injection pipeline system was 13-3/8 inches in diameter. The density of the slurry was increased during injection through D to an average of 1.9 pounds per gallon of water. The use of D for the Clark bed was abandoned at the end of 16 shifts in which 37,818 tons of solids were injected, when pressure at the injection borehole rose to 50 psig. No excessive pressures were encountered while injecting through C to the Clark bed. The average slurry density was further increased to 2.2 pounds per gallon. Injection proceeded for 15 shifts until the lower bed had received its proportionate share of the fill.

TABLE 6. - Heights to which fill material rose in monitor holes during second period of injection into Clark bed, feet

Monitor hole	February 1973													March 1973																				
	12	13	14	15	16	19	20	21	22	23	26	27	28	1	2	5	6	7	8	9	12	13	14	15	16	19	20	21	22	23	26	27	28	29
4	-	3	5	-	-	-	-	-	6	-	-	-	-	-	-	-	-	-	-	-	-	-	17	-	-	-	-	-	-	-	-	-	-	16
6	-	-	-	-	-	-	-	-	-	-	-	7	8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	14	-	-	-	-	-	-
11	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	13	8	-	-	19	-	-	-	-	-	-	27	-	-	-	-	28	-
12	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	17	10	-	-	14	-	-	-	13	-
13	-	-	-	-	4	-	-	-	7	4	-	9	13	20	22	23	-	4	-	-	11	7	8	-	-	-	-	-	33	34	-	-	22	-
15	32	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	7	-	-	-	-	10	31	7	9	26	-	-	-	28	-
16	-	-	31	-	30	-	-	-	-	-	-	35	-	-	-	21	-	90,27	-	9	-	-	-	-	-	-	-	10	33	55	-	-	-	-
34	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	6	10	-	-	-	-	26	20	21	-	6	53	-

NOTE.--Locations of boreholes are shown in figure 10.

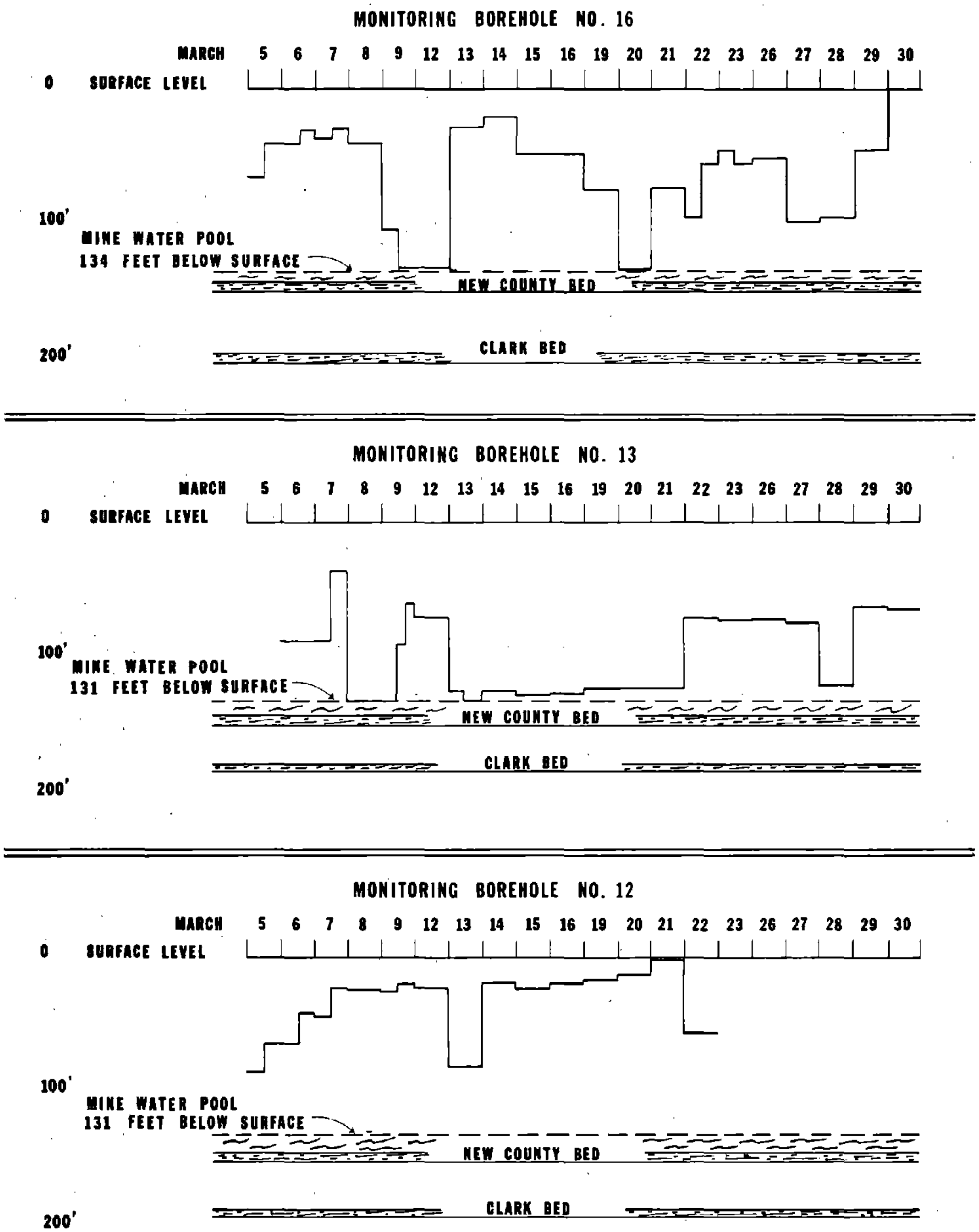

FIGURE 39. - Rapidly fluctuating levels of water trapped in monitor holes during second period of injection through M into Clark bed.

New County Bed

In the upper (New County) bed, conditions differed from those in the lower bed in that the area to be backfilled was believed to be segmented by substantial manmade partitions (for ventilation control) within the mine workings; also, part of the area was above the level of the mine pool. Injection into the part of the area above water level was through boreholes B and C. Boreholes A and D were used for injecting into the inundated portion. Most of the injection boreholes were used more than once for logistical reasons; the sequence of their use is shown in figure 38. In boreholes C and D, which had been cased through the New County bed and used for injection into the lower bed, access to the New County bed was reestablished by cutting the casings with the use of an explosive device. The casings could not be raised in the normal manner because they had been forced into holes that were not quite plumb.

The total quantity of fill material placed in the upper bed was 187,599 tons. The estimated extent of the backfill emplaced in the mine workings is shown in figure 13. Table 7 summarizes the progress of the underground distribution of fill material during injection through each of the four boreholes that were used to backfill the New County bed.

Operating data during the periods of injection into the upper bed are summarized in table 8. The waterflow rate, the rate of injection of solids, and the slurry density varied, in part reflecting pressure conditions. During the two periods in which the average slurry density was 2.2 pounds of solids per gallon of water or higher, the rise in back pressure after a day or two of injecting led to the decision to abandon further use of injection boreholes A and B.

Injection Into Area Above Mine Pool Level

Injection into borehole C was carried out in three separate periods covering a total time of thirty-six 8-hour shifts. A total of 71,363 tons of solids was injected into the New County bed through C. The level of the mine pool during these periods was between 11 and 6 feet below the floor of the mine at the location of C. Material injected through C filled the central and eastern parts of the project area. The area backfilled through C was elongate parallel to the long axis of the rooms in the mine workings. The manmade partitions appeared to control the general outline of the filled area, but the partitions were evidently breached at several locations. The fill extended downgrade about twice the distance of the upgrade extent from the point of injection.

Injection borehole B was 5 feet above pool level during the two periods of its use. The quantity of solids injected in two periods covering seventeen 8-hour shifts was 43,049 tons. Material injected through B filled the southern part of the project area.

TABLE 7. - Progress of fill distribution in New County bed

Injection borehole	Period of injection (Number of 8-hr shifts)	Number of tons injected	Distance traveled underground[1]				Water conditions at mine level
			Monitor hole[2]	Date(s) filled	Distance from injection point		
					Horizontal, ft	Vertical, ft	
C (1st) (dry)..	1/15-2/9 (20 shifts)	39,622	40	1/18	220	-10	Half inundated.
			41	1/24	125	+2	Dry, 9 ft above water level.
			44	1/26	230	+16	Dry, 23 ft above water level.
			3	1/29	190	+9	Dry, 15 ft above water level.
			43	1/29-1/31	220	-5	Dry, 1 ft above water level.
			42	2/9	315	-20	Under 13 ft of water.
			[3]27	2/9	450	-28	Under 21 ft of water.
C (2d) (dry)...	3/30-4/3 (4 shifts)	6,625	1	4/3	335	+24	Dry, 31 ft above water level.
A (1st)........	4/4-4/13 (15 shifts)	32,755	14	4/4	70	-4	Under 26 ft of water.
			[3]27	4/4	90	+3	Under 19 ft of water.
			17	4/13	215	-11	Under 33 ft of water.
			15	4/13	340	+16	Under 6 ft of water.
A (2d).........	5/21-5/22 (3 shifts)	8,716	([4])	-	-	-	-
B (1st) (dry)..	5/22-5/1 (14 shifts)	35,694	8	5/29-5/30	255	0	Dry, 5 ft above water level.
			[5]10	5/20-5/30	400	-7	Half inundated.
D..............	6/4-6/12 (13 shifts)	31,716	9	6/5	200	-1	Under 11 ft of water.
			20	6/8	410	-20	Under 30 ft of water.
			[5]10	6/11	200	+8	Half inundated.
B (2d) (dry)...	6/12-6/14 (3 shifts)	7,355	37	6/14	300	+18	Dry, 23 ft above water level.
C (3d) (dry)...	6/14-6/21 (12 shifts)	25,116	46	6/18	305	+5	Dry, 12 ft above water level.

[1]Evidence of distance traveled underground is depth of fill measured in monitor boreholes. Distances listed are measured in direct line; actual distances traveled are probably greater due to location of pillars. The vertical measurement is the difference in altitude of the bed between the injection borehole and the monitor hole.
[2]Monitor holes are listed for only those periods during which they first received fill material to the level of the mine roof. (See figs. 10, 12, and 13 for locations of monitor holes.)
[3]Partially filled from injection borehole C; filling was completed from injection borehole A. Mine pool level rose 2 feet between Feb. 9 and Apr. 4, 1973.
[4]No monitor holes received fill for the first time during this period. Additional fill was recorded in some previously filled boreholes, indicating fill material in the casing, above the level of the mine roof.
[5]Partially filled from injection borehole B; filling was completed from injection borehole D.

TABLE 8. - Operating data during injection into New County bed

Injection borehole	Injection period: Elapsed time (Number of 8-hr shifts)	Injection period: Hours of injecting solids[1]	Number of tons injected[2]	Av rates of injection: Water, gpm	Av rates of injection: Solids, tph	Pressure range: At slurry pump, psig	Pressure range: At injection borehole, psig[3]	Slurry composition: Pounds solids per gallon water	Slurry composition: Ratio of water to solids: By weight	Slurry composition: Ratio of water to solids: By volume
C (1st)...	1/15-2/9 (20 shifts)	145	39,622	5,871	273	45-90	-6.9 to -5.4	1.5	5.4	11.8
C (2d)....	3/30-4/3 (6 shifts)	22	6,625	6,580	301	65-85	-7.4 to -4.9	1.5	5.5	12.1
A (1st)...	4/4-4/13 (16 shifts)	107	32,755	7,417	306	40-75	-4.9 to 35	1.4	6.1	13.3
A (2d)....	5/21-5/22 (3 shifts)	22	8,716	5,894	396	60-100	0 to 58	2.2	3.8	8.3
B (1st)...	5/22-6/1 (15 shifts)	101	35,694	5,952	353	65-100	-3.9 to -4.9	2.0	4.2	9.2
D.........	6/4-6/12 (13 shifts)	90	31,716	5,822	352	70-95	-12.3 to 45	2.0	4.1	9.1
B (2d)....	6/12-6/14 (3 shifts)	21	7,355	4,945	350	75-95	-3.9 to 35	2.3	3.5	7.7
C (3d)....	6/14-6/21 (12 shifts)	86	25,116	5,312	292	70-90	-12.3 to 15	1.8	4.6	10.0

[1] Excludes times of pumping with clear water and times when pumps were shut down for repairs to pipeline.

[2] The volume occupied by 1 ton of crushed refuse is approximately 1 cubic yard.

[3] Pressure gages at the top of injection boreholes were compound gages, measuring vacuum (pressures below atmospheric) in inches of mercury and pressures above atmospheric in psi. Gage readings corresponding to the negative psi values above are as follows:

psig	in Hg
-3.9	22
-4.9	20
-5.4	19
-6.9	16
-7.4	15
-12.3	5

One objective of the demonstration project was to test the applicability of the pumped slurry method to mine cavities that are not inundated. Recent studies of backfilling mine cavities, carried out for the Bureau of Mines by the Bureau of Reclamation in its hydraulic laboratory, included two tests in which slurry was injected into dry cavities. The results of the model studies are published in the appendix to this report. The doughnut-like mound, radial to the injection point that formed in water-filled mine cavities was not produced in dry cavities. The fill material was transported and deposited according to principles governing open channel sediment transport. In the test in which the mine model was level, the fill material formed a gently sloping cone around the injection point (slope angle about 3° below horizontal). The steepness of the slope depends on the tractive force required to transport the fill material, which is a function of the particle size. In another test, the mine model was inclined to a 15° slope with the lower part submerged and the injection point above water level. In this test, the fill material flowed laterally and downslope along the rooms of the mine model into the submerged area below.

In the Green Ridge demonstration project, the underground movement of the fill material was consistent apparently with recognized principles of open channel transport of solids. In several of the monitor holes that were filled from injection points above pool level, the process of filling to mine roof level occurred gradually over a period of several days.

In the Green Ridge project area, the dip of the strata is about 4° in the part that was backfilled by injection through boreholes B and C. Evidence from monitor holes indicated that the slurry moved downgrade into the submerged area, and also laterally to areas above pool level, as in the model studies. In addition, however, fill material moved upgrade from C to the location of monitor hole 1 (335 feet distant and 24 feet higher in altitude) and from B to monitor hole 37 (300 feet distant and 18 feet higher in altitude). Apparently the downslope direction of a passageway became blocked by early fill deposition so that the path of least resistance was temporarily upgrade.

Injection into Inundated Area

Boreholes A and D, and the areas filled from these injection points, intercepted the mine below water level. Injection borehole A was used during two periods for a total time of eighteen 8-hour shifts in which 41,471 tons of solids was injected. Fill material injected through A moved downgrade parallel to the long axis of rooms in the mine and laterally to fill in the northern and northwestern parts of the project area in the New County bed.

Through borehole D, 31,716 tons of solids was injected during a single period of thirteen 8-hour shifts. The fill material moved downgrade and laterally to fill in the western part of the project area.

Project Evaluation

The Green Ridge hydraulic backfill demonstration project can be evaluated in terms of the completeness of fill emplacement in the mine workings, the feasibility of the operation, its costs, and benefits.

Completeness of Fill Emplacement

Backfill material was distributed in the mine workings throughout an area of approximately 30 acres according to observations from monitor holes (fig. 10). The presence of fill material in monitor holes at levels higher than the mine roof indicated that filling of the mine workings within the 30-acre area was essentially complete from floor to roof. Some fill material presumably extends beyond the limits of completely filled voids (see shaded area on figure 10) conforming to the angle of repose of the emplaced fill material. The estimated extent of the backfilled area is 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 on "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 results 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 flush (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 new pumped-slurry technique (actually another form of blind flushing) provides the most complete filling of mine workings that are flooded or otherwise inaccessible.

Feasibility of Operation

The pumped-slurry hydraulic backfill operations at the demonstration site in the Green Ridge section of Scranton proceeded as planned without difficulties other than minor repairs of equipment and the loss of 9 days while waiting for replacement of a water pump. The method proved successful under the conditions of the demonstration. The conditions were as follows:

Depth of mine workings to be filled, between 100 and 200 feet below the surface;

Mine workings submerged or not more than 11 feet above water level;

Dip angle of strata less than 5°;

Mine workings relatively unobstructed by caving of overlying strata;

Backfilled coalbeds separated by 50 feet of rock;

Average depth of alluvium, 30 feet;

Minimum rock cover, 78 feet;

Particle size of fill material, minus 1-1/4-inch and minus 1/2-inch;

Specific gravity of particles, 2.2;

Bulk density (dry), 74 pcf; and

Water available in large quantities.

