SPECIAL REPORT OF RESEARCH

conducted in

Department of Geology
and Geophysics
College of Earth and Mineral Sciences Experiment Station
The Pennsylvania State University

Coal Research Section

PREVENTION OF COAL MINE DRAINAGE FORMATION BY WELL DEWATERING

by

R.R. Parizek

An Investigation
Conducted Under the Auspices
of the

Department of Environmental Resources

of the

COMMONWEALTH OF PENNSYLVANIA

Contract Number CR-66

Special Research Report
Number SR-82
April 15, 1971

STATEMENT OF TRANSMITTAL

Special Report SR-82 transmitted herewith has been prepared by the Coal Research Section of the College of Earth and Mineral Sciences Experiment Station. Each of the Special Reports listed below presents results obtained in connection with one of the research projects supported by the Department of Mines and Mineral Industries of the Commonwealth of Pennsylvania or a technical discussion of related research. The following is a list of Special Research Reports issued to date:

SR-1	The Crushing of Anthracite	May 31, 1957
SR-2	Petrographic Composition and Sulfur Content of a Column of Pittsburgh Seam Coal	August 1, 1958
SR-3	The Thermal Decrepitation of Anthracite	September 15, 1958
SR-4	The Crushing of Anthracite with a Jaw Crusher	November 1, 1958
SR-5	Reactions of a Bituminous Coal with Sulfuric Acid	February 1, 1959
SR-6	Laboratory Studies on the Grindability of Anthracite and Other Coals	April 1, 1959
SR-7	Coal Characteristics and Their Relationship to Combustion Techniques	April 15, 1959
SR-8	The Crushing of Anthracite with an Impactor-Type Crusher	April 25, 1959
SR-9	The Ignitibility of Bituminous Coal (A Resume of a Literature Survey)	May 4, 1959
SR-10	Effect of Gamma Radiation and Oxygen at Ambient Temperatures on the Subsequent Plasticity of Bituminous Coals	May 6, 1959

SR-11	Properties and Reactions Exhibited by Anthracite Lithotypes Under Thermal Stress	May 11, 1959
SR-12	Removal of Mineral Matter from Anthracite by Chlorination at High Temperatures	June 22, 1959
SR-13	Removal Stability of a Coal Tar Pitch	June 25, 1959
SR-14	The effect of Nuclear Reactor Irradiation During Low Temperature Carbonization of Bituminous Coals	July 31, 1959
SR-15	Effect of Anthracite and Gamma Radiation at Ambient Temperature on the Subsequent Plasticity of Bituminous Coals	August 5, 1959
SR-16	The Isothermal Kinetics of Volatile Matter Release from Anthracite	August 25, 1959
SR-17	The Combustion of Dust Clouds: A Survey of the Literature	November 30, 1959
SR-18	The Ignitibility of Bituminous Coal	June 15, 1960
SR-19	Changes in Coal Sulfur During Carbonization	August 1, 1960
SR-20	The Radiation Chemistry of Coal in Various Atmospheres	September 12, 1960
SR-21	Reaction of Bituminous Coal with Concentrated Sulfuric Acid	October 1, 1960
SR-22	The Nature and Occurrence of Ash-Forming Minerals in Anthracite	December 30, 1960
SR-23	A Phenomenological Approach to the Batch Grinding of Coals	January 20, 1961
SR-24	The Unsteady State Diffusion of Gases from Anthracite at High Temperatures	January 21, 1961

SR-25	Some Advances in X-Ray Diffractometry and Their Application to the Study of Anthracites and Carbons	February 24, 1961
SR-26	The Filtration of Coal Solutions	March 17, 1961
SR-27	A Preliminary Investigation into the Application of Coal Petrography in the Blending of Anthracite and Bituminous Coals for the Production of Mettallurgical Coke	May 1, 1961
SR-28	Preparation and Properties of Activated Carbons Prepared from Nitric Acid Treatment of Bituminous Coal	August 15, 1961
SR-29	The Reactions of Selected Bituminous Coals with Concentrated Sulfuric Acid	August 31, 1961
SR-30	Investigations on the Operation of the Circular Concentrator for Cleaning Fine Coal	February 26, 1962
SR-31	Mineral Matter Removal from Anthracite by High Temperature Chlorination	March 26, 1962
SR-32	The Effect of Crusher Type on the Liberation of Sulfur in Bituminous Coal	April 29, 1962
SR-33	Investigation of the Circular Concentrator - Flotation Circle System for Cleaning Fine Coal	September 10, 1962
SR-34	Reactions of Coal with Atomic Species	September 24, 1962
SR-35	The Preparation Characteristics of the Bituminous Coal Reserves in Pennsylvania with Special Emphasis on Sulfur Reduction	October 31, 1962
SR-36	A Study of the Burning Velocity of Laminar Coal Dust Flames	November 5, 1962
SR-37	Molecular Sieve Material From Anthracite	November 16, 1962
SR-38	Studies of Anthracite Coals at High Pressures and Temperatures	April 29, 1963

SR-39	Coal Flotation of Low-Grade Pennsylvania Anthracite Silts	May 13, 1963
SR-40	Changes in the Physical Properties of Anthracite Upon Heat Treatment	July 10, 1963
SR-41	Some Aspects of the Chemistry of Sulfur in Relation to Its Presence in Coal	August 20, 1963
SR-42	The Unsteady State Diffusion of Gases from Coals	February 15, 1964
SR-43	The Effect of Concentration and Particle Size on the Burning Velocity of Laminar Coal Dust Flames	March 1, 1964
SR-44	The Electrokinetic Behavior of Anthracite Coals and Lithotypes	May 25, 1964
SR-45	An Investigation of the Cyclone for Fine Coal Cleaning	May 30, 1964
SR-46	The Utilization of Coal Refuse for the Manufacture of Lightweight Aggregate	September 1, 1964
SR-47	A Simulation Model on the Optimal Design of Belt Conveyor Systems	March 5, 1965
SR-48	Beneficiation of Fly Ash	April 12, 1965
SR-49	Application of Linear Programming Methods of Mine Planning and Scheduling	July 10, 1965
SR-50	Petrographic Composition and Sulfur Content of Selected Pennsylvania Bituminous Coal Seams	August 2, 1965
SR-51	Preliminary Investigations of Fog Disposal Methods Applicable to Greater Pittsburgh Airport	August 20, 1965
SR-52	Subsurface Disposal of Acid Mine Water by Injection Wells	August 10, 1965
SR-53	Roof Bolt Load and Differential Sag Measurements	September 3, 1965

SR-54	A Study of the Reactions Between Coal and Coal Mine Drainage	November 22, 1965
SR-55	Methods Employed for Underground Stowing (A Resume of a Literature Survey)	February 28, 1966
SR-56	Computer Simulation of Materials Handling in Open Pit Mining	June 6, 1966
SR-57	The Evaluation of Anthracite Refuse as a Highway Construction Material	July 30, 1966
SR-58	An Investigation of the Cleaning of Bituminous Coal Refuse Fines by an Experimental Hydrocyclone	August 15, 1966
SR-59	Chlorination and Activation of Pennsylvania Anthracites	October 24, 1966
SR-60	Development and Testing of an Injection Well for the Subsurface Disposal of Acid Mine Water	February 1, 1967
SR-61	Investigations of the Cyclone Washing of Fine Coal in Water	December 12, 1966
SR-62	Linear Programming Short Course	May 1, 1967
SR-63	Planning Belt Conveyor Networks Using Computer Simulation	May 15, 1967
SR-64	The Economic Importance of the Coal Industry to Pennsylvania	August 1, 1967
SR-65	An Evaluation of Factors Influencing Acid Mine Drainage Production from Various Strata of the Allegheny Group and the Ground Water Interactions in Selected Areas of Western Pennsylvania	August 15, 1967
SR-66	Potential Injection Well Strata for Acid Mine Water Disposal in Pennsylvania	October 25, 1967
SR-67	A Survey of the Location, Magnitude, Characteristics and Potential Uses of Pennsylvania Refuse	January 25, 1968
SR-68	A Landscape Architectural Approach to Reclamation of Development of Deep Anthracite Strip Pits	July 1, 1968

SR-69	The Oxygenation of Iron (II) - Relationship to Coal Mine Drainage Treatment	November 1, 1968
SR-70	A Method for Determining the Partition Curve of a Coal Washing Process	February 1, 1969
SR-71	The Revegetation of Highly Acid Spoil Banks in the Bituminous Coal Region of Pennsylvania	February 10, 1969
SR-72	Acid and Aluminum Toxicity as Related to Strip-Mine Spoil Banks in Western Pennsylvania	May 1, 1969
SR-73	Designing a Rock Bolting System	May 15, 1969
SR-74	An Electrokinetic Study of Bituminous Coal Froth Flotation and Flocculation	May 23, 1969
SR-75	A Complete Coal Mining Simulation	November 10, 1969
SR-76	An Investigation of the Natural Beneficiation of Coal Mine Drainage	May 15, 1970
SR-77	Application of a Continuous Mining System in a Medium Pitching Anthracite Bed of Northeastern Pennsylvania	May 31, 1970
SR-78	Evaluation of a Monorail Mine Haulage System	February 1, 1971
SR-79	Pennsylvania Anthracite Refuse: A Summary of a Literature Survey On Utilization and Disposal	March 15, 1971
SR-80	Investigation of the Haldex (Simdex) Process for Beneficiating Coal Refuse: Hungarian Practice - 1969	March 31, 1971
SR-81	Coal Mine Refuse Disposal in Great Britain	March 31, 1971

William Spackman, Director
Coal Research Section and
Office of Coal Research
Administration

SUMMATION OF RESULTS

Large quantities of ground-water may be encountered in deep coal mines which must be treated to meet water-quality standards before being discharged. Source beds supplying leakage to deep mines may be dewatered during and after mining under favorable hydrogeologic conditions to prevent pollution, thereby minimizing treatment costs and improving working conditions. Requisite hydrologeologic data to determine the feasibility of dewatering may be obtained during the coal exploration program provided that both hydrogeologic and coal exploration programs are planned in advance and coordinated. Hydrogeologic data required include the spatial distribution, thickness, and geometry of aquifer and non-aquifer units; hydraulic boundaries that either restrict the flow of ground-water, stratigraphic pinchouts, fault offsets, erosional unconformities, or serve as recharge sources, channel sandstones, flooded deep mines, fractured roofrock etc.; permeability and storage distribution; infiltration capacity of streambed sediments; vertical permeability of confining beds; and hydraulic heads among and between various aquifer and non-aquifer units.

It generally will not be possible to predict water level declines in source beds resulting from various pumping schemes using idealized mathematical models because rocks associated with coal beds vary considerably in their distribution and aquifer hydraulic properties. Rather digital or electrical analog modeling methods must be employed to determine if dewatering schemes are feasible. The number, depth, spacing, and pumping rate of wells required to produce desired water level declines can be determined using these analytical methods even under complex hydrogeologic conditions. Cost of field studies and hydrologic analyses are small compared to the cost of treating some mine waters hence, dewatering programs should not be rejected prematurely.

Two hypothetical mines were considered which are overlain by source beds. A four square mile roof area was assumed, two square miles of which were under a permeable channel sandstone. In one case the leakage rate before dewatering was calculated to be 1.96×10^5 gpd/sq. mile and 1.16×10^5 gpd/sq. mile after dewatering for 120 days using only three wells. A five well system would reduce leakage to 7.56×10^4 gpd/sq. mile in 120 days for the same system. Additional wells and prolonged pumping would reduce leakage dramatically.

The cost of dewatering can be significantly less than treating mine waters. The total annual cost in dollars per million gallons of water a day delivered to the surface for wells with poor yields (75 gpm each) might approach $19,000 per year, $10,000 per year for wells with 230 gpm yields, and $6,900 per year for wells with up to 760 gpm yields. A 25 year life of the well and equipment was assumed using 1965 figures. Treatmeant costs may range from $.05 to $1.25 per 1,000 gallons depending upon initial water quality and release standards. The lower figure is for waters with about 2 to 3 ppm of iron and up to 20 ppm of acidity, the upper figure for water with 500 to 700 ppm iron and 2,000 to 5,000 ppm acidity.

It is conceivable that 2.17 mgd may be pumped in a dewatering scheme at a cost of $15,000 to $25,000 per year. Maximum and minimum treatment costs for 2.17 mgd may range from $39,600 to $990,000 per year depending upon the quality of water treated. This does not include pumping costs. Not all mines will be suited for dewatering. Favorable and unfavorable conditions are outlined.