Further use of the new 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 depths, material injected under pressure may rise to the surface rather than being confined to the mine level, especially in areas where overburden 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 critical part of the injection process by the pumped-slurry method is determining the point at which to terminate the use of an injection borehole. To achieve the maximum area of backfilling from a single borehole, injection must be continued against back pressure. On the other hand, injection against increasing back pressure must be stopped before reduction in slurry velocity causes blockage of the pipeline system by solids settling out of suspension or high pressure induces equipment failure.

Because of its cost, the backfilling of abandoned mines is carried out in built-up areas where property values warrant protection, and the impact on the community may be a significant consideration in determining feasibility. In most hydraulic backfilling projects, the principal causes of disturbance are the drilling of boreholes in city streets and the actual injection operations.

For each injection borehole used in the pumped-slurry demonstration project, several hundred boreholes have been used in gravity blind flushing projects to inject a comparable quantity of fill material. Controlled flushing projects have used about four times as many injection holes as the pumped slurry project. Additional boreholes were drilled at the Green Ridge project to monitor the progress of the backfilling. It is likely that monitoring boreholes will become an integral part of the pumped-slurry technique, just as drilling of access, ventilation, and supply boreholes is usually part of the controlled flushing method. Considering borehole requirements for all purposes, drilling requirements for the pumped-slurry process are expected to be significantly less than those for the gravity blind flushing method.

The actual injection operations in both controlled and blind flushing methods require continuous truck traffic through city streets to bring fill material to injection boreholes. At the injection borehole, men are working in the street to direct the solids and water down the borehole. In the new pumped-slurry method, street disturbance within the project area is limited to installation of the distribution pipeline. In the Green Ridge demonstration project, slurry moved unseen through the buried pipeline system into injection boreholes over a period of 8 months. The only work required on the surface during most of the injection period was cleaning the street on one occasion when slurry was ejected from a monitor borehole as the cap was removed, and on three occasions when small quantities of slurry came to the surface around the outside of monitor hole casings. Toward the end of the project, repairs of leaks in the pipeline required the presence of workmen in the streets for brief periods.

Costs

The cost of the hydraulic backfill demonstration project in the Green Ridge section of Scranton, in which 451,117 cubic yards of crushed refuse was injected, was $2,165,915--a unit cost of $4.80 per cubic yard.

Cost comparisons of the different methods of backfilling are difficult to make. The total number of projects is small, they span a period of rising costs, and 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 of solids injected ranged from $1.84 to $2.38. For two blind flushing projects in 1965 and 1967, the costs were $2.46 per cubic yard--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 cubic yard ranged from $3.64 to $6.76.

Reduction in costs of the pumped-slurry technique are anticipated as additional experience is gained in applying the method. In the demonstration project, the injection operation was performed under a negotiated contract because of uncertainties inherent in the first large-scale trial. Lower costs are foreseen under bid contracts. Reuse of equipment should reduce future costs. Furthermore, the demonstration project included more instrumentation and monitoring than may be required for a routine operation.

Included in the total cost of the demonstration project were the following items:

Subsurface exploration.	Boreholes (drilling, casing, capping, logs). Sonar survey--checked with available mine maps.
Site preparation...	Rental. Grading. Drainage borehole. Assembling equipment. Lighting. Electric connections. Office trailers, heated and cooled as necessary. Sanitary facilities.
Water..............	Wells (drilling and casing). Pumps (submersible, stainless steel). Pipe and fittings (valves, rate recorder, and totalizer).
Backfill material..	Crushing (sizing screens, heated). Transport refuse to mixing plant (conveyors and front-end loaders).
Pipeline...........	Pipe and fittings (welding). Installation and burial. Pressure gages. Repair leaks. Replace valves.
Injection boreholes	Locate open space at mine level (probe holes; sonar). Drilling, casing, and grouting.
Mixing plant.......	Feeder-hopper, conveyor, belt scales. Mixing tank. Slurry pump, power assembly, muffler.
Site restoration...	Disassemble mixing plant. Remove pipeline, repair street pavement. Plug injection and exploratory boreholes. Remove pumps from wells, plug wells. Trim hazardous slopes on refuse banks.
Miscellaneous......	Power,[6] fuel,[7] and supplies for operation and maintenance of equipment.
Insurance..........	
Subsidence control records.	Levels on a network of surface altitude benchmarks overlying the backfilled area, taken before and after the injection period.

[6]Electric power used, 999,100 kW.
[7]Fuel used, 63,722 gallons.

The cost of the backfill material was about 45 percent of the total cost, though the mine refuse was provided without cost and its location at the project site eliminated transportation cost. The cost of crushing to the minus 1/4-inch size originally specified was estimated at $2.38 per ton. With the change to minus 1/2-inch size, the estimated crushing cost was $2.05 per ton. Experimentation with coarser material will provide a basis for evaluating cost savings in crushing in relation to the optimum particle size for maximum dispersal of the backfill in the underground channels.

The use of coarser material may prove successful because anthracite mine refuse characteristically yields a large proportion of fine sizes from any chrushing operation. Analysis of the material that was crushed to minus 1/2-inch size indicated that about 80 percent was of 1/4-inch size or smaller (fig. 21). The gradation of particle size may be more significant than the maximum particle size, depending on the degree of segregation that takes place during deposition of the fill material in the mine. Where segregation is not significant, density of the fill deposit is greater in a well-graded mixture, and its compactability would be less. On the other hand, any fine material that may become segregated in the backfilling process would be transported greater distances from the injection point and increase the areal extent of backfill but not necessarily fill the voids.

The sonar surveys, which accounted for 0.5 percent of the total cost, provided little new information on the condition of the mines at the demonstration project site because the local mine maps furnished accurate information and caving had not modified the conditions significantly. The value of the sonar surveys to the demonstration project was to test the applicability of the sonar method to mine workings. In areas where accurate mine maps are not available or where caving has been extensive, sonar surveys may provide essential information. The use of other sensing methods should also be explored for this purpose.

Costs as well as feasibility of the operation are affected by the relation of equipment capacity to the magnitude of the job. In the demonstration project, the pipeline system, which was adequate for injection of the quantity of slurry required by the original contract, developed leaks after the contract was modified to inject additional material. Costs were incurred both in repairs and in time lost by shutting down the injection operation until repairs were completed. The relative costs of patching, replacing pipe, rotating it to distribute wear (which involves decision as to whether the pipeline should be buried), or of using more costly pipe can be evaluated in relation to the size of the job as more experience is gained.

Benefits

In addition to the primary benefit of subsidence control in the backfill area, the corollary benefit to the larger community is the removal of banks of accumulated mine waste that have been stored on the surface for many years. Figure 40 is an aerial view of the slurry injection plant site and the site of the former Eureka bank, taken at the end of the demonstration project. The approximately 100,000 cubic yards remaining in the Eureka bank will be used,

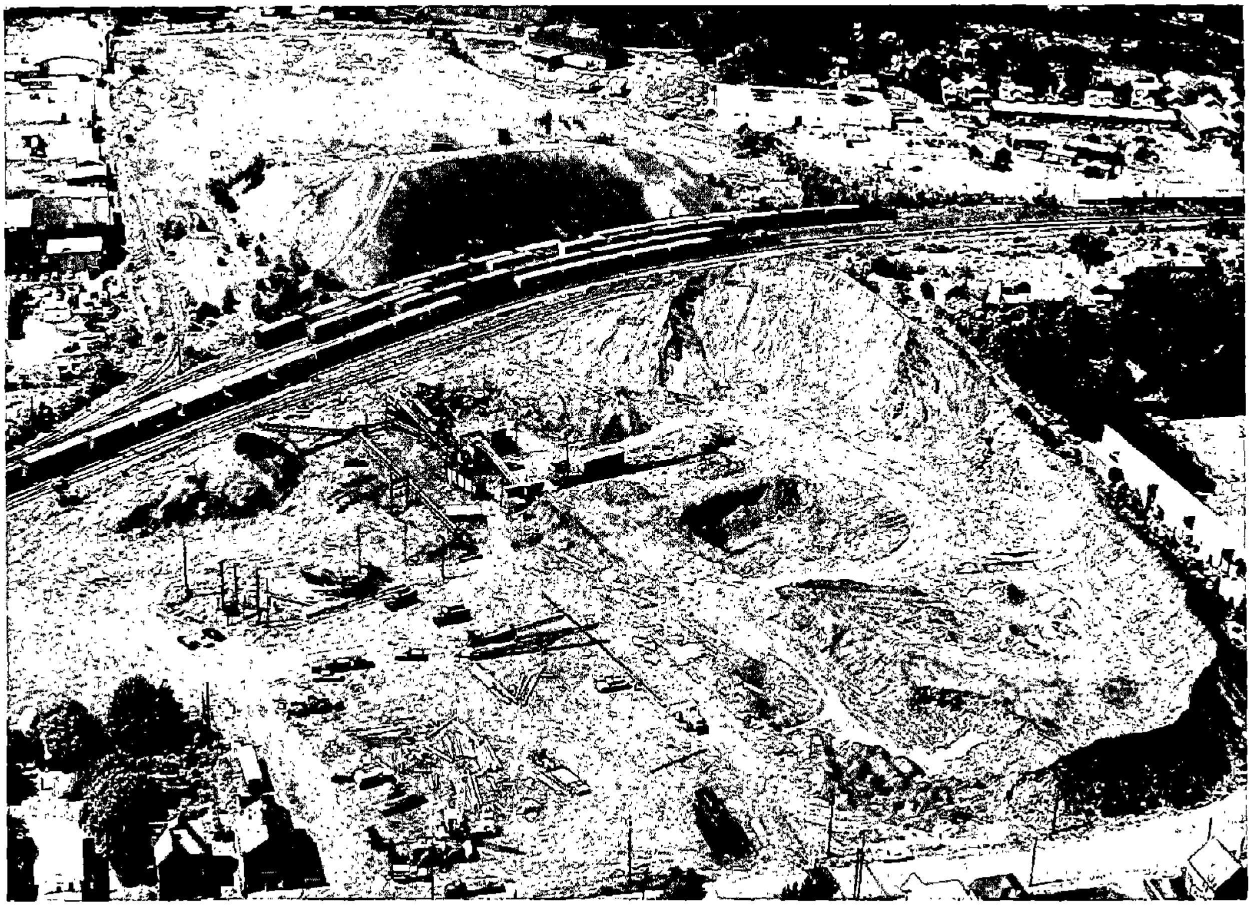

FIGURE 40. - View of plant site at end of demonstration project. After injection of more than 450,000 cubic yards of crushed refuse, most of the Eureka bank (right foreground) has been removed except for northern portion and a protective shell adjacent to neighboring houses. The former extent of the Eureka bank is shown in figure 8.

together with material from the Von Storch bank across the railroad tracks, in another hydraulic backfilling project.

Waste banks are unsightly and they pose environmental hazards. Fine material picked up by wind creates a nuisance to nearby residents. Refuse bank fires are difficult and costly to extinguish and emit poisonous gases and particulate matter to the atmosphere. Waste banks may also contribute acid water, which adds to the pollution of surface streams. Conversely, the placement of mine refuse in underground mine voids is expected to reduce the rate of air and water circulation at mine level and aid in controlling acid water formation.

Many waste banks are in urban areas, and their removal will provide needed space for future development. The subsidence potential of such restored land areas will need to be evaluated in planning for new uses of the property.

A potential benefit of developing the pumped-slurry method of hydraulic backfilling may prove to be its applicability to the control of fires in underground coalbeds.

Additional research may disclose that the process, under certain conditions, can be incorporated in the mining cycle to avoid disturbing the surface where minerals are being extracted from beneath urban areas.

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APPENDIX

REC-ERC-73-19

HYDRAULIC MODEL STUDIES FOR BACKFILLING MINE CAVITIES

by

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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
Rogers C. B. Morton, Secretary
Gilbert G. Stamm, Commissioner

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CONTENTS

PURPOSE

The purpose of this study was to obtain qualitative and quantitative information on the deposit pattern of fine sand when used for backfill material and injected into a typical coal mine cavity. Backfilling of mine cavities with sand and waste material is a method used to reduce land subsidence. The study was performed for the Bureau of Mines.

SUMMARY AND CONCLUSIONS

A model of an idealized coal mine was constructed and operated to determine the characteristics of backfilling a mine by pumping a sand slurry into the cavities. The model simulated the hydraulic action in the coal mine under the city of Rock Springs, Wyomimg, where subsidence due to coal mine cavities has been experienced. Fine, uniform blow sand with a median size of approximately 0.14 mm from the Rock Springs area was used in the model studies. Similar materials will be used in future backfilling operations for mines in the Rock Springs area.

Eighteen tests were made simulating the pumping of fine sand into a mine cavity with the following conditions:

1. Level floor with cavity submerged
2. Level floor with cavity dry
3. Sloping floor with injection pipe exit below the water surface
4. Sloping floor with injection pipe exit above the water surface
5. Selected corridors between pillars partially blocked and totally blocked
6. Solid walls on one and two adjacent sides of a rectangular section of pillars surrounding the injection hole

The test conditions are summarized in Table 1.

The approximate bearing strengths of the backfill material were determined by soils mechanics tests.

Data from the 18 tests lead to the following conclusions, which may be modified as additional information is obtained and analyzed:

1. Initial deposition of fine sand backfill material pumped vertically into a level submerged mine cavity takes the shape of a truncated broad-based cone which builds up to the roof of the unobstructed mine cavity, Figure 2.

2. The general pattern of the cone-shaped deposit in a level submerged mine backfill operation is not dependent on slurry concentration nor on injection pipe velocity. However, low pipe velocities result in a smaller radius of the initial deposit than for higher injection velocities.

3. Segregation of graded backfill material occurs in the central cavity when backfilling a level submerged mine. The larger particles deposit near the injection pipe and the finer particles deposit farther from the injection pipe in a radial direction. The particles at the bottom of the deposit ring are larger than the particles at the top of the deposit ring.

4. Fine sand backfill material injected into a submerged mine cavity having a 5^{o} slope deposits in an initial broad-cone pattern almost identical to the deposition pattern in a level submerged mine cavity (see Conclusion 1 above). A solid wall located a short distance from the injection pipe does not prevent slurry from flowing nor backfill from depositing in that direction.

5. Fine sand backfill material pumped into a submerged mine cavity having a 15^{o} slope will deposit in an initial cone around the injection pipe. Backfill material will then be transported and deposited downslope and laterally along breakout paths. As the back pressure on the injection area builds up from deposits, fine material is transported along breakout paths and deposited in directions of least resistance from the injection pipe.

6. All tests showed that fine sand backfill material will be transported past partial blocks in corridors as deposits build up. Fine sand backfill material will be transported into slack water areas and deposited by flow circulation.

7. Backfill material pumped into a dry mine cavity will develop a deposit with a surface slope of the deposited material which is dependent on a critical tractive force* required to move the backfill material at shallow flow depths in an open channel. For the fine sand material obtained from Rock Springs, the slope of the deposit surface was 0.05 to 0.06 in a dry mine cavity.

8. When the top of the deposit cone reaches the mine ceiling in a submerged mine cavity, back pressure builds up until a breakout channel forms between the mine ceiling and deposited material. Backfill material is then transported along the

*Tractive force $T_D = \gamma DS$ where γ = specific weight of water, D = depth of water flowing over the deposit, and S = slope of flowing water surface.

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channel and deposited in the pool at the end of the channel until back pressure builds up and forms a new breakout channel.

9. The advancing front of deposited backfill material in a submerged mine cavity takes the slope of the submerged angle of repose of the backfill material. For the fine Rock Springs sand deposited under water the angle of repose is 30^0 or a slope of 0.577.

APPLICATION

The results described in this report can be used in planning and executing an effective method of hydraulic backfilling of mine cavities.

INTRODUCTION

Backfilling mine cavities with sand or waste material is a method that has been used to reduce subsidence. Much information is needed to determine the pattern of backfilling deposition for the various fill materials and mine cavity conditions.

The Bureau of Mines requested the Bureau of Reclamation to perform hydraulic model studies to determine the pattern of deposition for various typical mine conditions.

THE MODEL

Model Box—Slurry Sump

A 15-foot-square box 2.5 feet deep was constructed from wood frame and 3/4-inch plywood. The box was made watertight by sealing the plywood joints with rubber strips and sealing compound. A flap gate hinged on the floor of the box was constructed in one corner. The flap gate position could be adjusted to hold the water surface in the box at desired mine water levels. The floor was sloped slightly to the flap gate corner for completely draining the box in a very short time.

Water flowing through the flap gate dropped into a metal slurry sump after passing through a 4-foot-long by 2.5-foot-wide sluice channel. The sump was 8 feet long by 2 feet wide by 3.5 feet deep mounted below the floor in a laboratory water-supply channel. Sand backfill material was washed into the slurry sump after being placed on the sluice channel floor. A propeller mixer was mounted vertically to maintain the backfill material in suspension during tests so it would be picked up as a slurry by the 2-1/2-inch Kimball-Krogh sand pump. Power was provided to the Model 100 pump with a 5-horsepower electric motor.

Piping and Measuring System

A 1-inch standard pipe was used for slurry injection so a velocity similar to the velocity used in the field injection system at Rock Springs, 16 to 18 feet per second, could be obtained in the model pipe. Previous backfilling operation pumped slurry through a 13-3/8-inch-inside-diameter pipe at a maximum rate of 7,800 gpm to give a pipe velocity of 17.7 feet per second. Because transport velocity of backfill material is the most important parameter when pumping and injecting through a pipeline, the model duplicated prototype transport velocities in the pipeline and in the mine cavity.

Water and slurry discharges were measured using Venturi meters and water and mercury manometers. For Tests 1 through 14, a 3- by 1.45-inch BIF Venturi meter was used. For Tests 16 through 18, a second Venturi meter, 2- by 1-inch throat, was installed in the discharge line.

Model Scales—Mine Pillars

Before designing the model, several maps of actual mine layouts were observed on microfilm in the Bureau of Mines Denver Office. The pillar, cavity, and corridor dimensions varied from mine to mine. From discussions between laboratory engineers and Bureau of Mines engineers, it was decided that a symmetrical layout of pillars should be used in the model with each pillar having dimensions of 40 feet long, 10 feet wide, and representing a mine cavity height of 6 feet.

The mine should have approximately 60 percent volume extraction of coal leaving a pillar volume of approximately 40 percent. The horizontal layout for a symmetrical pattern of pillars with above requirements was arranged to give approximately 8.5 feet of corridor space between ends of pillars and 11 feet between sides of pillars.

Distorted model scales—Equal transport velocities.—To obtain a horizontal velocity in the model mine cavity equal to the velocity in the prototype, the height of the model mine cavity required was 0.75 foot. For the above conditions and maintaining a water velocity in the injection pipe and the horizontal water velocity in the mine cavity the same as in a Rock Springs prototype operation, the model geometric scales were: horizontal scale—$1^m:24^p$ and vertical scale—$1^m:8^p$. This gave a vertical distortion of 1:3. The important considerations was that water transport velocities were the same in the model and the generalized Rock Springs, Wyoming, prototype mine backfill operation.

Undistorted model scales—Unequal transport velocities.—The tests with a sloping mine cavity were made with both geometrical horizontal and vertical scales undistorted at $1^m:24^p$. Tests were made this way to minimize the problems with different horizontal and vertical scales for a tilted geometric shape. Test 17 was made with a level mine cavity and with equal horizontal and vertical scales of $1^m:24^p$. This was done to compare backfill deposit pattern for the earlier level floor tests using a distorted scale with level floor tests and an undistorted scale. Deposit patterns were similar even though water velocity scales were different.

For all tests, prototype backfill material (a natural fine sand deposit obtained from Rock Springs, Wyoming) was used in the model. A distortion existed between prototype and model backfill sand based on the geometrical scale but water velocities and therefore transport and deposit characteristics were the same for a typical prototype (Rock Springs mine) and the model. Deposit patterns depicted by the model should very closely represent backfill deposit patterns in the prototype.

Model and Prototype Backfill Material

At the outset, a decision was made that the most important characteristic to be studied in the model was the transport and deposition of the mine backfill material. Therefore, a prototype fine sand backfill material obtained from Rock Springs, Wyoming, was selected as the model backfill material and the model was scaled for using the fine prototype sand. To do this, prototype pipeline velocities were used in the model pipeline. Prototype mine cavity velocities were used in the model considering radial distances from the centerline of the vertical injection pipe and horizontal distances scaled according to the horizontal scale of $1^m:24^p$.

The median size of the sand material used in the first backfilling operations at Rock Springs, Wyoming, was 0.14 mm. Approximately 6 yards of the fine sand were trucked to Denver. Portions of this sand were used in all of the model tests. In the Appendix, a size analysis is shown on the graph, Figure 1, of the soils test to determine bearing capacity.

THE INVESTIGATION

Tests With Level Floor

Eleven tests were conducted with backfill material pumped into a mine cavity having a level floor. All tests were conducted with the cavity in a submerged condition, except Test 10 which simulated a dry cavity.

Preliminary Tests Without Pillars

Test 1 was made to determine the adequacy of the slurry tank, propeller mixer, sand feed, sand pump, and piping system. Only a small amount of fine sand material (about 1 cubic yard) was fed into the system during the test. About 10 cubic feet of fine sand material reached the model box, Figure 1. The remainder of the sand settled and remained in the slurry tank during the test.

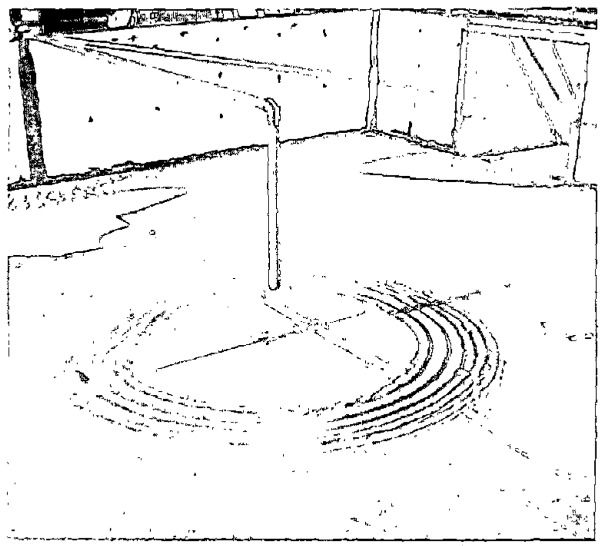


Figure 1. Test 1. Pattern of deposit resulting from initial operation of the slurry mixing and pumping system. Photo P801-D-73839

Test 2 illustrated the deposition pattern for backfill material pumped into a deep submerged mine cavity with no pillars, Figure 2. Slurry was fed at a concentration of approximately 12 percent, by weight. Velocity in the injection pipe was approximately 9 feet per second. The deposit was in a cone shape with a depression in the top of the cone caused by velocity and turbulence of the jet. The angle of repose of the material deposited under water was about 30^{o}. The maximum height of the deposited material was about 3 inches below the water surface when the test was stopped.

Test 3 was similar to Test 2 except a higher velocity of 16 feet per second was maintained in the injection pipe. Figure 3 shows the pattern of deposition in a simulated submerged cavity for this condition. Velocity from the submerged pipe was high enough to keep the floor free of sand material. A strip of sealing tape on the floor caused nonuniform velocity distribution and consequent nonuniform backfill material distribution.

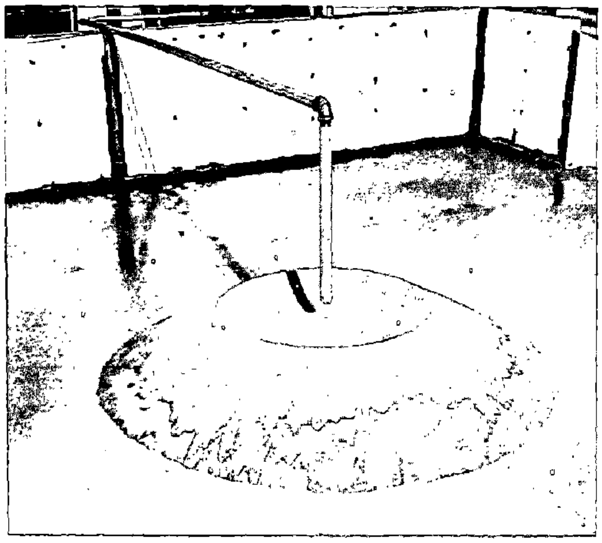


Figure 2. Test 2. Pattern of deposit with a slurry concentration of 12 percent, by weight, and velocity of approximately 9 feet per second in the injection pipe. Photo P801-D-73840

Figure 3. Test 3. Deposit pattern in a submerged mine cavity for backfill material pumped at a concentration of 12 percent, by weight, and 16 feet per second in the injection pipe. Photo P801-D-73841

Test 4 illustrated the deposition pattern as the backfill material deposit reached the water surface in a partially submerged cavity. A rather flat surface (slope 4^{o} to 5^{o}) or shear plane developed at the top of the cone, Figure 4. The depth of flow over this plane was very shallow. The material was transported over the flat, sloping

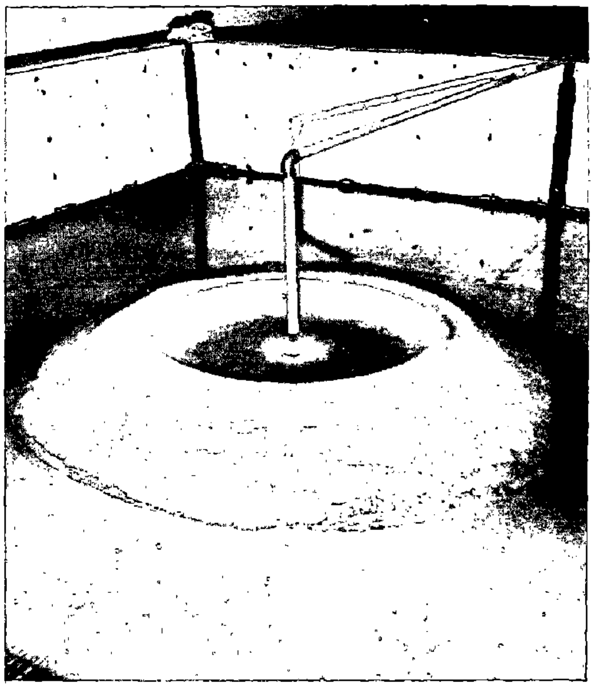


Figure 4. Test 4. Deposit pattern showing shear surface at the top of deposited cone. Water surface was held at the downslope elevation of the shear plane surface. Photo P801-D-73842

plane according to the tractive force of the water flowing over the plane and the size of fill material, and deposited at an angle of repose on the sides of the cone. Again a typical flow velocity of approximately 16 feet per second was used in the injection pipe.

The first four tests were operated at prototype injection velocities to observe the deposit pattern and determine how the fill material acted under the hydraulic conditions imposed. Thus, radial velocities in the mine cavities for the prototype and the model were similar. Transport velocity is the most important parameter when considering transport of backfill material.

Tests With Pillars, Except for Test 7

Test 5 simulated a submerged mine cavity with a roof and pillars confining the flow in the cavity. The fill material deposit in the cavities between the pillars is shown in Figure 5 after the mine roof was removed. Velocity in the injection pipe located at the geometrical center of the pillar arrangement was approximately 16 feet per second and sand concentration was 12 percent, by weight. Pillars 40 feet long by 10 feet wide by 6 feet high were constructed in the model at a horizontal scale of $1^{m}:24^{p}$ and a vertical scale of $1^{m}:8^{p}$ giving a vertical distortion of 1:3.

Figure 5. Test 5. Fine sand backfill material deposits in cavities between the mine pillars. Photo P801-D-73843

This distortion resulted in radial velocities nearly the same as those in the field operation at Rock Springs, Wyoming. The pillars were arranged in a symmetrical pattern to give approximately 60 percent cavity and 40 percent solid pillars in the mine. The test was run until the deposited material nearly reached the ceiling. Back pressure then built up, causing fine sand to break out of the initial ring and deposit outside the pillar area.

Test 6 was similar to Test 5, except four additional pillars (28 total) were installed in four rows, Figure 6. Sand was added to give approximately 17 percent concentration, by weight, and the deposit was similar to Test 5—Compare Figure 7 with Figure 5. Figure 8 shows contours of deposited backfill material of Test 6 in prototype dimensions.

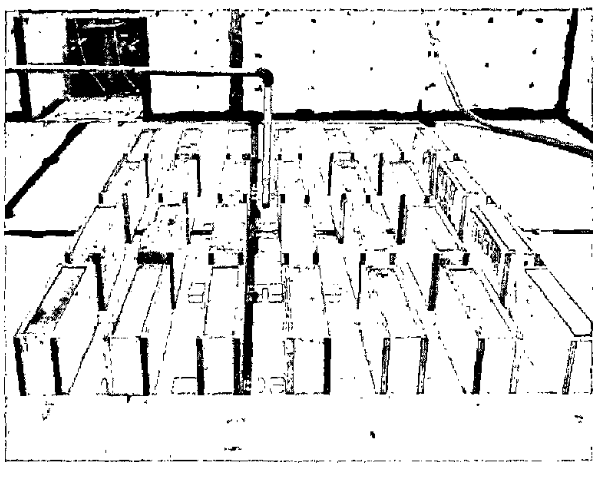


Figure 6. Test 6. Seven rows of four pillars each were arranged in a symmetrical pattern to give 60 percent cavity and 40 percent solid pillars in the mine before Test 6. Photo P801-D-73844

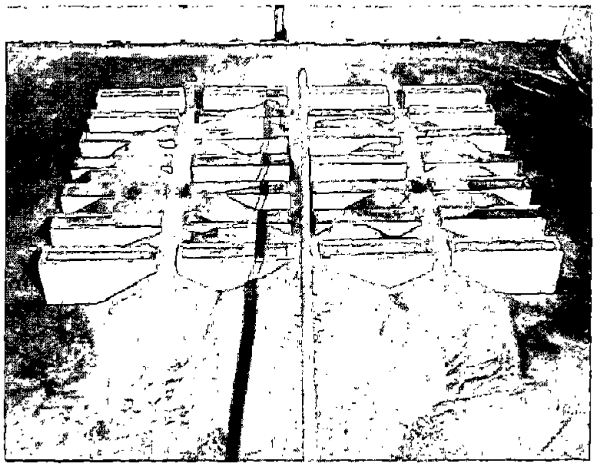
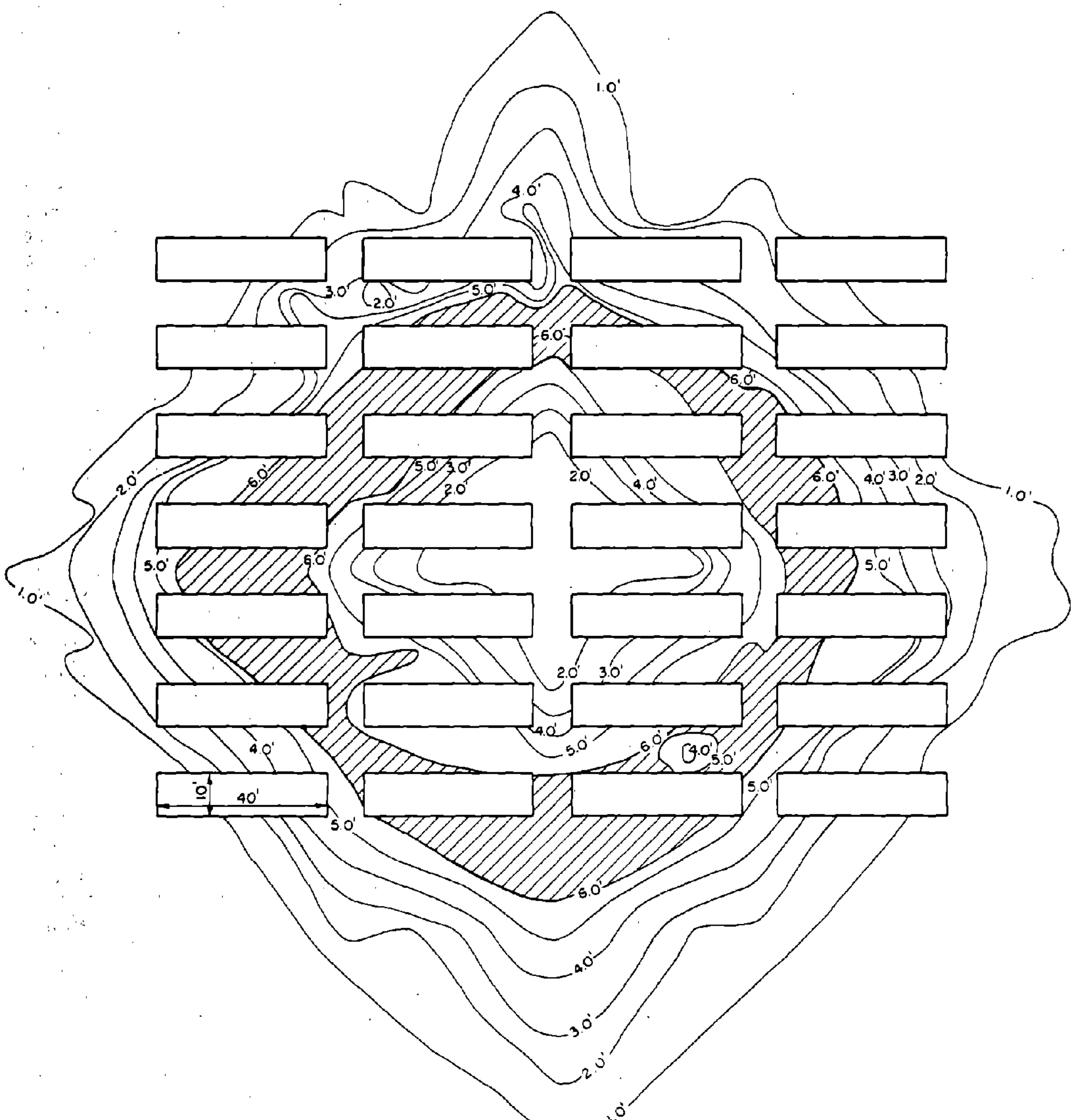