TABLE OF CONTENTS

LIST OF FIGURES

ACKNOWLEDGEMENTS

This study was supported in part by the Mineral Conservation Section, College of Earth and Mineral Sciences, The Pennsylvania State University and in part by a grant from the Coal Research Board of the Commonwealth of Pennsylvania as an amendment to Coal Research Board Contract CR-66.

To A. Prickett and C. G. Lonnquist of the Illinois State Water Survey, Urbana, Illinois who kindly provided a copy of their unpublished computer program which was used to compute drawdowns in hypothetical mines considered in this study.

The illustration of Silverthread Falls (Figure 8), was provided by the proprietors of Dingman Falls, Pennsylvania.

INTRODUCTION

Active and abandoned deep and strip coal mines are a source of potential or actual ground and surface water pollution. Mine operators are being subjected to stringent water quality standards which in some states requires that mine waters be collected and treated during and after mining to prevent pollution. In most mining districts coal was extracted before existing laws were passed or laws are still forthcoming. Mining companies may since have abandoned the area or have gone out of business leaving pollution problems to the present generation. Pollution ultimately may be abated in these regions and the existing mine operators or water user in the watershed must pay the bill. In Pennsylvania, the tax payer also is involved in this responsibility. It is clear that no one abatement or treatment scheme will suffice. Rather, a variety of techniques may be employed which are designed around local circumstances, needs, and know how. The cost will be horrendous. According to Pennsylvania Department of Mines and Mineral Industries estimates, more than 4.5 billion gallons of mine drainage is produced in Pennsylvania alone each day, and, contrary to popular opinion, mining activity is on the increase. Nationally, options on coal deposits are being purchased by oil companies who have their own plans for using coal as a raw material. All indications are that coal mining will continue to expand at a time when pollution from all activities is coming under increased attention and restrictions at the local and national level.

Only one potential solution to the coal mine drainage problem will be reviewed here, i.e., that of mine dewatering to prevent ground-water pollution within the mine environment. Little appears to have been done to evaluate the merrits of mine dewatering or apply this pollution prevention scheme on a routine basis. Deep mines have been planned routinely to minimize pumping costs by mining up slope, thus allowing water to drain freely to the surface by gravity. These discharges may be favorable or of very poor quality depending upon geochemical conditions within the mine. In general little has been done within mines to prevent mine water from coming in contact with acid producing strata except after the water reaches conveyances used to divert water from the mine. Some mining operations are being designed to minimize pollution after mining ceases. Coal is mined down dip with the idea that the mine will flood when abandoned thereby minimizing or eliminating air circulation and related chemical reactions responsible for pyrite oxidation. Mine seals may be placed on dry mine openings to restrict air circulation but these need not be as elaborate as mine seals required to dam water within the mine. For the latter, precautions must be taken to prevent blowouts under excessive hydrostatic pressures which can develop in flooded mines. Hazzards associated with potential blowouts should be apparent when one considers the vast volumes of water required to flood some deep mines and the fact that mine roof debris can exert additional pressures on mine seals during extensive roof falls.

By whatever mine layout used, mine waters may become contaminated and require treatment to meet water quality standards. The ideal solution to this problem is to collect or divert waters within or outside the mine in advance of it becoming contaminated, thereby eliminating costly treatment. Hopefully, waters would be diverted only during mining, and mine sealing would solve pollution problems after the mine is abandoned. Under many field circumstances, pumping may be required during mining and even after mines have been abandoned where other solutions are not feasible.

MINE DEWATERING

The normal engineering practices of collecting and removing waters from mines will not be discussed as these are routinely considered by mining engineers. Mine waters are inevitable in many mines but the volume of polluted water produced can be controlled to some extent. Anything that can be done to divert these waters from the mine with least contact with wall and waste rock containing acid forming minerals is desirable. Mine dewatering as used here is the practice by which the flow of water into the mine is curtailed by well development or is enhanced under controlled conditions to prevent it from becoming contaminated. It is unrealistic to assume that wet coal mines can be rendered entirely dry mainly because the hydrogeologic frameworks of most coal mines do not lend themselves to a massive dewatering scheme as is possible for some mines located within or adjacent to highly permeable sandstone or carbonate aquifers. Rather, it should be possible to control excessive flows into mines derived from under and overlying, more permeable strata.

DATA REQUIREMENTS

The hydrogeologic framework of each mine must be accessed to determine if and to what extent dewatering is feasible. Dewatering may be desirable to improve working conditions as well as to minimize pollution. To determine the feasibility of dewatering a mine, rather specific hydrologic and geologic data are required which must be collected from a carefully designed field exploration program. This program can be combined with exploration required to define coal reserves, thereby minimizing costs. Most of the expense is already borne by the test drilling program used to outline coal reserves. Specific additional data can be collected from these same test borings. These data are required to determine the number and spacing of wells required to reduce vertical leakage into deep mines.

Geometry of Beds: The spatial distribution and thickness of all aquifer and non-aquifer units above and immediately below the coal must be known. Drill cuttings of all beds penetrated and cores of selected units are required. Bed thickness, elevation of tops, structure maps, geologic cross-sections, and lithologic variation and other maps may be prepared from these data. In cases where coal and overlying rock sequences are complex, borehole geophysical surveys may be desirable or mandatory to aid in correlation studies. Gamma-ray, sonic and caliper logs are helpful and now can be run on 3 inch or NX diameter or larger holes at reasonable cost where a number of holes are logged.

Correlation studies are manditory because the various rocks associated with coal beds tend to be repetitious vertically and show abrupt changes laterally. Channels marking buried erosion surfaces may thin or cut away some units entirely. Hydrologic analyses presuppose a knowledge of the distribution and continuity of rock units.

Aquifer and Confining Bed Permeabilities: The coefficient of field permeability defined as the rate of flow of water, in gallons per day, through a cross-sectional area of 1 square foot of the aquifer under a hydraulic gradient of 1 foot per foot (gpd/sq. ft.) at the prevailing temperature of the water must be known for both aquifer and non-aquifer materials. Estimates of the spatial distribution of permeability is required within individual aquifers and confining beds. Coefficients of permeability normally will vary with rock type which may be predetermined during aquifer distribution studies. Rock types may include coal, underclay, shale, slate, siltstone, clean to dirty sandstone, limestone, residual and transported soils, and glacial deposits. In place permeability measurements are superior to values obtained from drill cores because openings contributed by joints and fractures are included with intergranular openings. Joint permeability is particularly significant if not dominant in shallow rock strata of western Pennsylvania (50 to 500 feet) and may increase permeability values by ten to a thousand times when compared to intergranular permeability values taken alone.

A measure of vertical and horizontal permeability of individual beds also is useful because beds associated with coals tend to be highly anisotropic. Horizontal to vertical permeability contrasts of 10 to 1, 20 to 1, and even 50 to 1 have been obtained from core analysis for an area near Kylertown, Pennsylvania. Values of the coefficients of permeability multiplied by bed thickness define coefficients of transmissivity expressed as the rate of flow of water, in gallons per day, through a vertical strip of the aquifer 1 foot wide, and extending the full saturated thickness under a hydraulic gradient of 1 foot at the prevailing temperature of the water (gpd/ft.). These, combined with aquifer storage properties may be used to calculate water level declines within aquifers induced by pumping wells.

The coefficient of vertical permeability of confining beds, often expressed as the rate of vertical flow of water, in gallons per day, through a horizontal cross-sectional area of 1 square foot of the confining bed under a hydraulic gradient of 1 foot per foot at the prevailing temperature (gpd/sq. ft.) must be known because it contributes to the rate of leakage into mines through roofrock or leakage rate into aquifer source beds supplying water to deep mines. These values are used with mine roof areas, confining bed thicknesses, and hydraulic head data to compute the potential leakage rate into a mine before and after a hypothetical dewatering scheme is initiated to determine if the inflow of water can be significantly reduced.

In place values for the coefficients of horizontal and vertical permeability are difficult to define where secondary fractures predominate. Core values set lower limits as a start. Field testing programs must be carefully defined using pumping test, slug test, packer test or other procedures to obtain meaningful permeability data. Because of the variety of beds encountered, tests must be run on individual units to determine their variability before beds are grouped together, for more general testing within a given region. In some mining districts, it may be reasonable to lump shales and siltstones and even limestones together for testing. Local experience will aid in making these decisions.

Vertical permeabilities of confining beds underlying or overlying a particular aquifer and aquifer hydraulic properties may be determined using pumping test data and recent theory developed by Neuman and Witherspoon (1969), or theory given by Hantush and Jacobs (1955), Hantush (1956) and Hantush (1960). Selected methods will be briefly outlined for those not familar with the procedure involved.

Hantush and Jacob (1955, p. 95-100) derived an equation describing the nonsteady-state drawdown distribution in a leaky artesian aquifer. Their formula assumes that the aquifer is infinite in areal extent and is of the same thickness throughout; that it is homogeneous and isotropic; that it is confined between an impermeable bed and a bed through which leakage can occur; that the coefficient of storage is constant; that water is released from storage instantaneously with a decline in head; that the well has an infinitesimal diameter and penetrates the entire thickness of the formation; that leakage through the confining bed into the aquifer is vertical and proportional to drawdown; that the hydraulic head in the deposits supplying leakage remains more or less uniform; and that the coefficient of vertical permeability remains constant. Despite this list of assumptions, suitable field conditions can be found for which the theory applies. In mining districts, vertical permeability of confining beds above coal may be determined using the Hantush-Jacob theory provided that the coal bed serves as an aquifer--this is commonly the case--and the underclay below the coal serves as the impermeable bed. Vertical leakage into sandstone aquifers or other source beds may be determined as well from seperate pumping tests conducted on each water-yielding unit.

The leaky artesian formula may be written as:

$$s = (114.6\ Q/T)W(u, r/B) \qquad [1]$$

where: $$u = 2693\ r^2\ S/Tt \qquad [2]$$

and $$r/B = r/\sqrt{T/(P'/m')} \qquad [3]$$

s = drawdown in observation well, in feet.

r = distance from pumped well to observation well, in feet.

Q = well discharge, in gallons per minute (gpm).

t = time after pumping started, in minutes (min.).

T = coefficient of transmissivity, in gallons per day per foot (gpd/ft.).

S = coefficient of storage of aquifer, fraction.

P' = coefficient of vertical permeability of the confining bed, in gallons per day per square foot (gpd/sq.ft.).

m' = thickness of the confining bed through which leakage occurs, in ft.

W(u,r/B) is read as the "well function for leaky artesian aquifers" and symbolically represents the integral of their equation.

The leaky artesian formula, Equation [1], may be solved using a modification of the type curve graphical method devised by Theis and described by Jacob (1940). Walton (1962, p. 5) describes the procedure as follows:

"Values of W(u,r/B) are plotted against values of 1/u on logarithmic paper and a family of leaky artesian type curves is constructed as shown in Plate 1" [Plate 1 is given by Walton (1962) or values of W(u,r/B) in terms of the practical range in u and r/B may be obtained from Hantush (1956)]. "Values of s plotted on logarithmic paper of the same scale as the type curves against values of t describe a time-drawdown field data curve that is analogous to one of the family of leaky artesian type curves.

"The time-drawdown field data curve is superposed on the family of leaky artesian type curves, keeping the W(u,r/B) axis parallel with the s axis and the 1/u axis parallel with the t axis. In the matched position a point at the intersection of the major axes of the leaky artesian type curve is selected and marked on the time-drawdown field data curve. The coordinates of their common point (match point) W(u,r/B), 1/u, s, and t are substituted into equations 1, 2, and 3 to determine the hydraulic properties of the aquifer and confining bed. T is calculated using equation 1 with the W(u,r/B) and s coordinates. S is determined using equation 2, the calculated value of T, and the 1/u and t coordinates of the match point . . . . The value of r/B used to construct the particular leaky artesian type curve found to be analogous to the time-drawdown field data curve is substituted in equation 3 to determine P'."

Walton (1962 p.5) also encourages the use of a distance-drawdown method of analysis to complement the time-drawdown method described above to calculate the hydraulic properties of aquifers and confining beds. He states that values of W(u,r/B) are plotted against values of r/B on logarithmic paper and again a family of leaky artesian type curves is constructed. "Values of s plotted against values of r on logarithmic paper of the same scale as the type curves describes a distance-drawdown field data curve that is analogous to one of the family of leaky artesian type curves. The distance-drawdown field data curve is superposed on the family of leaky artesian type curves, keeping the W(u,r/B) axis parallel with the s axis and the r/B axis parallel with the r axis. The distance-drawdown field data curve is matched to one of the family of leaky artesian type curves. In the matched position a point at the intersection of the major axis of the leaky artesian type curves is selected and marked on the distance-drawdown field data curve." Match-point coordinates W(u,r/B), r/B, s, and r are substituted in equations 1 and 3 to determine coefficients of transmissivity and vertical permeability. The value of u/r^2 used to construct the particular leaky artesian curve found to match the actual distance-drawdown field data curve is substituted in equation 2 to compute S.

From experiences in western Pennsylvania it is clear that wells used to collect field data for these analyses must be cased to the same aquifer being pumped otherwise a considerable variation in water levels may be noted in observation wells located even a few feet apart. Leakage from seemingly unimportant beds above the aquifer being tested may in fact cause a rise in water levels in the observation wells during testing.

An inspection of the family of leaky artesian type curves reveals that the curves flatten out in time indicating that the vertical leakage rate eventually balances the pumping rate. The cone of depression within an artesian aquifer under steady-state conditions is described by equation 4 given by Jacob (1946).

$$s = [229\ QK_0\ (r/B)]/T \tag{4}$$

where: $$r/B = r/\sqrt{T/(P'/m')} \tag{5}$$

s = drawdown in observation well, in ft.

r = distance from pumped well to observation well, in ft.

Q = well discharge, in gpm.

T = coefficient of tranmissivity, in gpd/ft.

P' = coefficient of vertical permeability of the confining bed, in gpd/sq. ft.

m' = thickness of confining bed through which leakage occurs, in ft.

K_0 (r/B) = modified Bessel function of the second kind and zero order.

Jacob's (1946) graphical method for determining values of T and P' under steady-state leaky artesian follows. A steady-state leaky artesian type curve is prepared by plotting values of K_0 (r/B) against r/B on logarithmic paper. An example of this plot and values of K_0 (r/B) in terms of the practical range at r/B is given in Walton (1962, Plate 2, Appendix B). Aquifer test data collected under steady-state conditions are plotted on logarithmic paper of the same scale on the type curve with r as the abscissa and s as the ordinate. This distance drawdown field data curve is matched to the steady-state leaky artesian type curve, keeping respective axes parallel. A match-point common to the two curves is selected and marked. Match-point coordinates K_0 (r/B), and r are substituted into equations 4 and 5 to determine T and P'. The coefficient of storage cannot be computed in this case because all water is derived from leakage from an overlying source bed and no water is being withdrawn from storage within the aquifer. The steady-state leaky artesian formula may be used with data collected from water supply wells drilled in the mining district which have been pumped for a prolonged but known period of time and for which only recent time-drawdown data are available.

For both steady-state and non steady-state leaky artesian equations to apply, the confining bed below the aquifer is assumed to be impermeable and only the overlying confining bed is assumed to be leaky. This can be assumed to be the case in some instances where the permeability contrasts between the two beds is likely to be great. In other cases, however, where the regional flow of ground water was upward to the source bed before pumping began and a second source bed and leaky confining bed overlaid the first, both confining beds may contribute water to the aquifer during pumping. Incorrect values for the aquifer and confining bed hydraulic properties will be obtained in this case using Hantush and Jacob leakage theory. In still other cases the duration of test pumping may be insufficient and induced vertical leakage effects will not be measurable in observation wells drilled to obtain time-drawdown data. Time-drawdown data will fall along the Theis type curve which assumes no leakage rather than along one of the family of leaky type curves. In this case only the coefficients of transmissivity and storage of the aquifer being pumped may be determined.

Theory developed by Neuman and Witherspoon (1968 and '69) appear suitable to overcome problems of short pumping duration and leakage through two confining beds.

The Neuman-Witherspoon (1969) solution to vertical leakage for multiple aquifer systems has definite advantages in that it provides a solution even when water is removed from storage within the confining beds above or below an aquifer under development, and when water level changes occur within source beds supplying water to the aquifer being developed. Their procedure should be applicable to systems that contain an arbitrary number of aquifers, aquitards and aquicludes, and in their opinion, may possibly enable one to evaluate the properties of all the layers in such multiple aquifer systems being stressed by pumping wells.

Shale and siltstone confining beds in western Pennsylvania are highly fractured as a rule and are relatively thin, hence leakage through confining beds should be appreciable and water levels in over or underlying source beds should be effected during pumping. Also numerous confining beds and aquifers are the rule within any given sequence of the beds associated with coal.

Neuman and Witherspoon (1969, p. 153-158) present a solution for the relatively simple but common system consisting of two aquifers and one aquitard or confining bed. They describe their matching procedure somewhat as follows.

A production well is completed to the bottom of the lower aquifer to determine the hydraulic properties of that aquifer and overlying source bed. A second observation well is drilled at a distance of r_0

feet from the pumping well and is completed in the lower aquifer. One or more observation wells are drilled in the aquitard at the same r_0 distance from the pumped well and are open opposite small segments of the aquitard at different depths. They state that the perforated intervals of observation wells in the aquitard should be short. These observation wells serve as piezometers. A constant rate pumping test is conducted on the production well and drawdowns are measured as a function of time in all observation wells. Drawdown data are plotted on transparent logarithmic paper against the time, t, and are compared with type curves of dimensionless time (see Figs. IV-2 through IV-9 of Neuman and Witherspoon, 1969).

Type curves for the unpumped aquifer are not used in the interpretation of the field data in this case, whereas, a number of other type curves are required.

Values of B_{11} and r/B_{11} are obtained by noting the particular type curve used to match field time-drawdown data. A match point is selected anywhere on the overlapping portion of the two sheets and values of s_D and t_D that correspond to the drawdown s and time t are recorded. The properties of the pumped aquifer and aquitard now can be determined. Given values of Q_1 and H_1 the pumping rate, and H_1, the thickness of the aquifer, the permeability of the aquifer can be obtained using equation 6.

$$K_1 = \frac{114.6Q\, s_D}{H_1 s} \qquad [6]$$

when Q = pumping rate in gallons per minute

s = drawdown in the observation well in feet

H_1 = thickness of aquifer in feet

K_1 = permeability of the aquifer in gallon per day/square foot

s_D = dimensionless drawdown or $\frac{4\pi Ts}{Q}$ where T is the coefficient of transmissivity

The specific storage of the aquifer is determined from

$$S_{s1} = \frac{K_1 t}{7.48\, r_0^2\, t_D} \qquad [7]$$

where t = time since pumping began in days

r_0 = distance from pumped well to observation well in feet

K_1 = permeability of the aquifer in gpd/ft.^{2}

t_D = dimensionless time in the pumped aquifer or

$\frac{\alpha i t}{r^2}$ where r = radial distance in feet from the pumped well to the observation well, t = time since pumping started, and αi, = hydraulic diffusivity of the aquifer or $\frac{K}{S_s}$

S_s = specific storage in feet^{-1}

The permeability of the aquitard can be calculated from

$$K_1' = \frac{(r/B_{11})^2\, K_1 H_1 H_1'}{r_0^2} \qquad [8]$$

where K_1' = coefficient of permeability of the aquitard

r/B_{11} = value obtained from the type curve used to match field time-drawdown data and used to compute the type curve.

K_1 = coefficient of permeability of the aquifer

H_1 and H_1' = thickness of aquifer and aquitard and r_0 as defined earlier.

The specific storage of the aquitard can be computed from equation 9.

$$S_{S1}' = \frac{16\ B_{11}^2\ H_1^2 K_1\ S_{S1}}{K_1'\ r_0^2} \qquad [9]$$

where: B_{11} = values used to compute the type curve which matched a particular set of field data.

H_1, K_1 K_1' and S_{S1} as defined earlier

and S_{S1}' = specific storage of the aquitard in feet^{-1}

Neuman and Witherspoon conclude that similar results could perhaps be obtained with only one or two observation wells in the aquitard but that it seems the larger the number of observation wells, the easier it may be to match field data uniquely to one of the families of type curves. Also, performing additional tests at different radial distances from the pumping well should increase the liklihood of obtaining meaningful results.

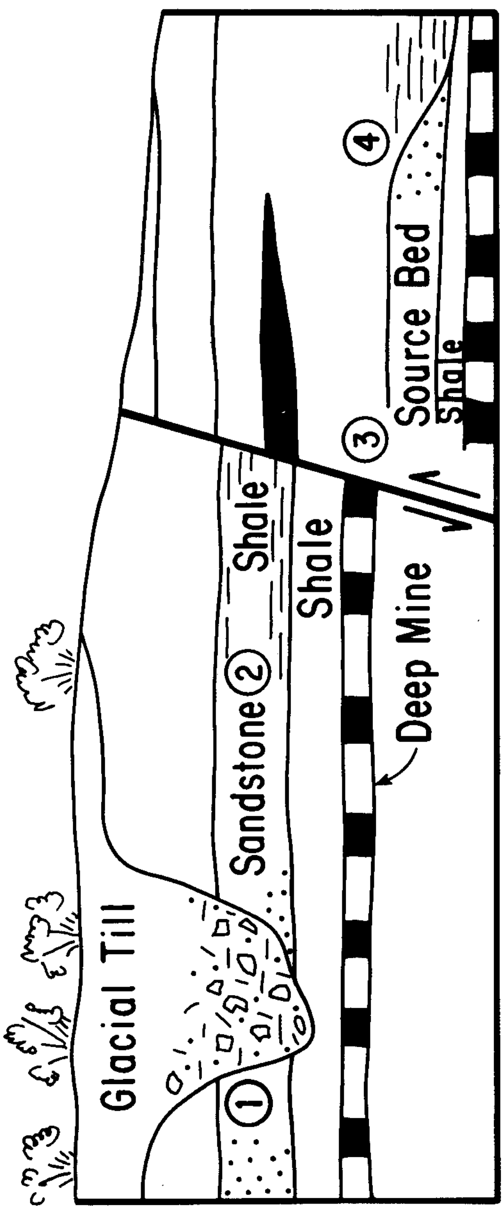
Coal exploration test holes can be used in these field pumping tests in combination with intentionally drilled wells which are designed to give maximum hydrologic data. These hydrologic tests and additional test holes represent the single largest expense associated with determining the feasibility of dewatering deep mines.

Aquifer Storage Properties: Storage properties of an aquifer may be expressed as the coefficient of storage for confined aquifers defined as the volume of water the aquifer releases from or takes into storage per unit surface area of the aquifer per unit change in the component of head normal to that surface, and as the gravity yield for unconfined aquifers which are essentially dewatered as the water table is lowered. Values of the coefficient of storage obtained from pumping tests normally are larger than values obtained from core porosity tests and may be determined at the same time field tests are run to determine the coefficients of permeability, see equations 1 and 2. Values for these coefficients are required to determine the time rate of water level decline accompanying various pumping rates from one or more dewatering wells.

Hydrologic Boundary Conditions: Mine dewatering feasibility studies require a hydrologic systems analysis approach because a number of water wells may be involved and the influence of dewatering may extend throughout a broad region until the limits of aquifers are reached or until sources of recharge are encountered capable of balancing pumping rates. Negative or no flow barrier boundaries may be defined from aquifer geometry studies and pumping test data. A sandstone pinchout against a tight shale bed, fault offsets and similar other features may serve as hydraulic barriers (Fig. 1). Recharge barriers may be represented by rivers and creeks, flooded mines, or beds with greatly increased permeability. Flooded spoil piles in backfilled strip mines, highly fractured, saturated roof rock above old abandoned mines, glacial gravels buried in deep bedrock channels that cut away coal beds and similar features also may serve as recharge boundaries (Fig. 2).