Figure 7. Test 6. Deposit pattern after injection of 16 percent backfill material at a pipe injection velocity of 15 feet per second. Photo P801-D-73845

For Test 7, the pillars were removed and the mine roof was placed at a simulated field position 6 feet above the floor. The test was made with the cavity in a submerged condition. A comparison of the deposited fill material for Test 7 with the previous Tests 1 through 4 in which a confining roof was not in place, shows a different pattern on the outside edge of the deposited ring of material, Figure 9. The material in Test 7 was deposited in a scalloped pattern, compared to a smooth, circular pattern in Tests 1 through 4. In Test 7, fill material deposited until the flow area between the top of the sand deposit and the ceiling was nearly closed off. A back pressure then built up, a channel broke out along the top of the sand deposit, and backfill material was transported in this channel until enough material was deposited to form a delta and closed the channel off. The flow then broke out in another channel depositing another delta. This procedure continued, forming a scalloped pattern around the outside edge of the doughnut-shaped ring of backfill material.

Tests 8 and 9 were made to show how partially or totally blocked openings in the mine corridors would affect the deposited pattern of backfill material. Figures 10, 11, 12, and 13 show where partially and fully blocked openings are located and how the fine sand backfill material flows to fill the cavities. Even a small flow over a considerable time period will transport fine sand around corners and into cavities that seem to be blocked. As a general rule, if water will flow into an area, fine backfill material transported by the water will be carried in to fill cavities or around corners. Velocity in the injection pipe varied from approximately 14 to 4 feet per second for Test 8 and



TEST 6

CONTOURS OF BACKFILL DEPOSIT
AT END OF TEST.
PILLARS – 40% OF MINE VOLUME
CAVITY – 60% OF MINE VOLUME
MINE CAVITY LEVEL AND SUBMERGED.

BACKFILL MATERIAL FILLED TO ROOF LEVEL.

Figure 8

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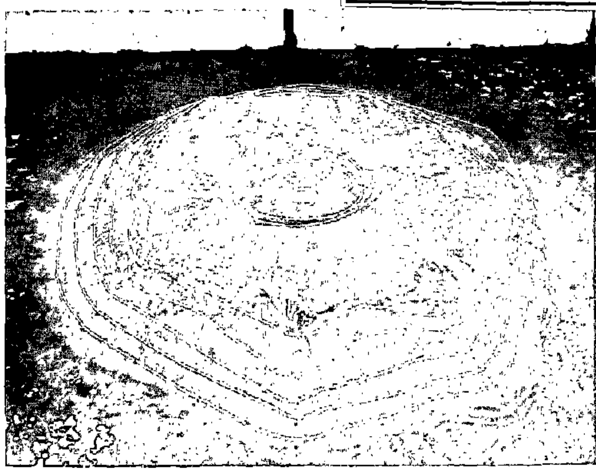


Figure 9. Test 7. Contours simulate 1-foot intervals, 0-6 feet, in prototype mine cavity after injection at 14.5 feet per second with slurry concentration of 16 percent. Photo P801-D-73846

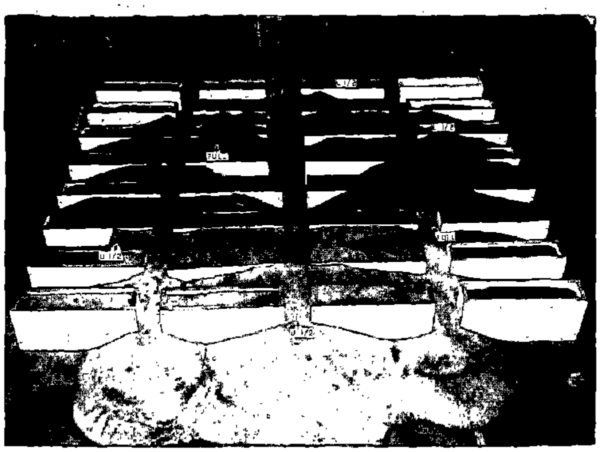


Figure 10. Test 8. Deposition pattern for corridors partially blocked and fully blocked at points indicated. Slurry concentration was about 35 percent, by weight. Corridor block designations are: U 1/2 indicates upper one-half of corridor is blocked; L 1/2 indicates lower one-half of corridor is blocked; full indicates full corridor is blocked. P801-D-73847

was steady at 14 feet per second for Test 9. Sand concentration for Test 8 was approximately 30 to 35 percent, by weight, and for Test 9 approximately 25 percent, by weight.

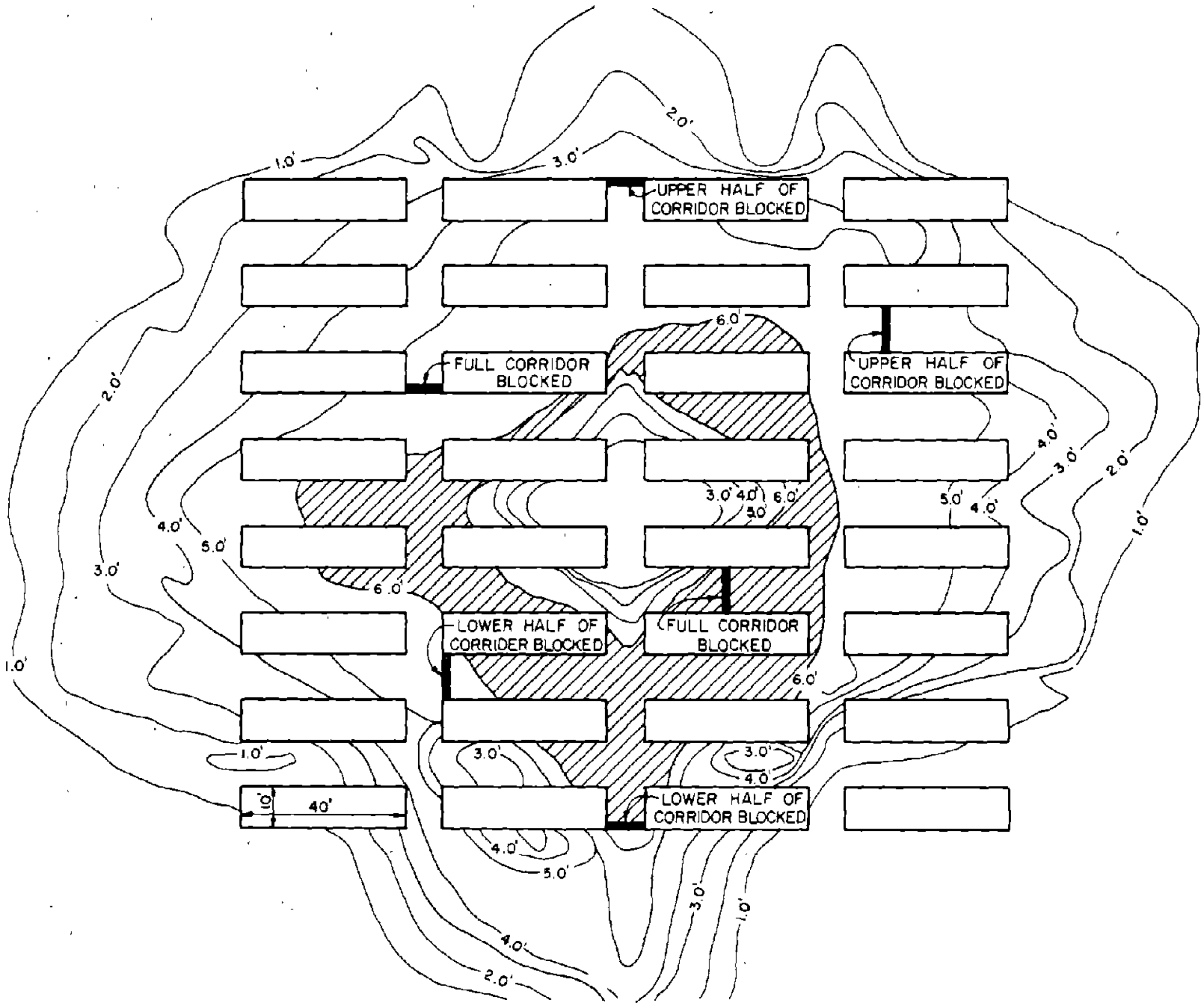
In Test 10 a dry mine cavity was simulated with the sides and an end of one corridor blocked before the test, Figure 14. No roof was used and the injection pipe velocity varied from approximately 15 to 8 feet per second with a 20 percent slurry concentration. A slight amount of back pressure caused a reduction of discharge and velocity in the injection pipe; however, no adjustment to the discharge was made after the test was started. The end block allowed a small amount of water to flow through the corridor, Figure 15. The blocked corridor filled with backfill material to about the same depth as the corridors outside of the blocked area. In the dry cavity backfill, material is transported and deposited according to open channel sediment transport laws. The bed slope from the top of the cone around the injection pipe to the outside of the cone was 0.05 in the longitudinal corridor direction and 0.06 in the cross-corridor direction, Figure 16. In a dry cavity a shallow depth of water transports backfill material and the steepness of the resulting bed slope depends on the tractive force required to transport the size of backfill material. Figure 17 shows prototype contours of deposited backfill material at the end of Test 10.

Sloping floor Tests 11, 12, and 13 are discussed in the following sections.

For Test 14 the height of the model pillars was reduced from 0.75 to 0.25 foot to represent the 6-foot-high prototype pillars without vertical distortion. This test was conducted to compare the pattern of deposition of backfill material in distorted and undistorted models. Test 14 had a simulated wall on two of the four sides. These walls seemed to have very little effect on the initial deposit pattern of backfill material as compared to Test 5 having no walls on the sides. The general distribution of backfill material in Test 5 (distorted vertical dimensions test) is very similar to Test 14 (undistorted vertical dimensions tests). Velocity in the injection pipe, Test 5, was approximately 16 feet per second with a 12 percent concentration and for Test 14 about 10 feet per second with a 10 percent concentration. Proportionately for the depth of cavity, more sand was pumped in Test 14 than in Test 5. Figures 18 and 19 show deposit pattern and prototype contours at the end of Test 14.

Test 15 was conducted with water only to check the water calibration of the Venturi meters.

For Test 16, the vertical height of the model pillars was 0.25 foot (no vertical distortion), and the arrangement of pillars was the same as for previous tests. Distribution of backfill material at 9 percent concentration was very similar to the pattern of deposit in the tests with higher pillars, Figure 20. The velocity of slurry in the injection pipe was approximately 16.5 feet per second. The velocity and turbulence were high enough to clear the floor area below the end of the injection pipe.



TEST 8

CONTOURS OF BACKFILL DEPOSIT AT END OF TEST.
PILLARS-40% OF MINE VOLUME.
CAVITY-60% OF MINE VOLUME.
MINE CAVITY LEVEL AND SUBMERGED.

(hatched) BACKFILL MATERIAL FILLED TO ROOF LEVEL.

Figure 11

Figure 12. Test 9. Blocks to reduce flow areas by one-half in the corridors were placed at several locations (see arrows). Slurry concentration was approximately 25 percent with velocity in the injection pipe approximately 14 feet per second. Photo P801-D-73848

Tests With Sloping Floor and Pillars

Tests 11, 12, and 13 were made with the mine cavity floor on a 15° slope. The water surface in the mine area was lower than the injection pipe exit for Test 11. Backfill material was pumped into the mine cavity on a dry floor and the slurry flowed laterally and downslope along the corridors into the ponded water table below. Slurry concentration was 24 percent and the injection pipe velocity was about 3 feet per second. Horizontal and vertical scales were 1:24. The deposit pattern is shown in Figure 21.

Test 12 was similar to Test 11, except the water table was above the injection pipe exit in the 15° sloping mine cavity. The backfill material was injected at a concentration of about 35 percent with a velocity of approximately 8.5 feet per second. Material flowed radially from the injection pipe, filling the cavity downslope and also upslope to the water surface. Figure 22 shows the pattern of deposition.

Test 13 was a duplicate of Test 12. Comparing pictures of Figures 22 and 23 shows that the deposit pattern for these two tests was very similar. Figure 24 shows the prototype contours of deposited material at the end of Test 13.

The mine cavity for Test 17 had a solid wall at the downslope end of the corridors. The cavity was submerged, the floor was on a 5° slope, and the velocity of the 8 percent slurry in the injection pipe was about 10 feet per second. The deposition pattern for Test 17 was very similar to the deposition patterns for Tests 5 and 14 made with a level floor and submerged condition, Figure 25. For the initial ring of backfill material that deposited up to the ceiling of the mine cavity, the deposit pattern was very nearly symmetrical. The first breakout and channelization occurred in an upslope direction. The distance from the injection pipe to the initial ring deposit was slightly smaller in an upslope direction than in a downslope direction, allowing the first breakout in an upslope direction. The gravity component, in the 5° slope direction, was very likely a reason for the slightly less deposit downslope than in the upslope direction. Figure 26 shows the prototype contours of deposited backfill material at the end of Test 17.

Test 18 was made as a duplicate of Test 17 except Test 18 was continued longer (32 minutes as compared to 18 minutes for Test 17 model time). Figures 27 and 28 when compared with Figures 25 and 26 show that there is a more extensive deposit at the end of Test 18 and the deposit to the roof has a wider band.