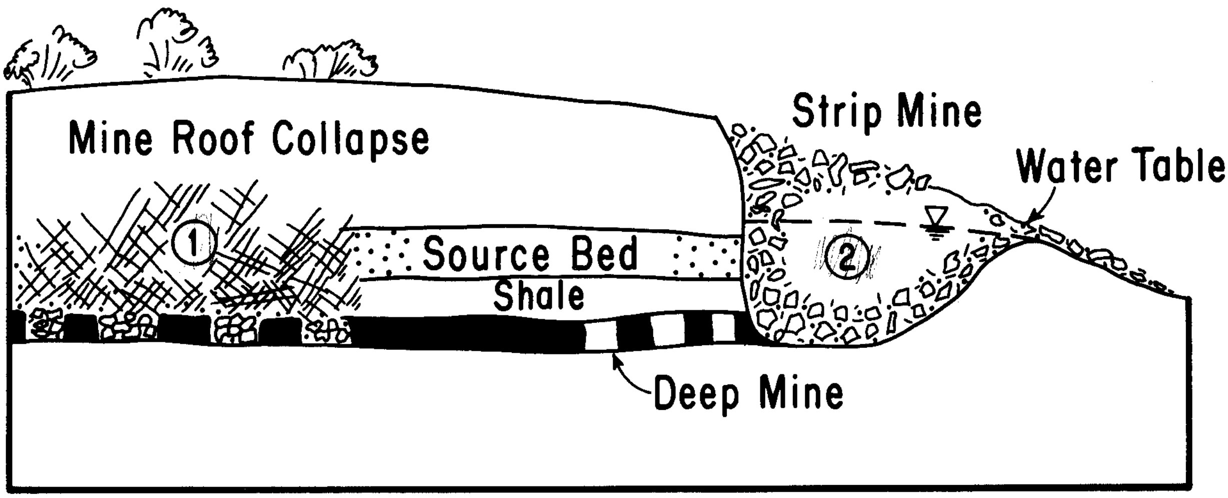
Water Table and Piezometric Surfaces: Hydraulic heads acting between and within beds influences flow rates within aquifer and non-aquifer units. Head changes are routinely measured accompanying controlled pumping tests and are of value when collected on a regional basis as well. Changes in head between a source bed and proposed deep mine will dictate the leakage rate into the mine all factors remaining constant. Mine dewatering schemes will be concerned largely with reducing these

Figure 1. Hydraulic barrier boundary conditions caused by glacial till deposited against sandstone source bed (1), sandstone replaced by shale (2), fault offset of source bed (3), and erosional truncation of source bed capped by shale (4).

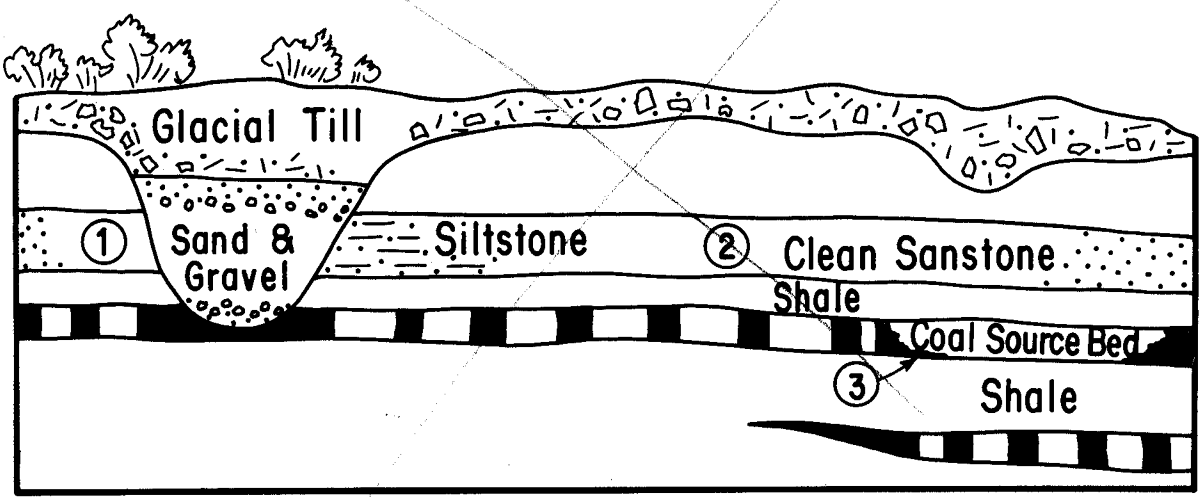


Glacial Till
1
Sandstone
2
Shale
Shale
Deep Mine
3
Source Bed
Shale
4

Figure 2. Recharge barrier boundaries caused by mine roof collapse which is a source of recharge to the source bed (1a), back fill in strip mine with shallow water table (2a), permeable glacial outwash gravel deposited against less permeable source bed (1b), siltstone grading to well sorted sandstone (2b), coal source bed in contact with flooded deep mine (3b).


Mine Roof Collapse
1
Source Bed
Shale
Strip Mine
Water Table
2
Deep Mine

(a)


Glacial Till
1
Sand &
Gravel
Siltstone
2
Clean Sanstone
Shale
Coal Source Bed
3
Shale

(b)

heads because the leakage rate is a direct function of head equation 10. Equation 10 is a modification of Darcy's law given by Walton (1960 p. 23).

$$Qc = \frac{P'}{M'} \Delta h \; Ac \qquad [10]$$

Where:

Qc = leakage through confining beds, in gallons per day.

P' = vertical permeability of confining beds, in gallons per day per square foot.

M' = thickness of confining bed through which leakage occurs in feet.

Δh = difference between the head in the aquifer and the source bed above the confining bed, in feet.

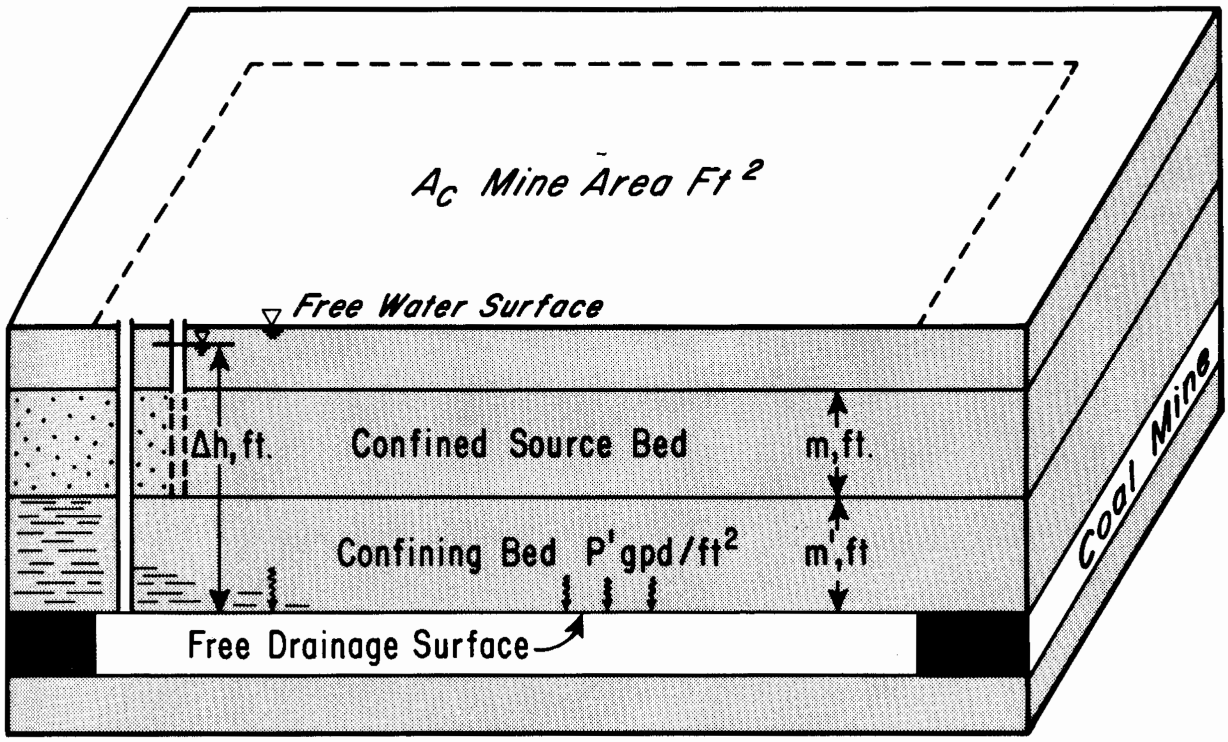
Ac = area of confining bed through which leakage occurs, in square feet.

Elements in equation 10 are depicted in Figure 3a for the case of a confined source bed and in Figure 3b for unconfined source beds.

Water level data may be collected for each aquifer unit and for coal beds to define piezometric surfaces and the water table or free water surfaces. Three or four multiple level piezometers may be installed in individual drill holes to record heads in various beds. Water level readings obtained within an open or uncased borehole penetrating more than one bed may be useless for this purpose as there may have been a 10 to 100 foot head difference between two adjacent beds before drilling.

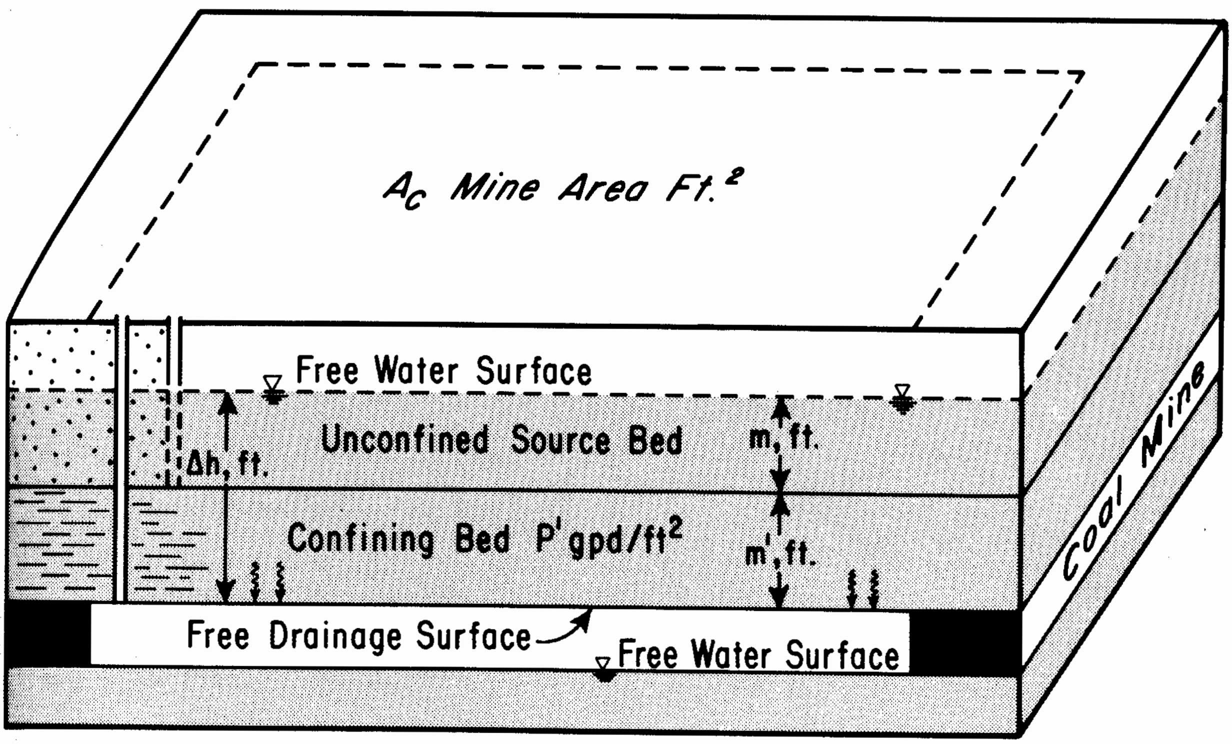
The success of the dewatering program can be determined by routinely recording water level changes within these piezometers once dewatering and mining begins. Continuous water stage recorders with monthly charts are ideal for this purpose. Water level data should be collected before and during mining to determine the response of pumping, changes in

Figure 3. Block diagram showing terms in equation 10 used to calculate vertical leakage into deep mines. Horizontal flow of ground water through the coal bed is neglected. Fig. 3a is for confined source beds and 3b is for unconfined source beds.



A_c Mine Area Ft ^{2}
Free Water Surface
Δh, ft.
Confined Source Bed
m, ft.
Confining Bed P' gpd/ft^2
m', ft
Free Drainage Surface
Coal Mine

(a)



A_c Mine Area Ft.^{2}
Free Water Surface
Unconfined Source Bed
Δh, ft.
m, ft.
Confining Bed P' gpd/ft^2
m', ft.
Free Drainage Surface
Free Water Surface
Coal Mine

(b)

precipitation, etc. Roofs of mines serve as free drains during mining, hence, mean head differences may be determined by comparing mine roof levels with piezometric levels in the first overlying and/or underlying source bed depending upon the hydrogeologic setting. The same is true for gravity drained abandoned deep mines.