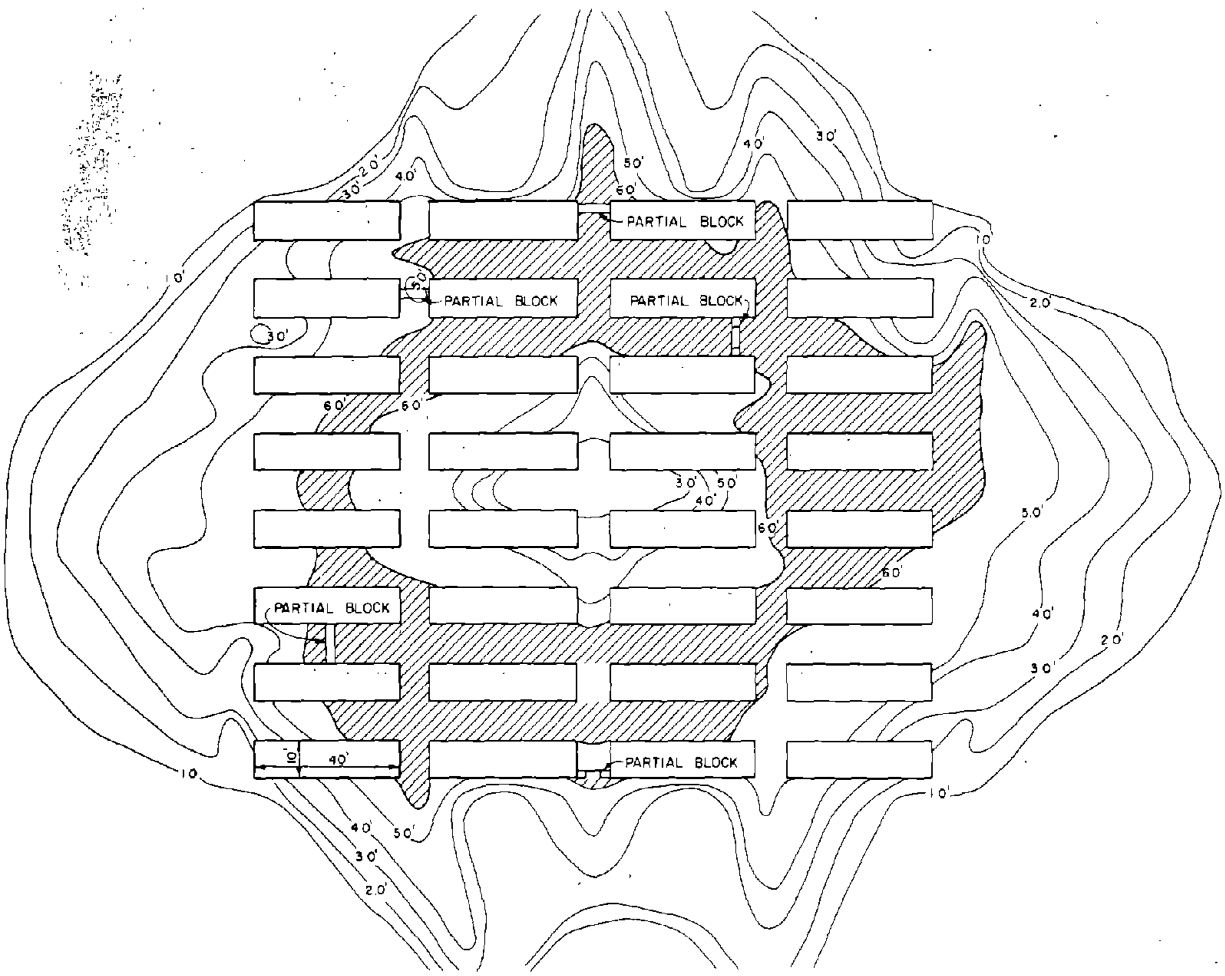
General

All tests with pillars were performed with the injection pipe geometrically centered in a symmetrical pattern of pillars. As a result the pattern of deposition on the level and near-level floor conditions were very nearly symmetrical about the injection pipe. Transport and deposit of the backfill material depend on the flow of slurry material in the mine cavity. Rock falls which block or partially block corridors will affect the radial flow and deposit patterns.

Deposition of backfill material in a field operation depends on the extent of open corridors which may or may not be symmetrical.

Bearing and Settlement Tests

Information was obtained from the Denver Bureau of Mines Office giving the minimum dry density and maximum dry density of the Rock Springs backfill material as 85 and 102.7 pounds per cubic foot (pcf), respectively. Earth Sciences Branch, USBR, determined these values as 85 and 108.4 pcf, respectively, as shown in the Appendix. The maximum settlement that could occur was calculated for material deposited at minimum dry density being compacted by subsidence above the mine cavity to a maximum dry density. Table 2 and Figure 29 gives the maximum settlement for backfill deposit at minimum dry density of 85 pcf, and subsidence causing vertical compaction to a maximum dry density of 102.7 and 108.4 pcf, up to a mine cavity depth of 6 feet. The above settlement calculations and Figure 29 are based on the assumption that no lateral movement of fill material occurs during compaction.



TEST 9
CONTOURS OF BACKFILL DEPOSIT AT END OF TEST.
PILLARS - 40% OF MINE VOLUME
CAVITY - 60% OF MINE VOLUME
MINE CAVITY LEVEL AND SUBMERGED.

BACKFILL MATERIAL FILLED TO ROOF LEVEL.

Figure 13

96

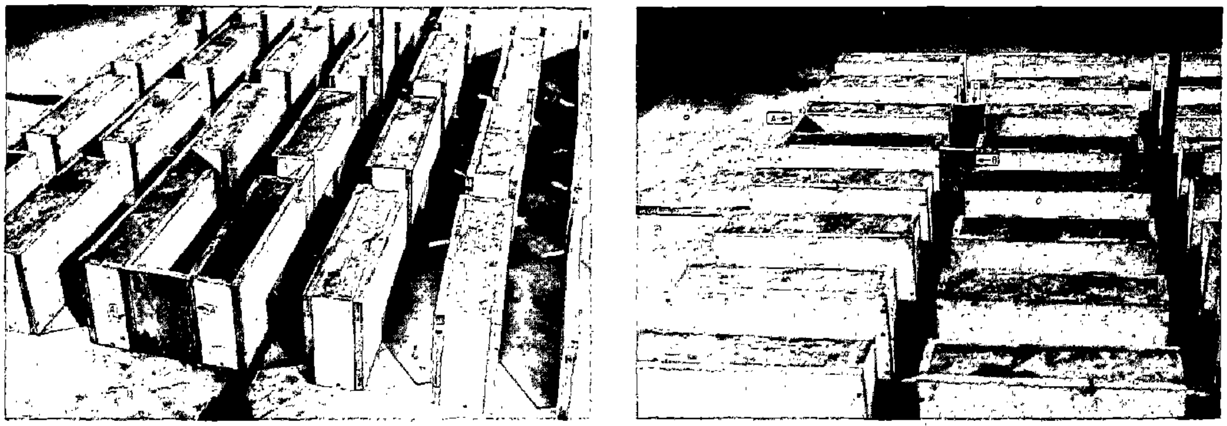


Figure 14. Test 10. Arrangement of pillars for backfill material pumped into a dry cavity. Blocks in one corridor were placed at A, B, and C as shown. Photo P801-D-73849

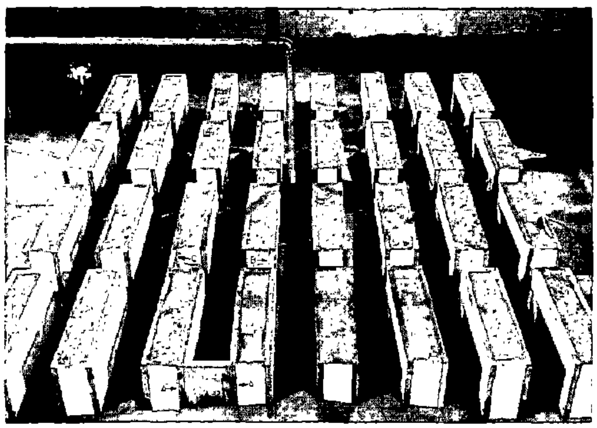


Figure 15. Test 10. Backfill material pumped into a dry cavity. Note deposit in corridor that is blocked. Slurry concentration was 20 percent. Photo P801-D-73851

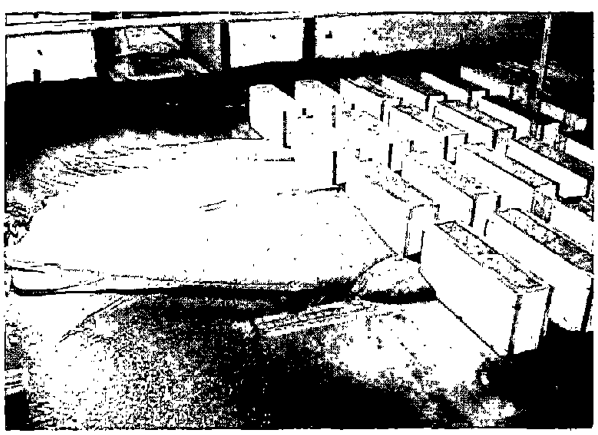
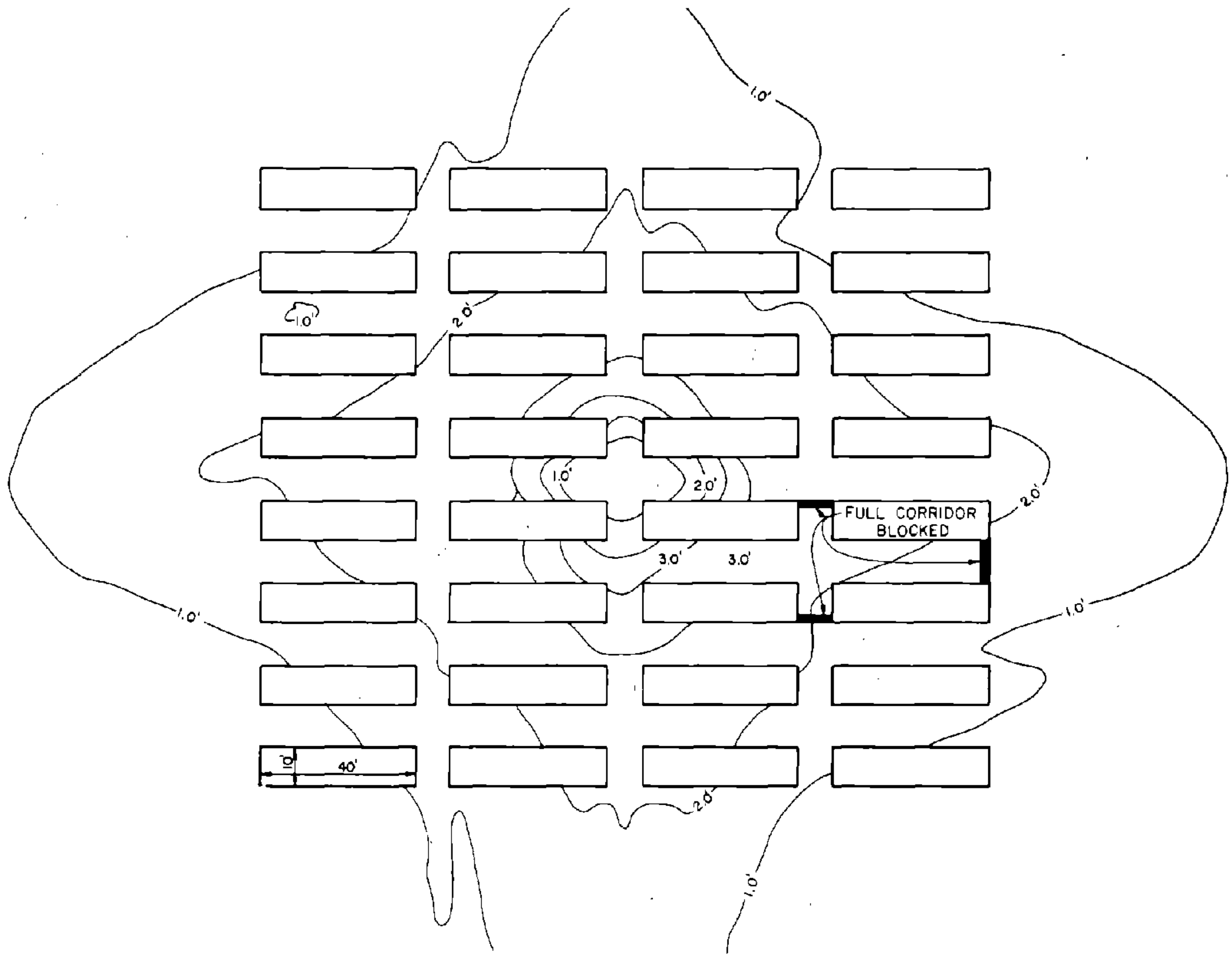


Figure 16. Test 10. Deposit at edge of pillar area shows slope of deposited backfill material for a dry cavity. Photo P801-D-73852



TEST 10
CONTOURS OF BACKFILL DEPOSIT AT END OF TEST.
PILLARS - 40% OF MINE VOLUME
CAVITY - 60% OF MINE VOLUME
MINE CAVITY LEVEL AND DRY.
SURFACE SLOPE OF BACKFILL DEPOSIT .050 TO .06

NO MATERIAL FILLED TO ROOF LEVEL

Figure 17

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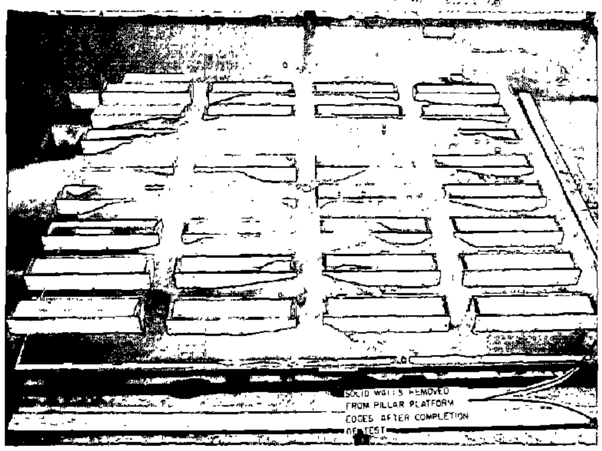

Figure 18. Test 14. Mine cavity was level and the horizontal and vertical geometrical scale 1:24. Slurry concentration was 10 percent. Photo P801-D-73853

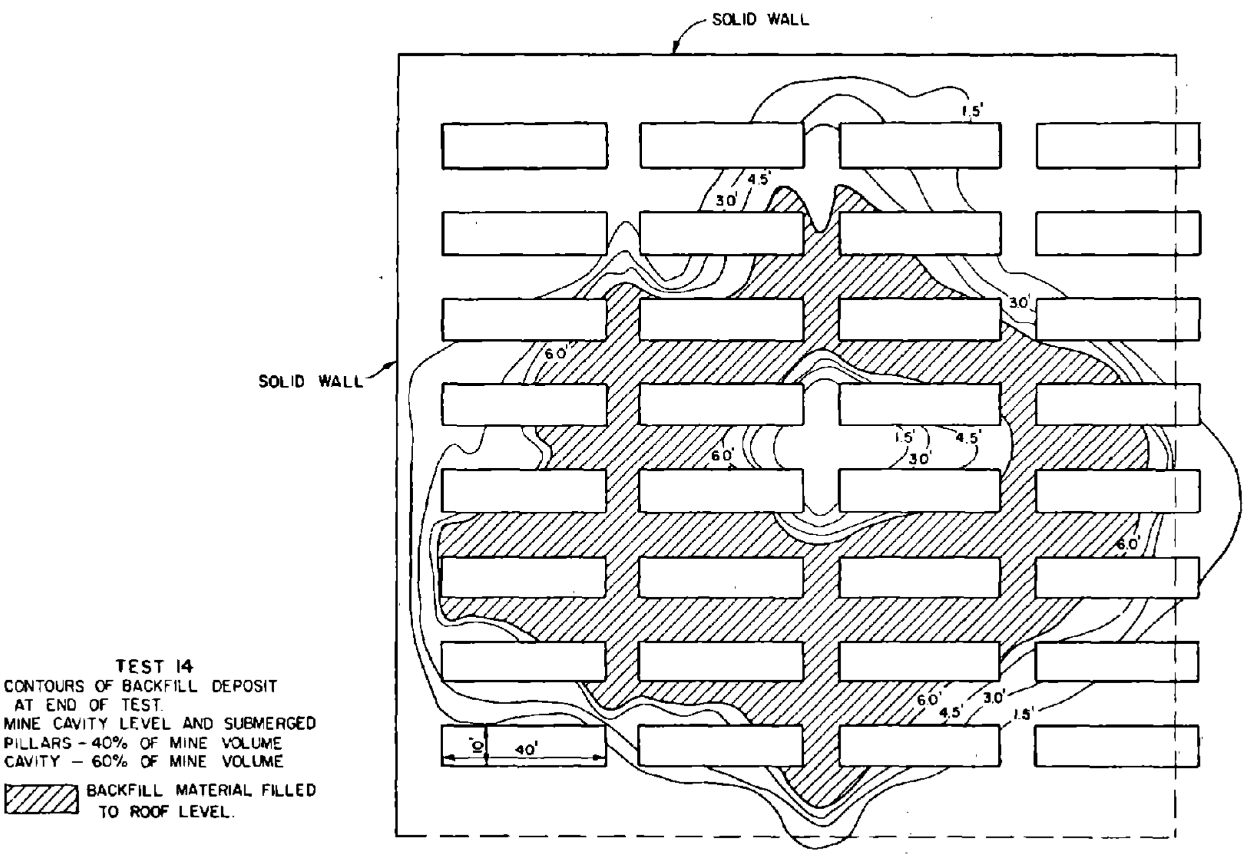

Figure 19

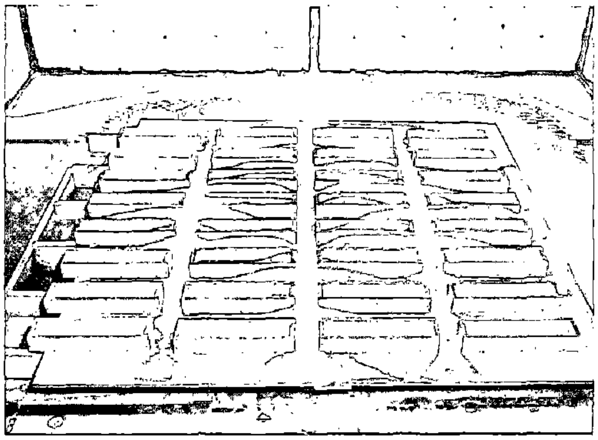

Figure 20. Test 16. Floor and roof of the mine cavity was level. Slurry concentration was 9 percent and the injection pipe velocity was about 16.5 feet per second. Photo P801-D-73854

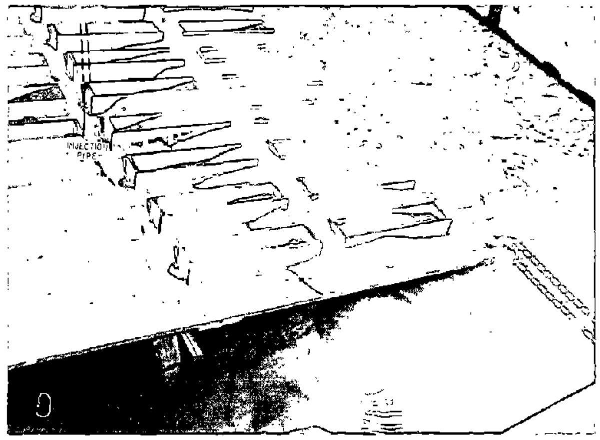

Figure 21. Test 11. Simulated backfilling of a mine cavity on a 15° slope. Water surface in the cavity was lower than the floor position under the injection pipe. Photo P801-D-73855

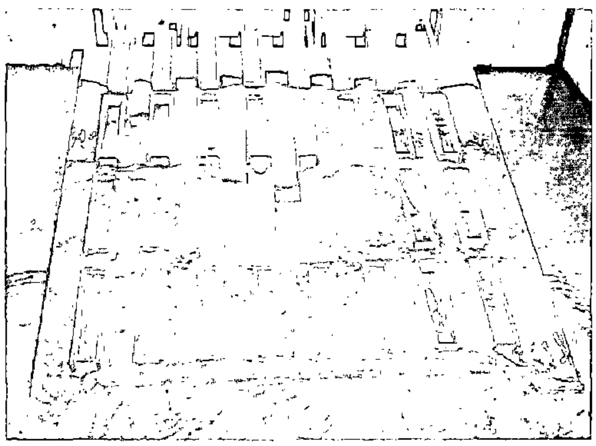

Figure 22. Test 12. Distribution of material on a 15° slope with the water surface in the cavity higher than the injection pipe exit. Photo P801-D-73856

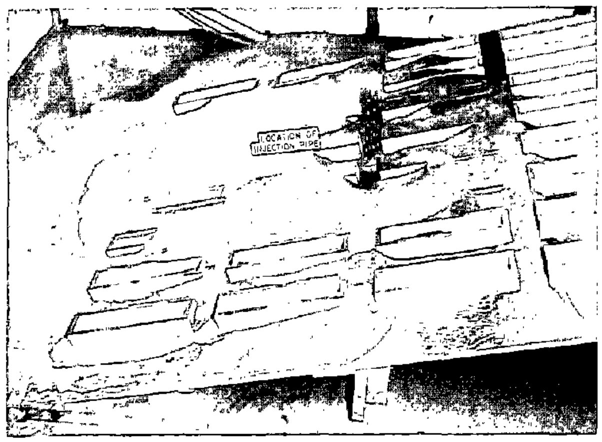

Figure 23. Test 13. A duplicate to Test 12, Figure 22.
Photo P801-D-73857

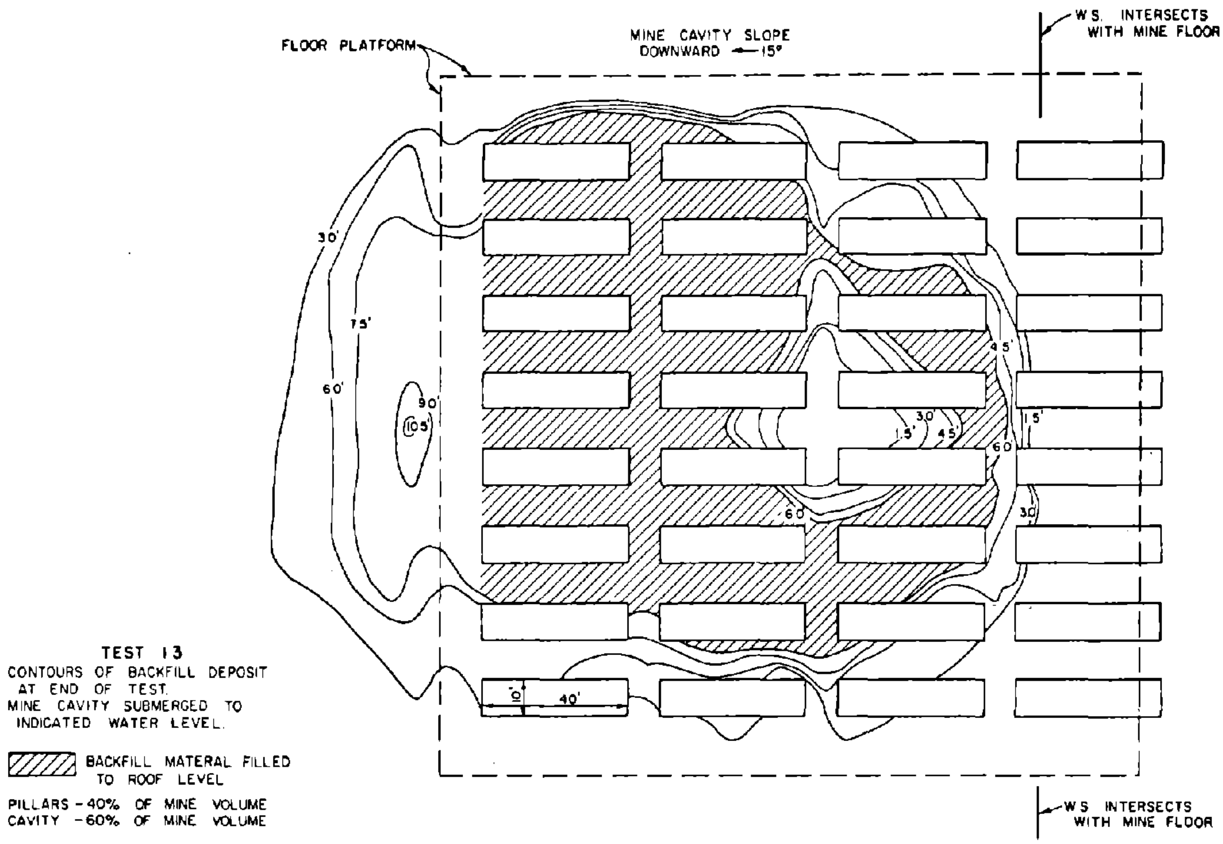

Figure 24

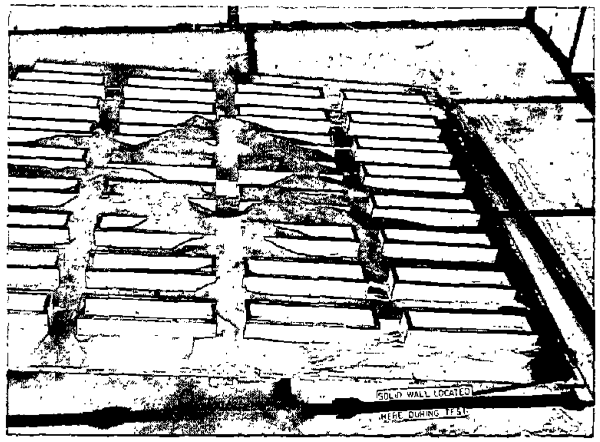

Figure 25. Test 17. The floor and roof of the mine cavity was sloping 5° from the horizontal. A solid wall was simulated at the downslope end of the pillars. Photo P801-D-73858

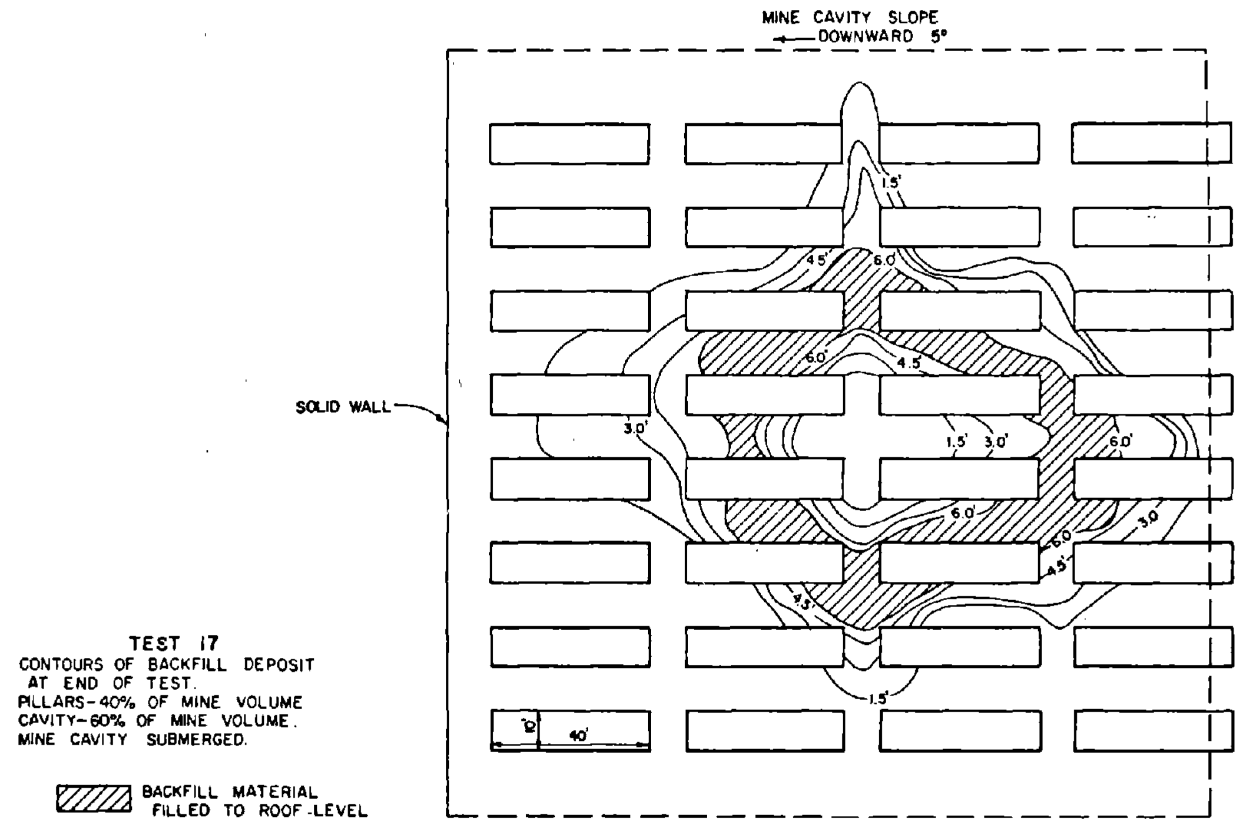

Figure 26

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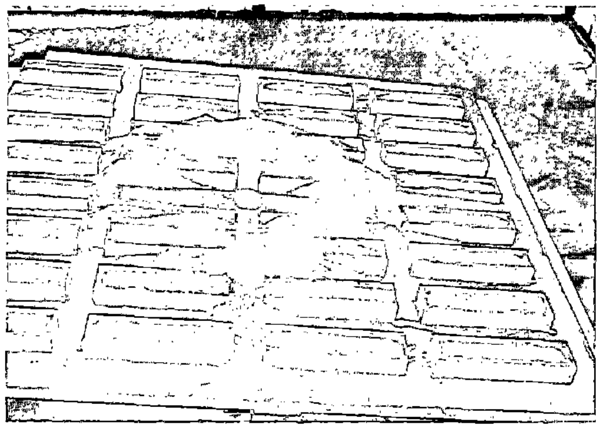

Figure 27. Test 18. Test conditions the same as Test 17 except Test 17 was continued for 18 minutes and Test 18 was continued for 32 minutes. Photo P801-D-73859

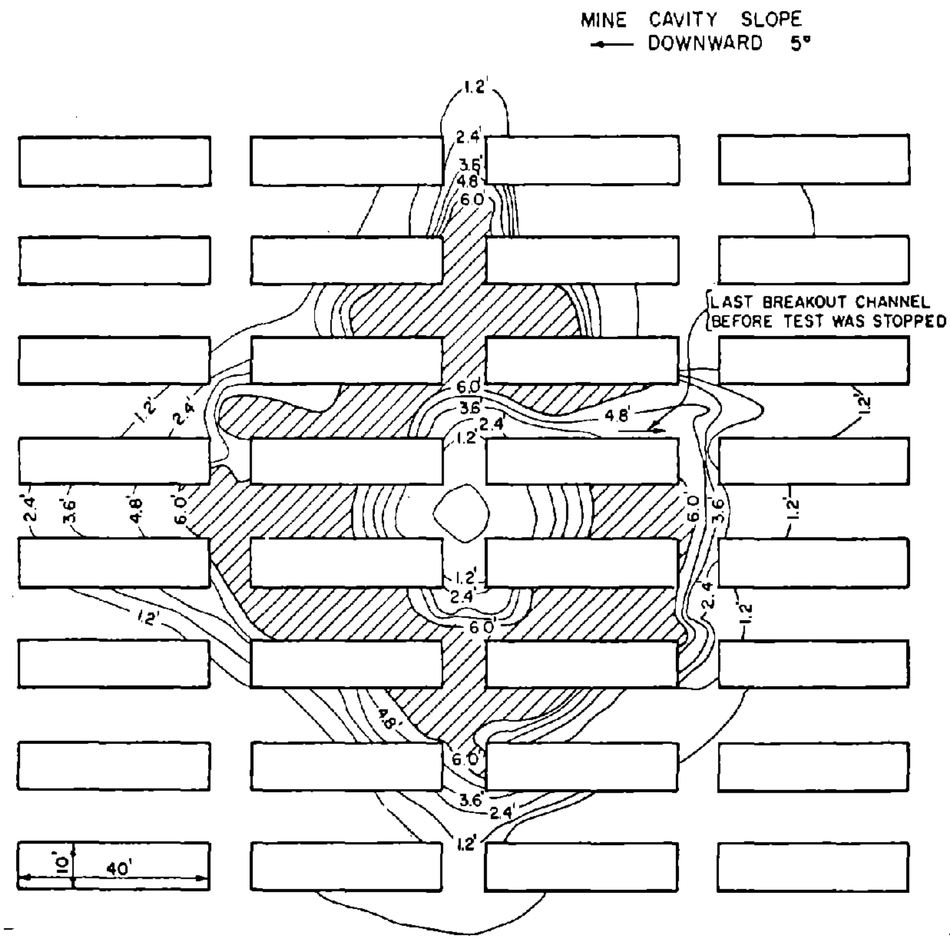

Figure 28

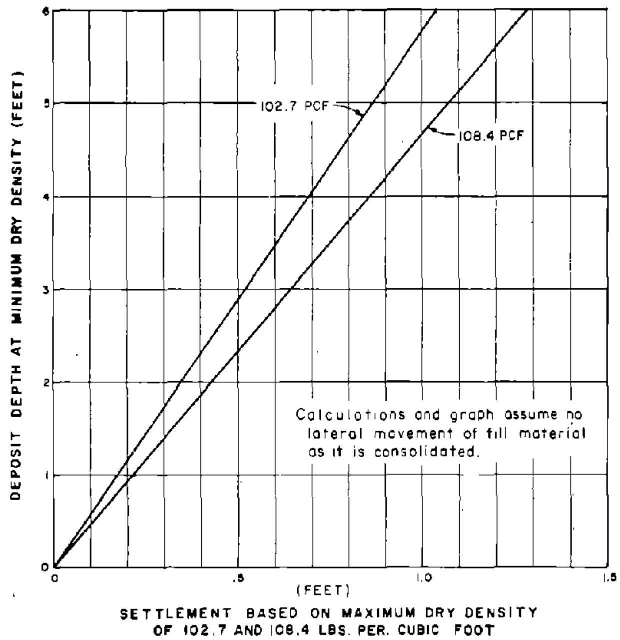

Figure 29. Consolidation of backfill material based on deposit depth and maximum dry density.