It may be desirable to control leakage into abandoned deep, flooded mines as well. The piezometric level on the mine pool (this may be irregular due to roof collapse) and the mean level within the source bed would be used to compute the mean hydraulic head in this case.

Pumpage: Variations in the proposed distribution of pumpage with time and by location are also used to compute the potential water level declines that can be achieved by dewatering. The number, spacing, depth, and diameter of wells combined with pipeline, pump, powerline, pumping and maintenance costs are calculated to determine the cost and feasibility of the various dewatering schemes. Careful pumping records are required in advance of and during mining to determine the quantity of water pumped and resulting water level changes. The cost differential for dewatering by wells and by pumping directly from the mine would be less than for the case where mine waters are free to discharge by gravity. However, the largest single benefit in the absence of a market for the water is the reductions in treatment costs which vary with the seriousness of the pollution problem and quality standards that must be met.

Data From Existing Mines: Valuable data on vertical permeabilities, leakage rate as a function of mining method (longwall continous mining versus room and pillar), roofrock type and thickness, age of mine, etc.

can be obtained from operating mines, or abandoned mines. Simple field testing procedures can be devised to compute values of the in place coefficient of vertical permeability of roofrock. This is the best scale and conditions under which to assess flow behavior.

The average coefficient of vertical permeability of roofrock in a mine can be determined using equation 10 expressed as:

$$P' = Q_c M'/\Delta h \; A_c \qquad [11]$$

Where:

P' = coefficient of vertical permeability of confining bed, in gpd/sq. ft.

Q_c = leakage through confining bed through which leakage occurs, in gpd.

A_c = area of confining bed through which leakage occurs, in sq. ft.

Δh = difference between the head in the aquifer and in the source bed above the confining bed in ft.

A more convenient form of equation 11 is given as:

$$P' = Q_c M'/[\Delta h \; A_c \; (2.8 \times 10^7)] \qquad [12]$$

Where:

Q_c/A_c = leakage rate, in gpd/sq. mile.

Q_c = leakage through roofrock in gpd.

A_c = mine roof area through which leakage is occuring in sq. mi.

And where P', M' and Δh as previously defined.

The quantity of leakage (Q_c) through the confining bed into the mine is essentially the volume of water pumped or discharged from the mine in gallons per day, M' is the average thickness of confining bed above the mine roof or below the floor depending upon the direction of ground-water flow, A_c is the area of the existing mine to which water is being divert-ed and h is the difference between the average head in the first source bed above the mine and the mine roof elevation which is taken as a free drainage surface, (Fig. 3a and b).

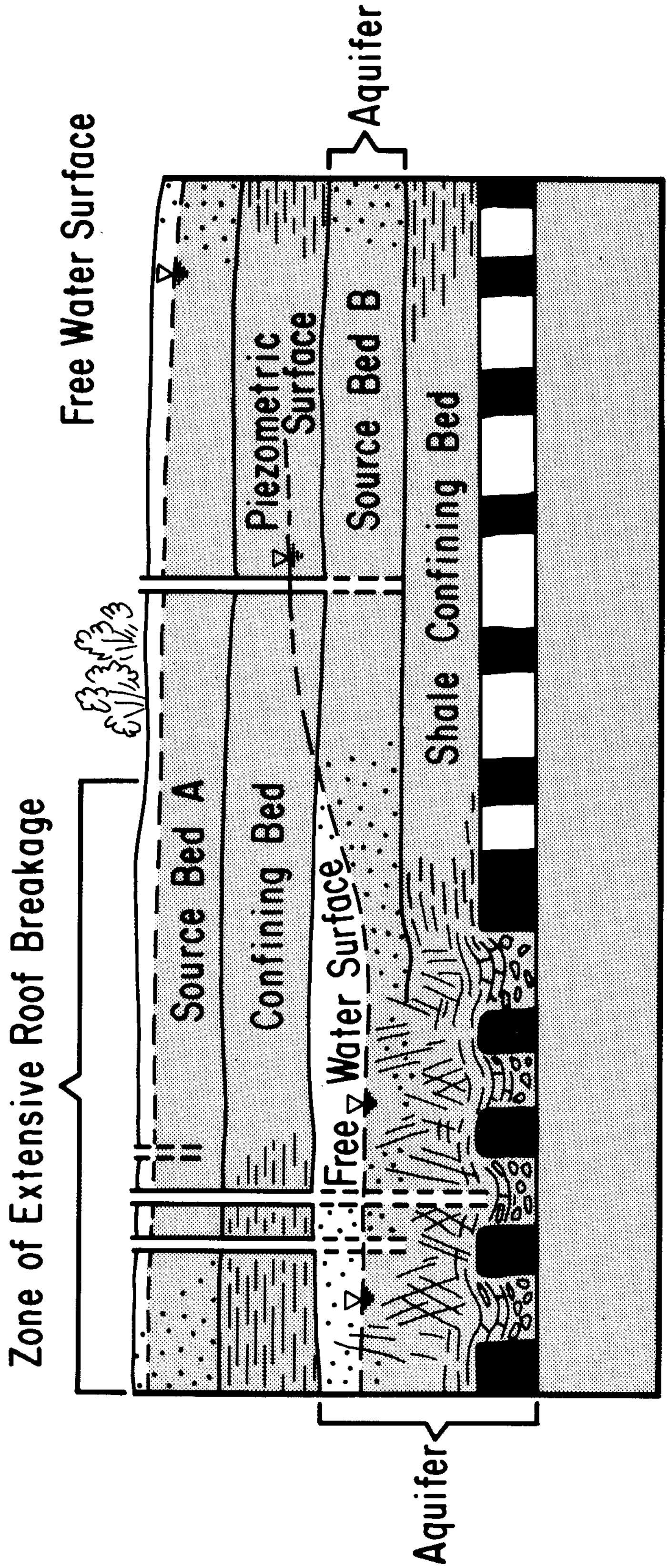
It should be apparent that P' and M' are not likely to remain constant during mining. Rather, roof breakage and subsidence is likely to greatly increase the value of the coefficient of vertical permeability over that of the original value, (Fig. 4). It should be a maximum in old mines where roof breakage is complete and a minimum in newly opened mines. Examples may be cited where overlying source beds have been dewatered by flow into mines after confining beds were intensly fractured, (Fig. 5). Longwall mining should increase the average value of P' but probably not as extensively as when the room and the pillar method of mining is used depending upon rock strength, room and pillar geometry, etc. This fact should be considered in designing the mining operation where extensive mine water pollution is anticipated. The hope is to keep the volume of polluted water to a minimum. The hydrogeologic setting at the mine under study can be compared with the proposed mine and an analogy established. First approximation estimates can be made of the potential water problem likely to be encountered and potential benefits likely to be derived from mine dewatering schemes.

WELL LOCATIONS

Dewatering wells may be equipped with pumps and serviced above ground or be gravity wells which pump themselves by draining into pipe-lines located within the mine or to diversion channels and other open drainage facilities, (Fig. 6). In the latter case water control may be provided to improve working conditions, rather than to minimize pollution. Gravity wells would be ideal in gravity drained mines because pumping

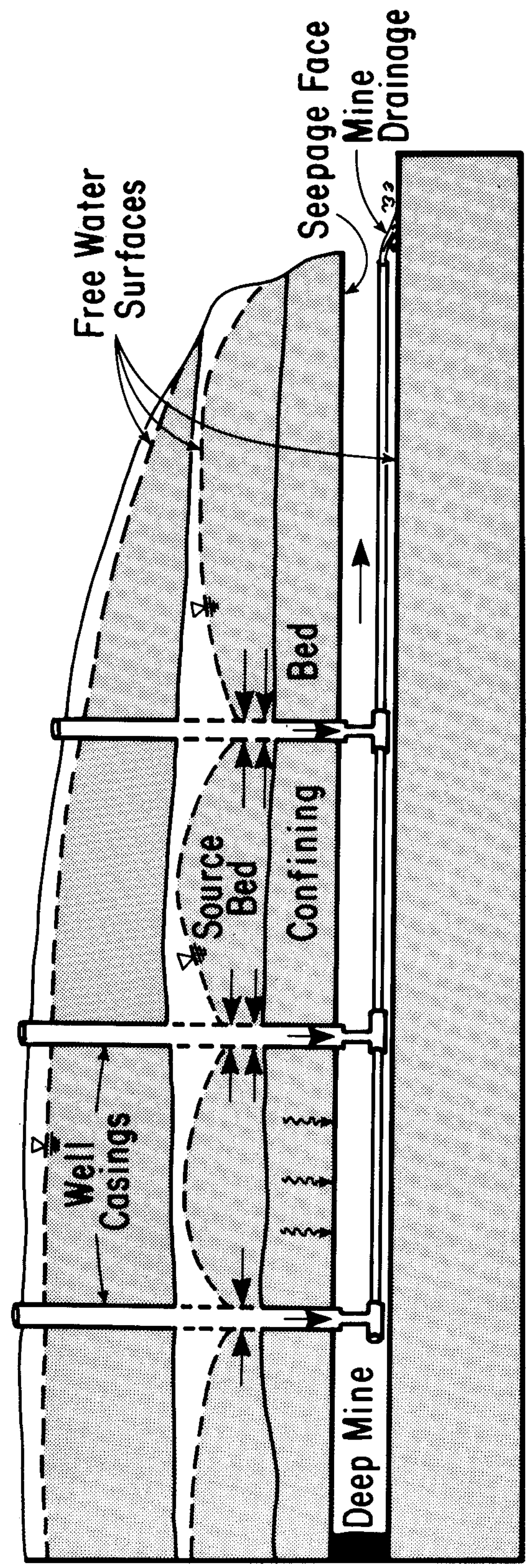
Figure 4. Roof collapse above an abandoned deep mine. Note that pillars of coal are still visible marking the original mine location. Siltstones and shales above the coal are highly brecciated and cease to serve as confining beds.

Figure 5. Zone of roof collapse above a deep mine. Former confining bed is now part of the aquifer (source bed B) which has a relatively high coefficient of permeability and contains a free water surface.



Zone of Extensive Roof Breakage
Free Water Surface
Source Bed A
Confining Bed
Free Water Surface
Piezometric Surface
Source Bed B
Shale Confining Bed
Aquifer
Aquifer

Figure 6. Gravity drainage wells used to dewater source beds above a deep mine. Water wells are drilled from landsurface and cased above source beds. Water is conveyed from the mine along pipe lines to prevent its contact with wall rock. For abandoned mines, water may be pumped to surface at each well.


Free Water
Surfaces
Seepage Face
Mine
Drainage
Well
Casings
Source
Bed
Confining
Bed
Deep Mine

costs would be all but eliminated. Gravity drainage wells would not be as desirable in abandoned mines where roof collapse is likely to disrupt pipelines.

The density and distribution of wells would be determined from preliminary hydrologic analysis. As dewatering is attempted and mining progresses, additional wells may be drilled as required.

The most productive well sites no matter what the bedrock type, can be located at fracture trace intersections. These natural lineations seen on aerial photographs, (Fig. 7) have been shown to be underlain by zones of increased permeability and porosity by Lattman and Parizek (1964), Parizek (1969), and Siddiqui and Parizek (1969). They are generally less than a mile in length, and are underlain by relatively narrow (10 to 40 feet) zones of nearly vertical fracture concentration (Fig. 8). Zones of fracture concentration increase rock permeabilities by 1 to over 1,000 times compared to those of adjacent rock depending upon the rock type. These zones are abundant enough in their distribution to provide more well sites than would be required in dewatering schemes for a given region and occur in all coal mining districts. Fracture traces should be mapped to locate test well sites whenever maximum volumes of water are required at least risk using normal well pumps or gravity well systems. Zones of fracture concentration should serve as relatively efficient collectors of water draining intervening more massive blocks of rock.

DESIRABLE FIELD CONDITIONS FOR DEWATERING SCHEMES

Ideally it is desirable to have the least number of highly productive wells. This requires favorable aquifer conditions such as a thin, highly permeable sandstone directly or a few feet above a coal with a relatively

Figure 7. Examples of fracture traces for an area underlain by coal, underclay, siltstone and sandstone. Fracture trace intersections are underlain by intersecting zones of nearly vertical fracture concentration which greatly increase rock permeabilities.