In-place Density

Measurements of in-place density of backfill material were made by taking samples of deposited backfill material after completion of Tests 7, 8, and 9. In-place density of backfill material deposited under water and not drained was measured as 73.6 pcf for approximately 3-inch depth of deposited material. After the material was drained, the in-place density was 93.8 pcf.

A few settlement tests were made on deposited backfill material between the pillars and outside the pillar area. Four-, six-, and twelve-inch-square platforms were loaded with 176.5 pounds and the vertical deformation of the sand was measured after water was drained from the deposited material. Variations in the increase of deformation for increase in load per unit area were apparent. The depth of deposited material was small (3 to 8 inches) in the areas the bearing tests were made, accounting for some of the differences in deformation per unit load. Settlement measured for load tests made between pillars was less than settlement measured for tests made without pillars (Test 7). The pillars provided a limited amount of confinement.

Earth Sciences Laboratory Tests

Because of the small depths of deposited backfill materials in the hydraulic model tests and variations of measurements of the bearing load tests, the Earth Sciences Branch of the Bureau was asked to make standard laboratory tests on a sample of the fine sand backfill material shipped from Rock Springs and used in the hydraulic model tests. Physical properties tests, bearing capacity, and stress-strain characteristics for estimating the amount of surface subsidence above a mine due to rock pressure loads were made. The results given in a memorandum from the Earth Sciences Branch to the Hydraulics Branch are included as an Appendix to this report.

REFERENCES

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Table 1

DATA SHEET—BACKFILLING MINE CAVITIES
HYDRAULIC MODEL TESTS FOR BUREAU OF MINES

Test number	Date	Approx conc % by wt	Q slurry cfs	V in 1" pipe slurry ft/sec	Center cavity		Level or slope degrees	Roof above floor inches	WS in box above floor ft	Comment	Piers in place #
					Radius inches	Sed dep above floor inches					
1	12-14-72	12	.049	≈ 9	NA	0	L	None	About 1.0		0
2	12-19-72	12	.049	≈ 9	12-1/4	5-5/8	L	None	1.0		0
3	12-21-72	12	.086	≈ 16	25-1/2	0	L	None	1.0		0
4	1-3-73	12	.086	≈ 16	17-1/2	5-3/4	L	None	.75	Surface slope tan = .08 = 4.4°	0
5	1-10-73	12	.086	≈ 16	25	5.4	L	9	1.0		20
6	1-12-73	17	.072–.084	13.2–16.9	13	1/2	L	9	1.0		28
7	1-19-73	16	.076–.082	14.1–15.1	13±	?	L	9	.95–.88		0
8	1-26-73	35	.020	3.67–13.4	18	4	L	9	.98–.86	Pillars partial & full block	32
9	2-1-73	25	.073–.078	13.6–14.3	21	4	L	9	.95	Pillar partial blocks	32
10	2-7-73	20	.044–.082	8.07–15.1	14	0	L	None	0–Dry	1 corridor blocked	32
11	2-14-73	24	.044	4.72	No cavity WS below injection pipe	0	15	3	Below pipe		32

Table 1—Continued

Test number	Date	Approx conc % by wt	Q slurry cfs	V in 1" pipe slurry ft/sec	Center cavity: Radius inches	Center cavity: Sed dep above floor inches	Level or slope degrees	Roof above floor inches	WS in box above floor ft	Comment	Piers in place #
12	2-14-73	35	.047	8.5	13	1/2	15	3	Above pipe 1.9		32
13	2-16-73	High	.053	10	13	0	15	3	Above pipe 1.9		32
14	2-23-73	10	.058	10	12	0	L	3	1.20	2 sides blocked solid wall	32
15	3-8-73	1.58–4.72	.046–.092	No deposit in box—Recirculating—Calibrating Venturi's.							
16	3-8-73	9	.089–.093	16.2–16.9	17	0	L	3	Above pipe	2 sand rates	32
17	3-14-73	8	.051–.063	9.3–11.4	13	0	5	3	Above pipe	1 side downslope blocked solid wall	32
18	6-5-73	4.3	.050–.053	9.1–9.6	13	0	5	3	Above pipe	Downslope side and sloping side blocked	32

Table 2

SETTLEMENT BASED ON THE ASSUMPTION THAT BACKFILL MATERIAL IS DEPOSITED AT THE MINIMUM DRY DENSITY AND VERTICAL SETTLEMENT OCCURS (NO LATERAL COMPACTION) UNTIL MAXIMUM DRY DENSITY IS REACHED

Original deposit depth ft	Depth after settlement ft	Depth of settlement ft	% settlement
	For minimum dry density = 85 pcf and maximum dry density = 102.7 pcf		
6.0	4.96	1.04	17.3
4.0	3.31	0.69	17.3
2.0	1.65	0.35	17.5
	For minimum dry density = 85 pcf and maximum dry density = 108.4 pcf		
6.0	4.71	1.29	21.5
4.0	3.14	.86	21.5
2.0	1.57	.43	21.5

APPENDIX

APPENDIX

Laboratory Studies on Sand for Mine Backfill, USBM, Rock Springs, Wyoming

Laboratory studies performed by the Earth Sciences Branch on sand for backfilling mines by the U.S. Bureau of Mines of Rock Springs, Wyoming. These results were transmitted by memorandum dated April 4, 1973.

Standard Properties Tests

1. The backfill material tested (Sample No. 54S-1) was classified as a silty sand (SM) containing 16 percent nonplastic fines and 84 percent predominantly fine sand (minus No. 50). The sample had a median grain size of 0.140 mm and a coefficient of uniformity of 5 (see Table 1 and Figure 1).

2. The results of the relative density test (Designation E-12, Earth Manual) indicated a minimum dry density of 85.0 pcf and a maximum dry density (dry method) of 108.4 pcf (see Table 1 and Figure 1). Based on these test results, the average in-place condition of the hydraulic backfill in the model tests (average dry density = 93.8 pcf) is approximately 44 percent relative density, which corresponds to a medium dense condition (see page 314, Earth Manual).

3. Ko-test—In order to simulate the high compressive ground pressures existing in underground mines, a triaxial shear test with no lateral strain was performed on a specimen (2 inches in diameter and 5 inches in length) to determine the value of "earth pressure-at-rest" (Ko). On an effective stress basis, Ko is defined as the ratio of the developed lateral pressure ($\overline{\sigma}_3$) to the applied axial pressure ($\overline{\sigma}_1$) under conditions of zero lateral strain. Other soil parameters determined in the Ko-test include Poisson's ratio (μ) and a modulus of deformation (E_c) which differs from Young's modulus (E) because the specimen is tested in a constrained manner by the application of a lateral pressure during the test and the strain is nonlinear and nonrecoverable.

The test specimen was placed at a dry density of 95.1 pcf, corresponding to 50 percent relative density, and sealed in a rubber membrane. The specimen is then placed in a triaxial pressure chamber which is filled with water to completely surround the specimen. During application of the axial load ($\overline{\sigma}_1$) to the specimen, the specimen is prevented from straining laterally by adjusting the lateral pressure ($\overline{\sigma}_3$) on the specimen to maintain zero lateral strain.

The maximum modulus of deformation (E_c) determined in the Ko-test was 14,682 psi and the corresponding value for Poisson's ratio was 0.28. The test results are summarized briefly below and more completely in Table 2 and Figure 2.

A "constrained" modulus, such as the secant modulus (M) or tangent modulus (ΔM), computed by dividing the axial stress by the axial strain, can be used to compute the vertical settlement under large loaded areas. The value of the secant modulus ($M = \overline{\sigma}_1/\epsilon_1$) for the overall range of stress is 11,474 psi while the tangent modulus ($\Delta M = \Delta\overline{\sigma}_1/\Delta\epsilon_1$) for the intermediate stress range of the test is 12,069 psi. See Figure 3.

The modulus of deformation is also a function of the effective stress acting on the specimen because the stress-strain is nonlinear and change in strain usually becomes less under higher increments of stress; therefore, the higher the effective stress the greater the value of E_c. In this test, the maximum applied axial stress was limited by the pressure limitations of the triaxial chamber which was 200-psi lateral pressure.

Bureau of Mines Study

A report[1] by the Bureau of Mines presents the results of an extensive study performed on mine backfill materials simulating high compressive ground pressures

Test Summary

Initial placement conditions			Modulus of deformation (E_c) psi	Ko	Poisson's ratio
Dry density pcf	Water content percent	Void ratio			
95.1	9.8	0.7588	14,682	0.39	0.38

[1] RI 7198 "Earth Pressure at Rest and One-dimensional Compression in Mine Hydraulic Backfills," D. E. Nicholson and R. A. Busch, October 1968.

 (Pages 22 thru 24 blank)

(up to 2,000-psi applied pressure) which resulted in a one-dimensional Earth-Pressure-at-Rest Model. These tests were performed on test specimens using a high-pressure compression chamber which permits no lateral strain to determine values of Ko for several different backfill materials placed at three density levels (loose, medium, and maximum densities). From these tests, the values of the tangent modulus (ΔM) are plotted against the void ratio (e) on Figure 22 of the referenced report to establish a trend line (see Figure 4). The tangent modulus determined from the Ko-test for Sample No. 54S-1 is also plotted on Figure 4 and it is seen to plot on the trend line indicating that the test results are comparable.

The conclusion made in the Bureau of Mines report (on page 38) is that under equivalent wall pressures a mine using loose backfill can expect an eight-fold increase in yield or compression of the backfill when compared to a mine using a compacted backfill.

Bearing Capacity

1. For a given soil pressure the settlement of a footing on sand depends upon the relative density and position of the water table. During placement of the hydraulic backfill in the mine it is not expected that complete filling will occur and that voids will exist between the top of the backfill (as deposited) and the mine crown. This void may eventually fill in with fallen rock and thus rock pressure may be transmitted to the backfill.

2. The allowable bearing capacity of submerged sand may be estimated by Terzaghi's general bearing-capacity equation for cohesionless soils:

$$q = \frac{\gamma' B}{2} N_\gamma$$

[2] *Foundation Engineering*, Leonards, G. A., pp 542-545.

where

q = allowable bearing capacity (psf)
γ' = submerged unit weight (pcf)
B = width of footing (ft)
$N\gamma$ = bearing capacity factor

Assumptions: backfill material

a. Density

γ_d = 93.8 pcf
W.C. = 29.2 percent (100 percent saturation)

Therefore γ' = (93.8 x 1.292) – 62.4
= 58.8 pcf.

b. Friction angle (ϕ) for fine sand of medium density = 32° (Leonards, p. 219)
c. Bearing capacity factor $N\gamma$ (ϕ = 32°) = 12 (Leonards, p. 542)[2]
d. For bearing width of 10 feet or B = 10 feet

Substituting in equation above:

$$q = 58.8 \left[\frac{10}{2}\right] 12$$

q = 3,528 psf for 1-inch settlement

Say q_a = 4,000 psf or 2 tsf

3. The settlement of the backfill is governed by the stress-deformation characteristics rather than the bearing capacity of the sand. The compressibility of the sand can be determined by the tangent modulus (ΔM) which increases with increasing relative density. The modulus also increases with an increase of the confining pressure which, in turn, is roughly proportional to the vertical pressure.

7-1735 (8-71)
Bureau of Reclamation

SUMMARY OF PHYSICAL PROPERTIES TEST RESULTS (Relative Density)

TABLE 1
SHEET 1 OF 1

PROJECT USBM ROCK SPRINGS, WYOMING FEATURE COAL MINE BACKFILL STUDY

IDENTIFICATION				PARTICLE-SIZE FRACTIONS IN PERCENT						CONSISTENCY LIMITS			SPECIFIC GRAVITY				RELATIVE DENSITY			
				FINES										PLUS NO. 4						
SAMPLE NUMBER 545	HOLE NUMBER	DEPTH - feet (m)	CLASSIFICATION SYMBOL	SMALLER THAN 0.005 mm	0.005 mm TO 0.074 mm	SAND NO. 200 (0.074 mm) TO NO. 4 (4.76 mm)	GRAVEL NO. 4 (4.76 mm) TO 3 IN. (76.2 mm)	COBBLES 3 IN. (76.2 mm) TO 5 IN. (127 mm)	OVERSIZE LARGER THAN 5 IN. (127 mm)	LIQUID LIMIT - %	PLASTICITY INDEX - %	SHRINKAGE LIMIT - %	MINUS NO. 4	BULK	APPARENT	ABSORPTION - %	MAXIMUM DRY DENSITY - pcf (gm/cm³)	MINIMUM DRY DENSITY - pcf (gm/cm³)	INPLACE DRY DENSITY - pcf (gm/cm³) 1/	RELATIVE DENSITY - %
1			SM	6	10	84	0				NP		2.68				108.4	85.0	93.8	44

1/ determined in hydraulic laboratory model study.

NOTE: Number in parentheses are metric equivalents of numbers directly above.

TABLE 1 SHEET 1 OF 1

EL-615B (8-71)
Bureau of Reclamation

SUMMARY OF TRIAXIAL SHEAR WITH ZERO LATERAL STRAIN (K_0) TEST RESULTS

TABLE 2
SHEET 1 OF 1

PROJECT Rock Springs, Wyo. FEATURE Coal Mine Backfill

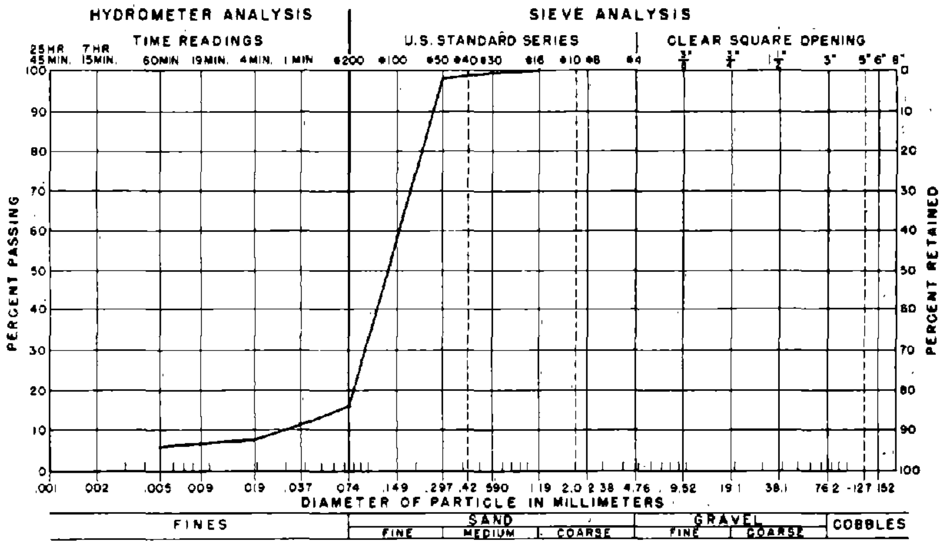
IDENTIFICATION			CLASSIFICATION SYMBOL	SPECIFIC GRAVITY	INITIAL SPECIMEN DATA			VOID RATIO "e"	VOLUME CHANGE - % = VERTICAL STRAIN - %	MAJOR PRINCIPAL EFFECTIVE STRESS "$\bar{\sigma}_1$" - psi (kg/cm^2)	POISSON'S RATIO "μ"	COEFFICIENT OF EARTH PRESSURE AT REST "K_o"	MODULUS OF DEFORMATION - "E" - psi (kg/cm^2)		
SAMPLE NUMBER	HOLE NUMBER	DEPTH - feet (m)			DRY DENSITY - pcf (gm/cm^3)	WATER CONTENT - %	DEGREE OF SATURATION - %								
545															
1			SM	268	95.1	9.8	34.6	.7588	0.00	0.0	—	—	—		
								.7501	0.49	41.7	0.24	0.31	7,127		
								.7413	0.99	120.7	0.26	0.35	9,870		
								.7344	1.38	196.6	0.26	0.35	11,484		
								.7310	1.58	241.5	0.27	0.37	12,106		
								.7274	1.78	278.6	0.27	0.37	12,359		
								.7257	1.88	356.4	0.28	0.39	14,682		
								.7239	1.98	356.5	0.30	0.43	13,213		
								.7221	2.08	384.4	0.30	0.42	13,664		
								.7204	2.18	427.7	0.32	0.46	13,724		

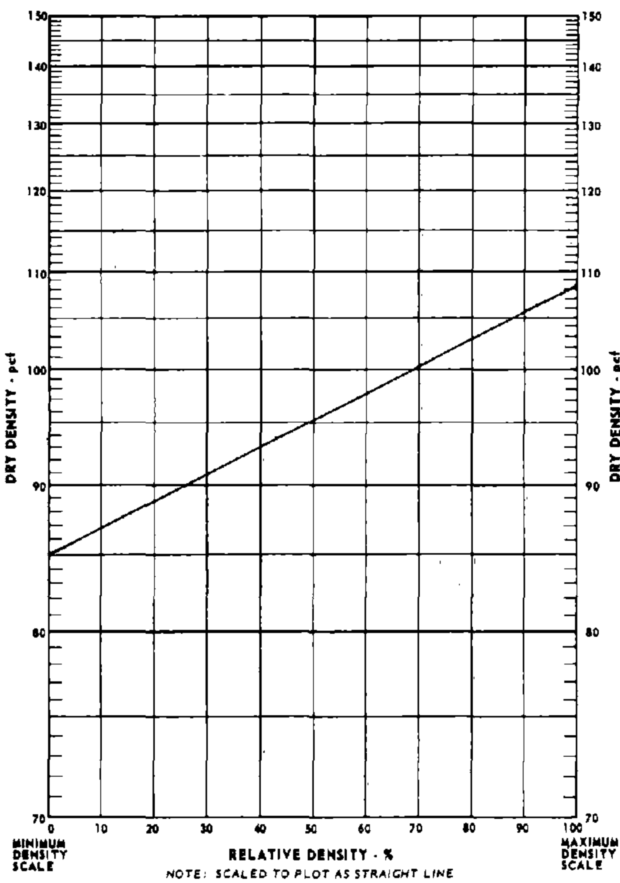
NOTE: Numbers in parentheses are metric equivalents of numbers directly above.

TABLE 2 SHEET 1 OF 1

7-1742 (8-70)
Bureau of Reclamation

PHYSICAL PROPERTIES SUMMARY PLOT (Relative Density)





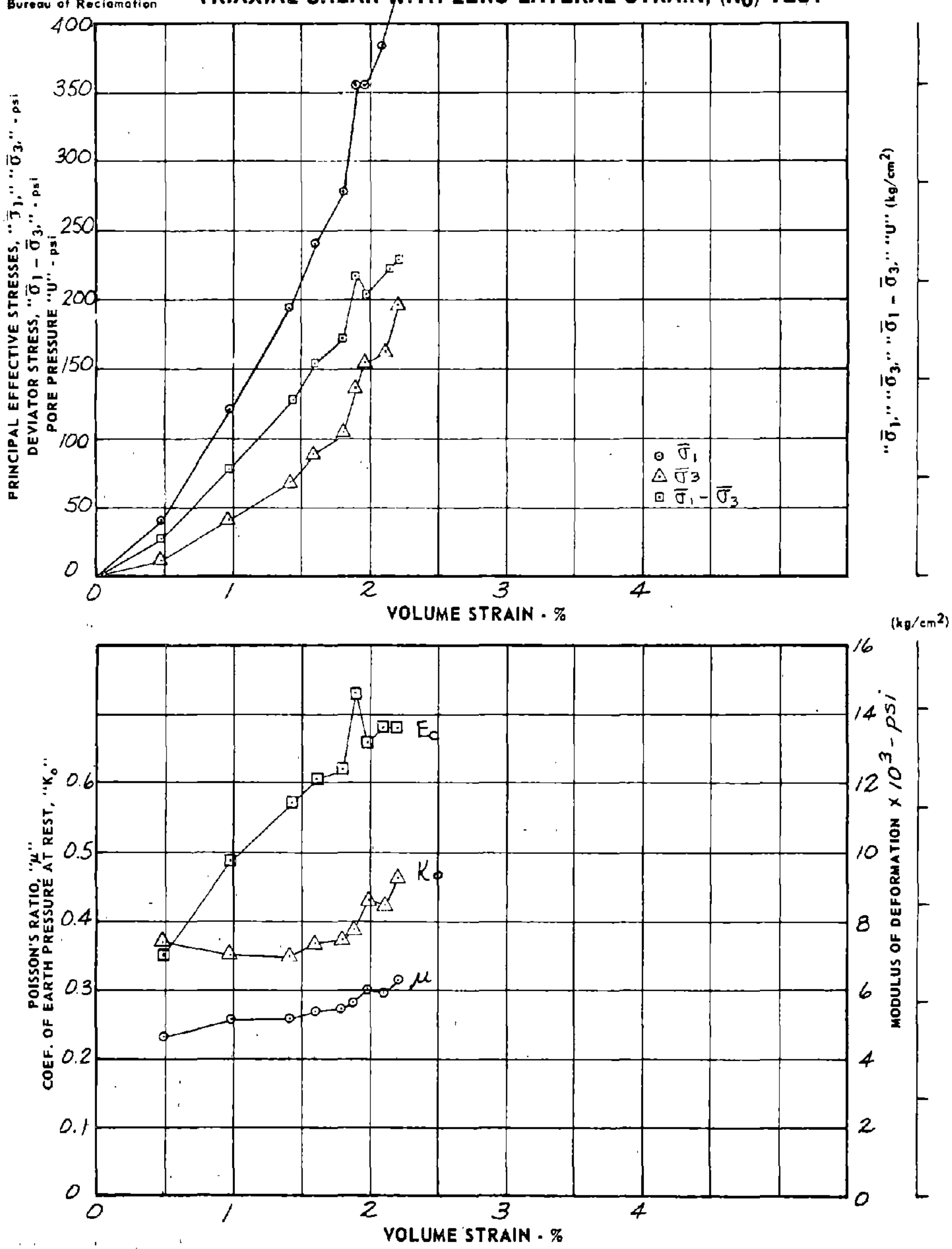
Classification Symbol SM
Gradation Summary
Gravel 0 %
Sand 84 %
Fines 16 %
Atterberg Limits
Liquid Limit ____ %
Plasticity Index NP %
Shrinkage Limit ____ %
Specific Gravity
Minus No. 4 2.68
Plus No. 4
Bulk ____
Apparent ____
Absorption ____ %
Relative Density
Minimum Density 85.0 PCF
(1.36 gm/cm3)
Maximum 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 7

EL-615A (12-70)
Bureau of Reclamation

TRIAXIAL SHEAR WITH ZERO LATERAL STRAIN; (K_0) TEST



Specimen Size 2" x 5" in (______ cm)

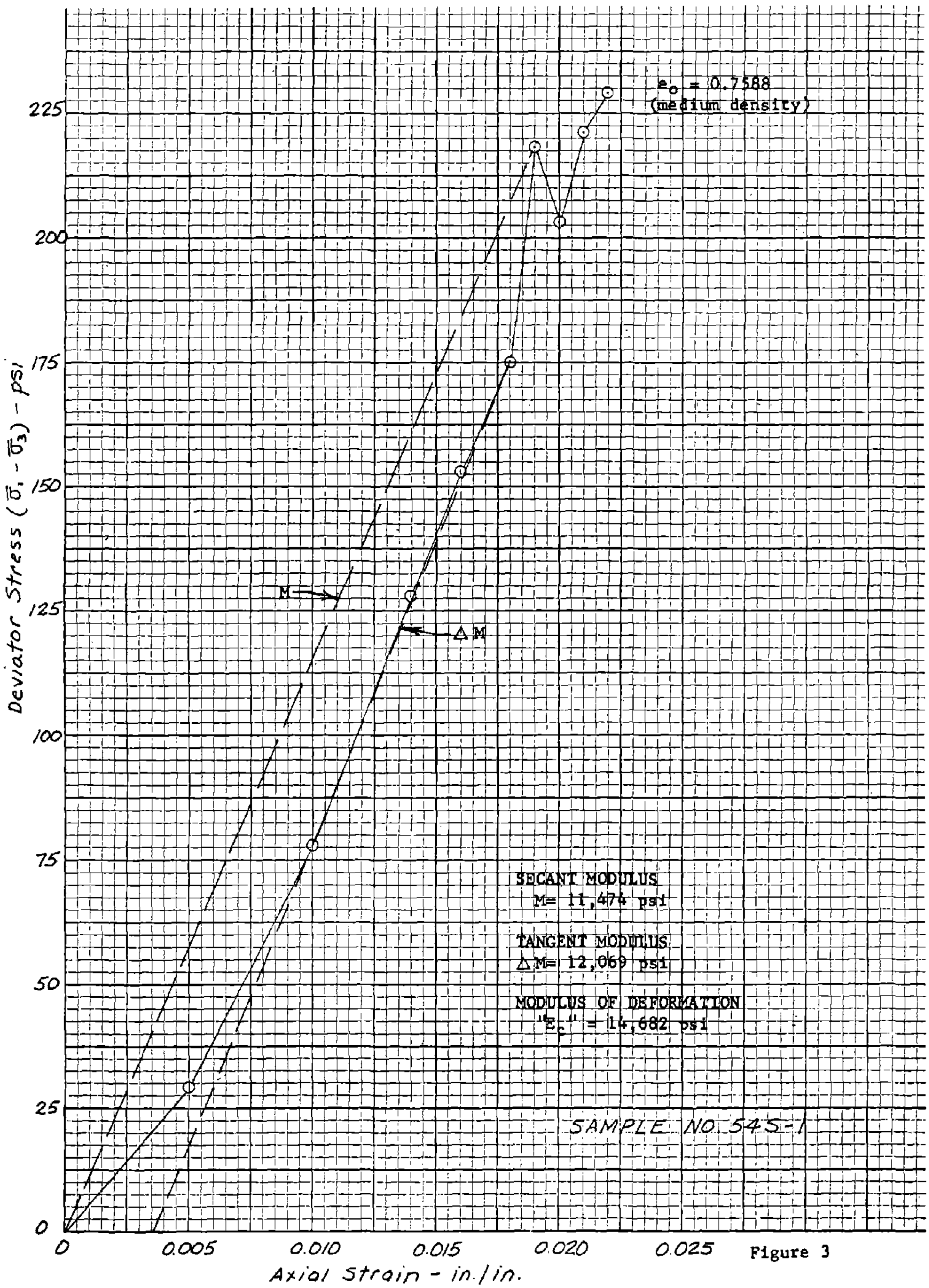
☒ Remolded

☐ Undisturbed

Sample No. 54S-1 Hole No. ______ Depth ______ ft (______ m)

Figure 2

113



e_o = 0.7588
(medium density)
SECANT MODULUS
M= 11,474 psi
TANGENT MODULUS
ΔM= 12,069 psi
MODULUS OF DEFORMATION
"E_c" = 14,682 psi
SAMPLE NO. 545-1
M
ΔM
Deviator Stress ($\bar{\sigma}_1 - \bar{\sigma}_3$) - psi
0
25
50
75
100
125
150
175
200
225
0
0.005
0.010
0.015
0.020
0.025
Axial Strain - in./in.

Figure 3

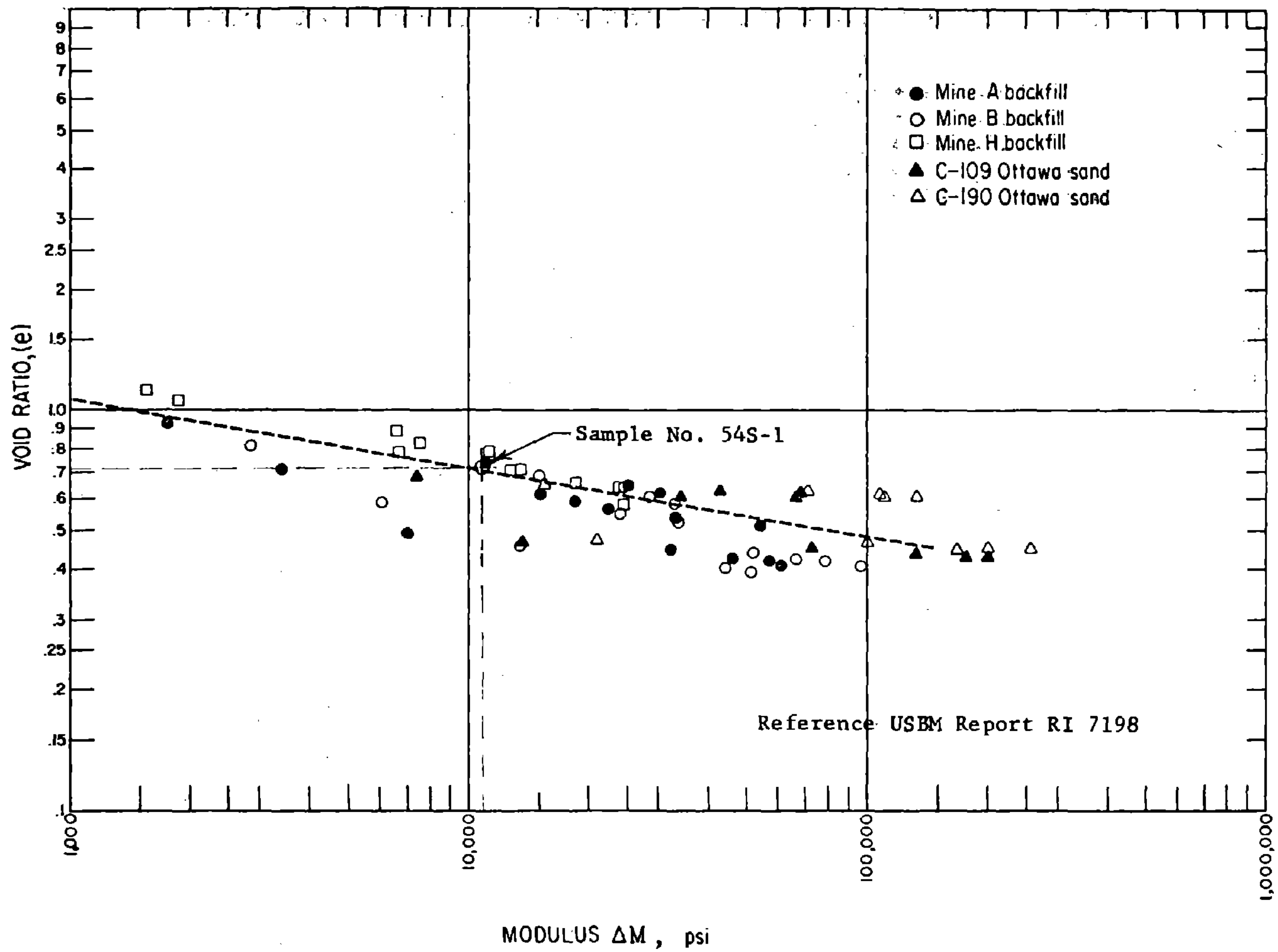


FIGURE 4. - Trend of Tangent Modulus of All Samples With Change in Void Ratio.

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	*0.965873×10^{-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	*4.4482×10^5	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)	4.1668	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