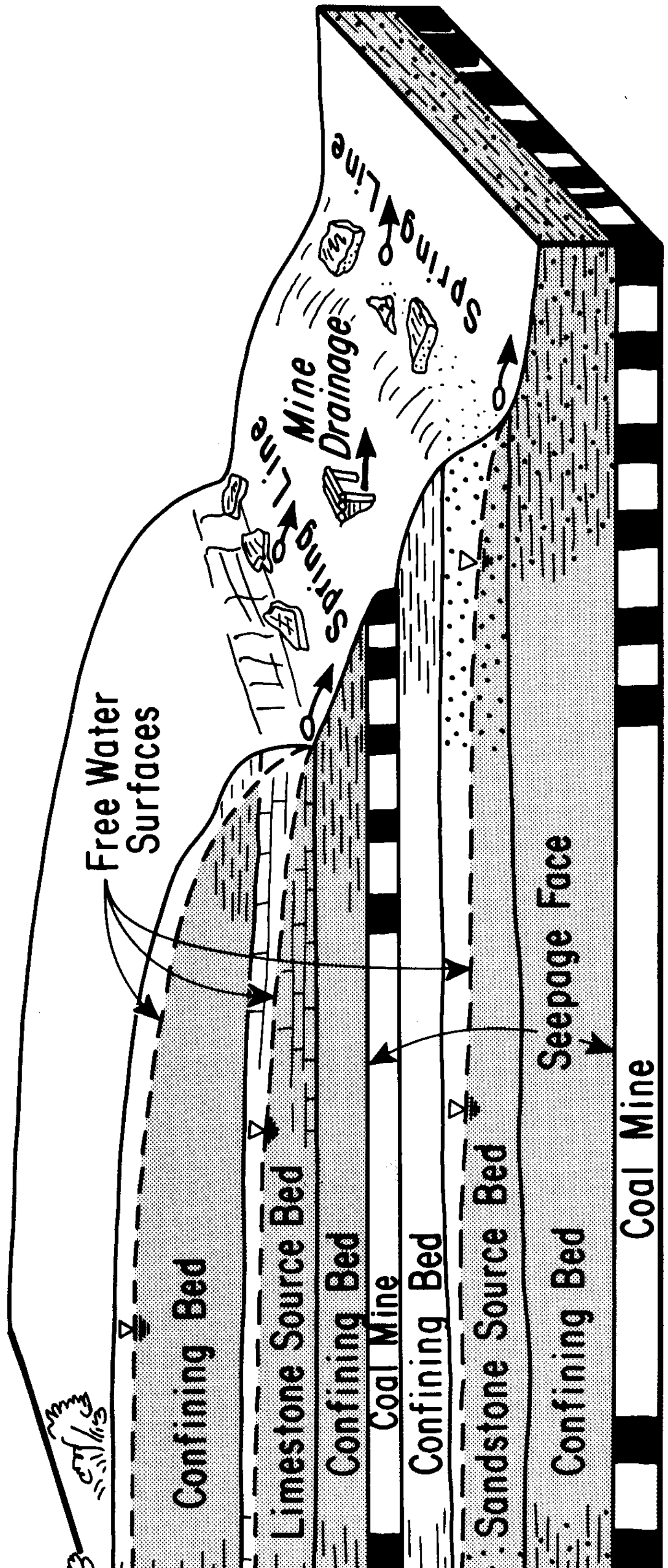
<u>Figure 8</u>. Zone of fracture concentration underlying a fracture trace developed on Devonian aged siltstones in northeastern Pennsylvania. The waterfall is localized along a zone of intense jointing compared to adjacent strata. The zone is nearly vertical and 39 feet wide.

high hydraulic head difference between the source bed and coal. It is unlikely that shale and most siltstone beds could be treated as aquifers and dewatered successfully because many low capacity wells would be required to reduce hydraulic heads compared to the case where more permeable beds are present.

Thin limestone beds may also be desirable provided that water-yielding openings can be located rather consistently. Massive, thick channel sandstones, deltaic or other extensive sandstone bodies may be highly productive aquifers and costly to dewater. If extensively exposed at landsurface, recharge rates may be excessive making dewatering on a sustained basis expensive. Also, dewatering may not be feasible where extensive pumping is required to reduce heads in the source bed even a few feet because leakage rates may still be accessive. Ideally, other productive aquifers should overlie the source bed to be dewatered which serves as the local ground-water supply for farm and domestic purposes. This would minimize damage to existing water supplies unless the water pumped by dewatering operations is marketed as a regional source of water to landowners located above the mine. Such a scheme of water sale would help to reduce costs of the operation and provide side benefits. This is critical where pumping is to be continued indefinitely after a mine is abandoned.

The horizontal permeability contrast between sandstone and some limestone beds and siltstone, shales and underclays normally is such that the vertical leakage rate through confining beds is far less than the horizontal flow rate. Where topographic relief is relatively high and aquifers are exposed, ground water may be freely discharged to surface and confined aquifers partly dewatered. (Fig. 9). The extent

Figure 9. Multiple free-water surfaces where ground water is free to drain to landsurface from highly permeable aquifers and where confining beds restrict vertical ground-water flow.


Free Water Surfaces
Spring Line
Mine Drainage
Spring Line
Seepage Face
Confining Bed
Limestone Source Bed
Confining Bed
Coal Mine
Confining Bed
Sandstone Source Bed
Confining Bed
Coal Mine

of dewatering back from outcrops can vary considerably depending upon the local field setting. James Urban, hydrologist with the Northeast Water Research Center, U.S. Dept of Agriculture, (1970, personal communications), reported that multiple free water surfaces are present in some localities in Ohio where more than one aquifer is exposed. Where this condition obtains and aquifers are thin, aquifer dewatering attempts to control leakage to deep mines may be foolhardy. The magnitude of excessive heads within a confined aquifer should be defined during the hydrogeologic exploration program to establish this or other conditions. To significantly influence vertical leakage from source beds under water-table conditions, the available drawdown within the aquifer must be more than three or four feet as calculations using equation 8 will reveal for dewatering to be successful.

HYDROLOGIC SYSTEMS ANALYSIS

Analytical Methods: Under simple hydrogeologic settings, conventional analytical methods can be used to predict water level and piezometric surface declines within aquifers that result when one or more wells are pumped at a known rate for a known duration at specified sites. Where confined aquifers serve as a source bed to an overlying or underlying deep mine, leaky artesian aquifer theory may be used (Hantush and Jacob, 1955, p. 95-100; Hantush, 1956; and Jacob, 1946; Neuman and Witherspoon, 1968, 1969). Where source beds are unconfined, the nonleaky artesian formula developed by Theis (1935) can be applied to aquifer test data under certain limiting conditions. Boulton (1954) specified the condition under which the nonleaky artesian formula may be expected to give a good approximation of the drawdown in a well under water-table conditions, and more recently Prickett (1965) presented a graphical solution that allows water level declines to be estimated for unconfined aquifers that accounts for slow gravity drainage of stored water. Theoretical vertical leakage rates into a deep mine can be computed using equation 1 to determine if significant reductions in leakage can be achieved. This may be done once an analysis has been made to determine the amount of water level decline that might be brought about by one or more pumping wells.

Costs of treating mine waters can be compared with additional costs of dewatering programs to see if a savings can be achieved. Although leakage volumes can be reduced by dewatering schemes, it is unlikely that mine waters can be eliminated entirely except for the case where source beds underlie the mine. Horizontal flow into deep mines along coal beds would have to be computed in addition to vertical leakage, because most coal beds serve as aquifers when compared to shale, siltstone and underclay. This water most likely would have to be collected and treated.

Electrical Analog Models: Under more complex field conditions, i.e., where aquifer storage and transmission properties, thickness, boundary conditions, vertical permeability distribution, and thickness of confining beds are highly variable, other system analysis procedures are required. Electrical analog simulation techniques are ideally suited to handle these field complexities provided that they can be defined accurately. This systems analysis procedure has received increased attention by ground-water hydrologists, since about 1964. Equations governing the flows of electricity in electrical circuts (Ohms Law and related equations) are considered identical to the equations governing the flow of water in a porous medium (Darcy's Law), Skibitzke (1961). Although an aquifer and its confining beds are a continuous phenomena, while resistor-capacitor networks used to simulate rock hydraulic properties, consist of many discrete branches, it can be shown mathematically that if the grid size is small in comparison to the size of the aquifer, the behavior of the network describes very closely the response of an aquifer to pumping.

A comparison of finite difference form of the partial differential equation governing the nonsteady-state two dimensional flow of ground water in an infinitely wide porous media, equation 13, (Stallman, 1956) with the same form of equation governing the flow of electrical current in a resistor-capacitor network, equation 14, (Millman and Seely, 1941), reveals that there is a direct analogy between the two equations.

$$T\ (\Sigma_2^5 hi - 4h_1) = a^2\ S\ (\partial h/\partial t) \qquad [13]$$

Where:

h_1 = head at node 1, hi (i = 2,3,4, and 5) = heads at surrounding nodes 2 to 5

a = width of grid interval (the aquifer is subdivided into small squares of equal area which have a common length of side, the grid interval)

T = Coefficient of transmissivity

S = the coefficient of storage

$$\frac{1}{R}\ (\Sigma_2^5 Vi - 4V_1) = C(\partial V/\partial t) \qquad [14]$$

Where:

V_1 - 5 = Electrical potential at each of resistors;
Ra - d = resistance of four resistors; C = capacitance; and Vi (i = 2,3,4, and 5) = electrical potential at ends of resistors a - d.

The analogy between equations 13 and 14 is apparent. Hydraulic heads, h, are analogous to electrical potentials V. The coefficient of transmissivity, T, is analogous to the reciprocal of the electrical resistance 1/R. The product of the coefficient of storage, S, and a^2 is analogous to the electrical capacitance C. Four scale factors are required to connect each unit in one system to the analogous unit in the other system (Bermes, 1960).

The resistor-capacitor network is analogous to the aquifer. Excitation-response equipment is required to excite the model to determine time-voltage changes within the model. A wave form generator, pulse generator and oscilloscope are used for this purpose. Time-voltage graphs, which are analogous to time-drawdown data in observation wells, are displayed on the oscilloscope, which shows the influence of various schemes of pumping.

One or more aquifers may be modeled with various interbedded confining beds to simulate a complexity of field conditions which can not be analyzed using analytical methods and image well theory. The design, construction and probing costs are small for these models compared to field data collection costs and, are in routine use by various federal and state agencies and by some consultants.

NUMERICAL METHODS

More recently finite difference and finite element schemes have been used to evaluate aquifer systems behavior. Iterative digital computer simulations and models have been described by Prickett and Lonnquist (1968); Remson, et al., (1965), Tyson and Weber (1964); Pindar and Bredehoeft (1968), Pindar (1970); and others.

Computer programs are rapidly becoming available which allow for solutions of a wide range of field problems. Source bed dewatering schemes can be tested using a general program recently developed by Pindar (1970) and calculations using equation 1. Pindar's program simulates the response of a confined or unconfined aquifer to pumping at a constant or variable rate from one or more wells. The ground-water reservoir may be irregular in shape and non-homogeneous with infiltration from one or more lakes and streams. Leakage into a source bed aquifer through a confining bed is considered analogous to leakage from beneath a lake or stream. The program is written in FORTRAN IV for the IBM 360 system and will record numerical values of time-drawdown data at each node and an alphameric contour map of the drawdown in the aquifer at selected time steps.

Field data input for numerical methods of simulation are identical for electrical analog models. A rectangular net or grid is superimposed on a plan view of the aquifer system to be analyzed for which field data have been compiled. The grid line intersections define nodes similar to the case for electrical analog models. At each node the transmissivity, storage coefficient and initial heads are recorded. The pumping rate is recorded at each node where a well is to be located and the hydraulic conductivities of stream or lake beds or confining beds are included. In Pindar's program, the elevation of the base of the aquifer is also recorded for the case of unconfined aquifers and hydraulic conductivity values replace the values for the coefficient of transmissivity which changes as the aquifer is dewatered. A parameter card provides dimensions of the grid, the head in the stream, thickness of stream, lake or confining beds, information concerning the maximum duration of pumping and other constants used in the computational scheme.

HYPOTHETICAL EXAMPLE

The stratigraphic sequence associated with coalbeds is complex. Channel sandstones, pointbar deposits, deltaic sequences, limestone, and related rock types show abrupt changes in distributions and thickness both vertically and horizontally as do less permeable siltstone, shale and underclay deposits. For purposes of illustration two hypothetical settings will be considered. Values for the coefficients of storage and vertical and horizontal permeabilities are poorly known for coal related strata in Pennsylvania, hence values were estimated based on a knowledge of lithology of rocks involved and preliminary aquifer test data collected by Mr. Robert Brown, Graduate Student, Department of Geology and Geophysics, The Pennsylvania State University.

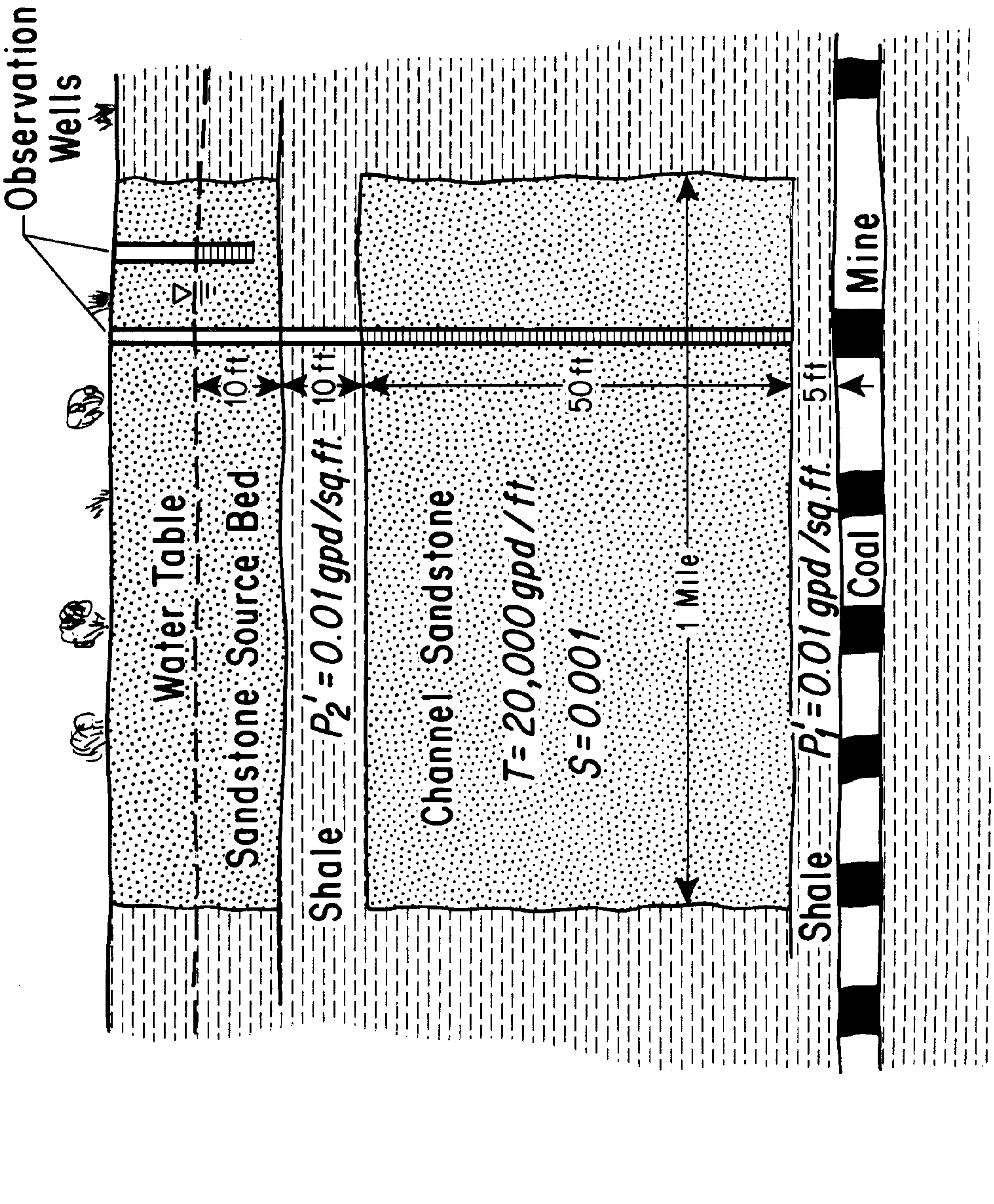
As a rule of thumb in the Indiana, Pennsylvania area the leakage rate to deep coal mines is assumed to be approximately 4.32×10^5 gpd/sq. mile. The vertical coefficient of permeability for confining beds in that area may approach average values from 0.015 to 0.0015 gpd/sq. ft. Given a confining bed 10 feet thick, a hydraulic head difference between the mine and the source bed of 20 feet, a coefficient of vertical permeability of 1.5×10^{-2} gpd/sq. ft., a leakage rate of 8.4×10^5 gpd/sq. mile is obtained using equation 12. By increasing the hydraulic head by 10 feet, $\Delta h = 30$ ft., the leakage rate is increased to 1.26×10^6 gpd/sq. mile and to 1.68×10^6 gpd/sq. mile for a case where $\Delta h = 40$ ft. An average reduction in the hydraulic head Δh, by 20 feet within a source bed above a deep mine initially containing a hydraulic head difference of 40 feet would reduce the leakage rate by approximately 8.4×10^5 gpd/sq. mile or it would reduce the total leakage rate by 4.2×10^6 gpd for a mine with a 5 square mile roof area. This hypothetical mine would still produce 8.4×10^5 gpd/sq. mile if the remaining average value for the hydraulic head was 20 feet and total leakage would approximate 4.2×10^6 gpd if the roof area was 5 sq. miles. The cost of pumping and treating 8.4×10^6 gpd of acid mine water would have to be compared with the cost of dewatering wells, pumping and treatment costs for the remaining 4.2×10^6 gpd to determine if a significant savings could be achieved during the life of the mine.

Consider an idealized case where a highly permeable channel sandstone just overlies the roof of a mine and has hydraulic properties and boundary conditions shown in Figure 10a and b. A 5 foot thick shaley sandstone underlies the channel sandstone and has a P' of 0.01 gpd/sq. ft.

Vertical leakage into a mine with a four square mile roof area (Fig. 11) would approach 3.22×10^5 gpd/sq. mile where 65 feet of shale overlies the mine and 4.2×10^6 gpd/sq. mile for the area beneath the channel sandstone for a combined total leakage of approximately 9.04×10^6 gpd. A reduction in the average hydraulic head Δh, by 20 feet within the channel sandstone by wells would bring about a 1.12×10^6 gpd/sq. mile reduction in the leakage rate to the mine and would reduce the total leakage to the mine to nearly 6.59×10^6 gpd. Greater reductions in leakage would be brought about by still further reductions in head.

The question arises, how many wells would be required to reduce the hydraulic head by 20 feet within source bed 2 given the vertical leakage from source bed 1 (Fig. 10a). Also, how many days of pumping would be required? The specific capacity (yield of a well in gallons per minute per foot of drawdown, gpm/ft.) of an efficient well drilled into an aquifer with a coefficient of storage of 0.001 and transmissivity of 20,000 gpd/ft. should approach 10 gpm/ft. Given an available drawdown

Figure 10. Hydrogeologic setting for a hypothetical deep coal mine (a) and the distribution of three and five dewatering wells (b).


Observation Wells
Water Table
Sandstone Source Bed
10 ft
Shale
P'2 = 0.01 gpd/sq.ft.
10 ft
Channel Sandstone
T = 20,000 gpd/ft.
S = 0.001
50 ft
1 Mile
Shale
P'1 = 0.01 gpd/sq.ft.
5 ft
Coal
Mine

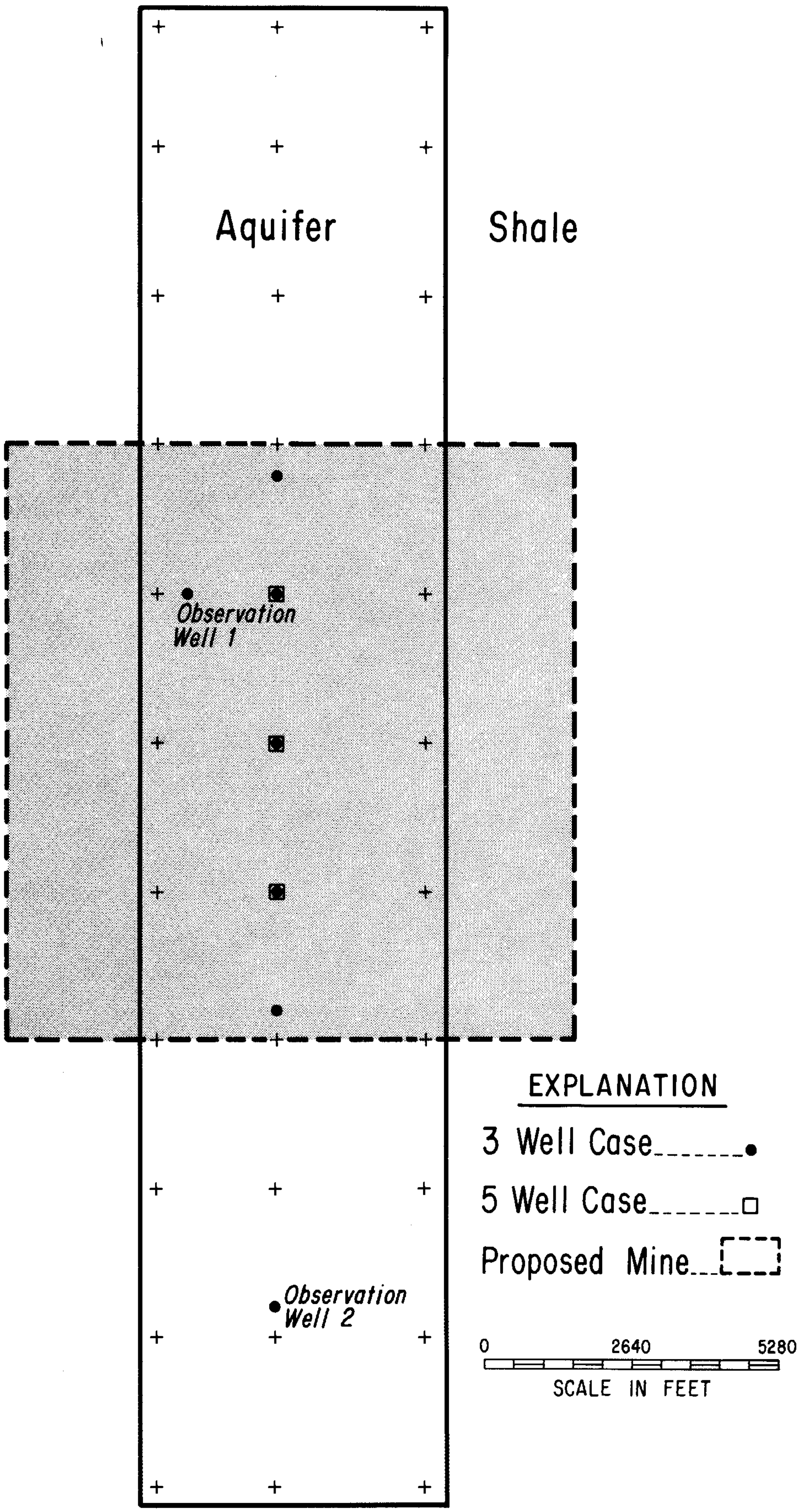
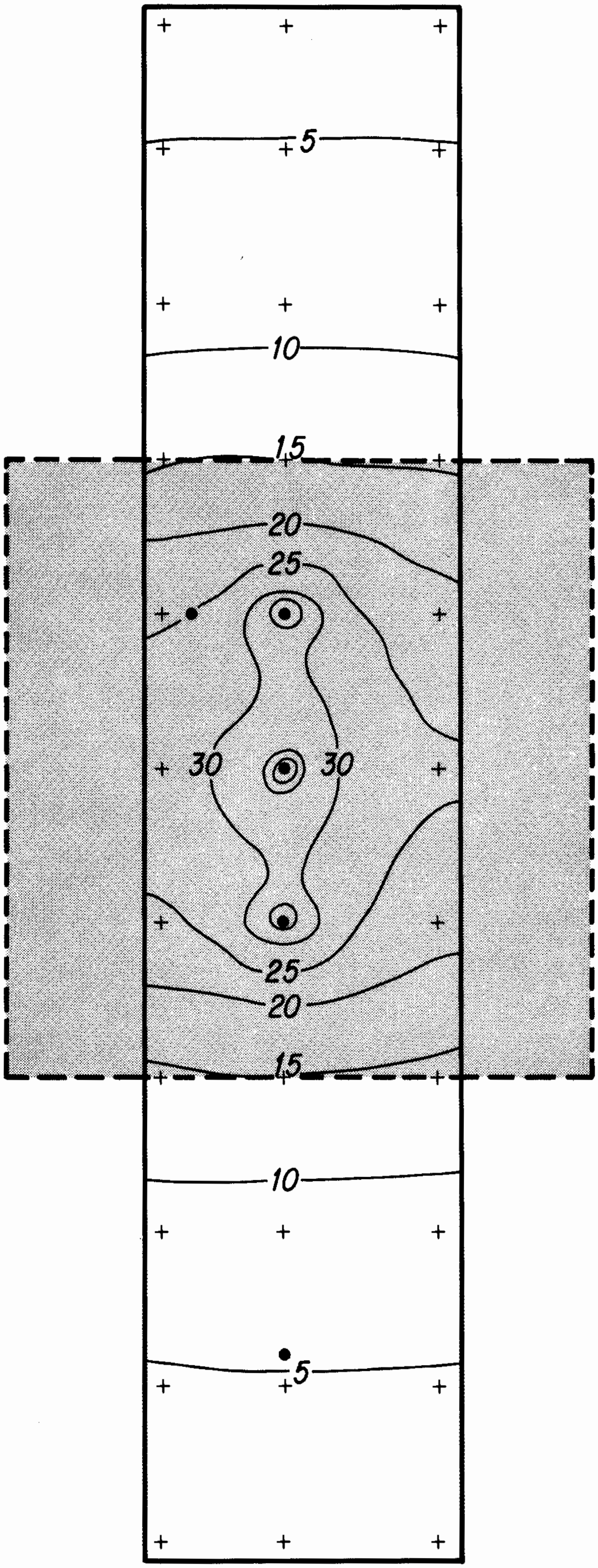
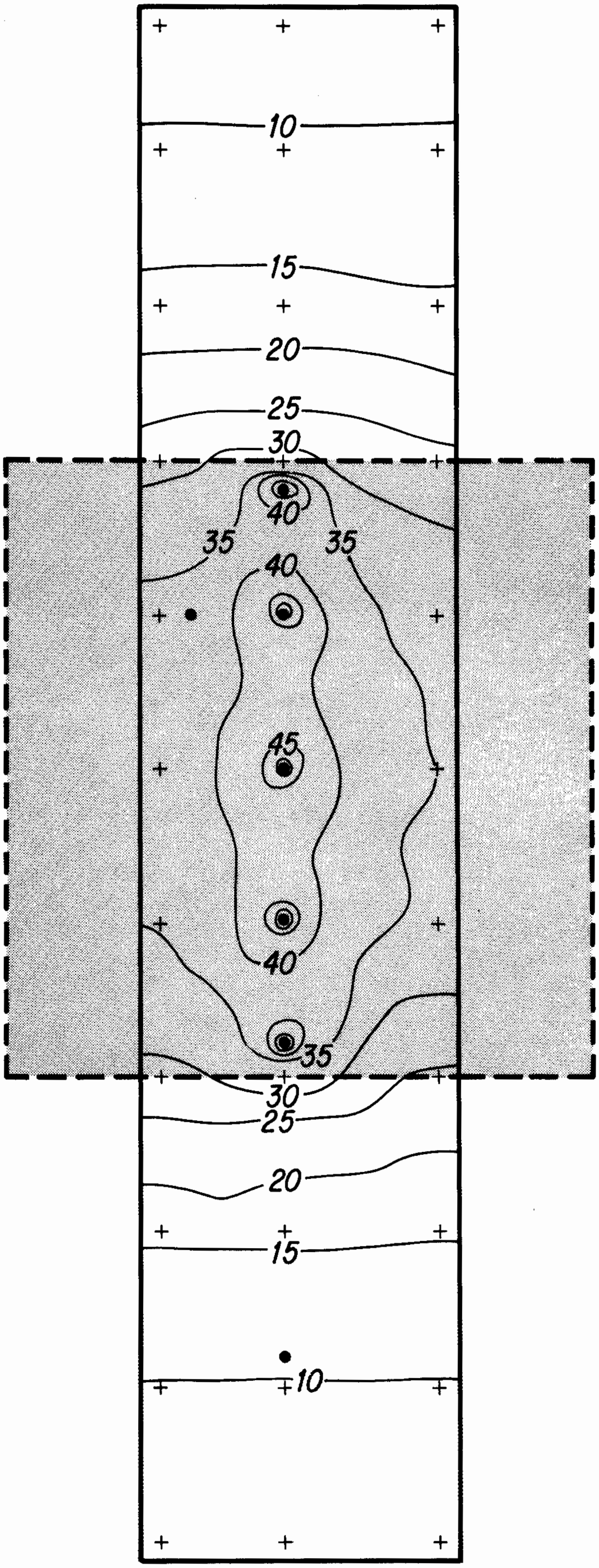

Aquifer
Shale
Observation Well 1
Observation Well 2
EXPLANATION
3 Well Case
5 Well Case
Proposed Mine
0
2640
5280
SCALE IN FEET

Figure 11. Composite cones of depression for a 3 well system after 180 days of continuous pumping (a) and for a 5 well system after 180 days (b).


5
10
15
20
25
30
30
25
20
15
10
5


10
15
20
25
30
40
35
35
40
45
40
35
30
25
20
15
10

of 50 feet, the upper 30 feet of the sandstone aquifer would be dewatered locally, a well should yield about 500 gpm. Three efficient 500 gpm wells given the spacing shown in Figure 10b should produce the composite cones of depression shown in Figure 11a after 180 days of pumping and a five well system (Fig. 10b) the cones of depression shown in Figure 11b after 180 days of continuous pumping. This and subsequent drawdown analyses were achieved with the aid of an unpublished program of a digitally simulated aquifer model kindly provided by Mr. T.A. Prickett and C.G. Lonnquist, of the Illinois State Water Survey, Urbana, Illinois.

The mean hydraulic head between the source bed and mine roof would approximate 51 feet after pumping the three well system for 180 days and 38 feet after 180 days for the five well system. Leakage rate beneath the channel sandstone would drop to approximately 2.13×10^6 gpd/sq. mile for the latter compared to an original value of 4.2×10^6 gpd/sq. mile. This figure could be reduced significantly by prolonged pumping or by adding additional wells to the dewatering systems.

Drilling and casing costs per well would depend upon a number of factors. At least a 100 foot deep, 10 inch well would be required with 30 feet of casing to insure that a trouble free 500 gallon a minute pumping rate could be maintained. It is reasonable to assume that a production well could be drilled for approximately $2,500 to $5,000 per well, particularly if a number of wells were to be drilled. Pipeline, power line, and a pump would add to this figure. Fortunately water could be discharged to creeks and small tributary drainages in close proximity to pumping wells, thereby eliminating pipeline costs. Induced streambed infiltration losses from most tributary channels underlain by silt and clay is minimal, hence the cost for pipelines and right of ways is eliminated or greatly reduced.

The model considered undoubtedly represents an extreme leakage case which could be encountered where permeable limestones, clean, well-sorted channel sandstones or glacial sand and gravel overlies the mine roof.

A more representative case for mine leakage beneath a channel sandstone might be given. Consider a potential coal mine overlain by 10 feet of shale with a coefficient of vertical permeability, P' of 0.001 gpd/sq. ft., an overlying sandstone source bed 20 feet thick which in turn is overlain by a shale confining bed with a P' of 0.001 gpd/sq. ft. The source bed or aquifer is assumed to have a coefficient of storage of 0.001, (Fig. 12). As is the case of the first model, the channel sandstone is assumed to be one mile wide, and five miles long (Fig. 10b). A coal mine with a roof area of 4 square miles is assumed, 2 square miles of which are beneath the channel sandstone, and 2 square miles of which are beneath thicker shale on either side. Figure 12 shows that the available drawdown approaches 40 feet. Wells located at fracture trace intersections could produce on the order of 200 gpm each from the deeper source bed.

Three and five well dewatering schemes are again considered given a well spacing similar to that of the first model (Fig. 10b).

Cones of depression for the 3 and 5 well cases are shown in Figure 13a and b after only 120 days of continuous pumping. The average head decline within the lower source bed approaches 28.4 feet after 120 days of pumping for the 3 well case. The leakage rate before dewatering from the channel sandstone might approach 1.96×10^5 gpd/sq. mile and 1.16×10^5 gpd/sq. mile after dewatering for 120 days by 3 wells.

The 5 well case would bring about an average head loss within the

Figure 12. Hydrogeologic setting for a hypothetical deep coal mine. Distribution of three and five dewatering wells, are the same as in Figure 10b.

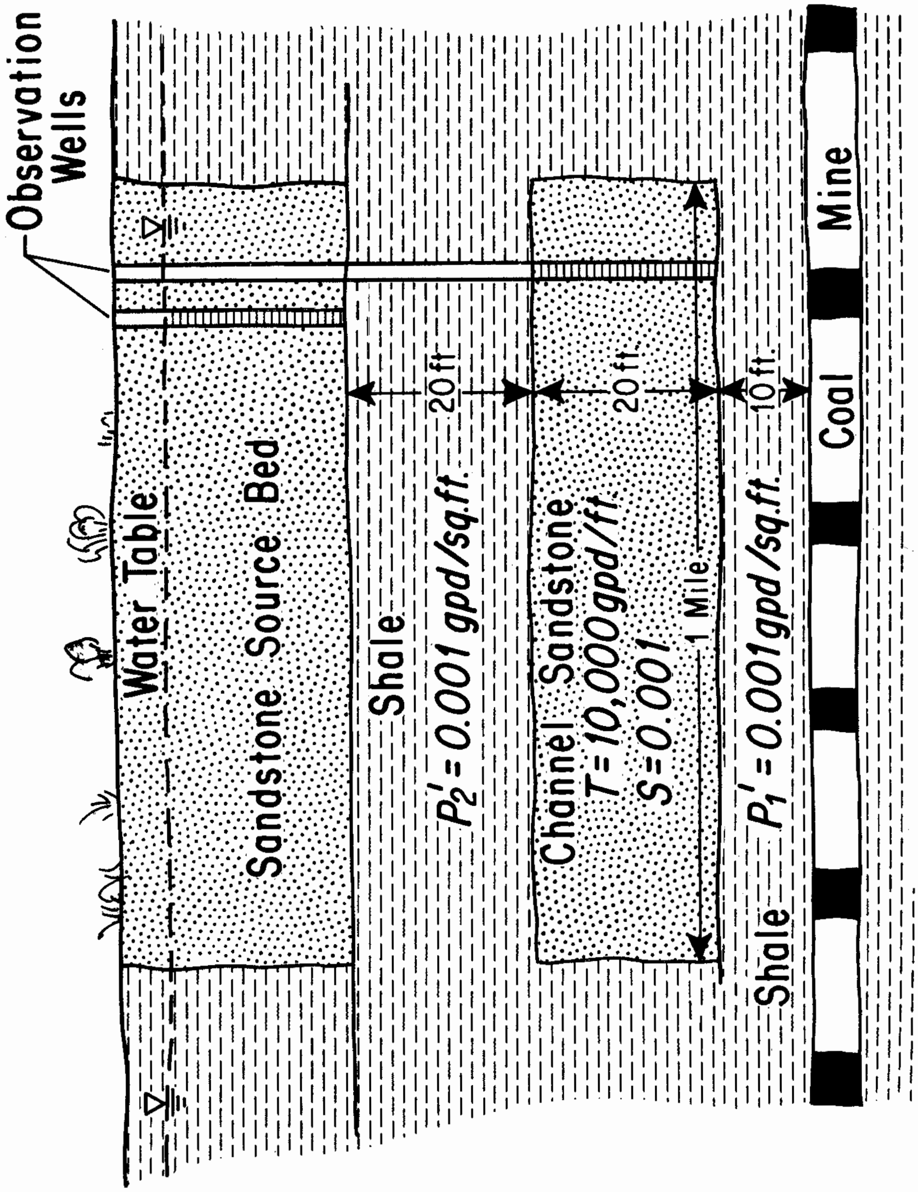
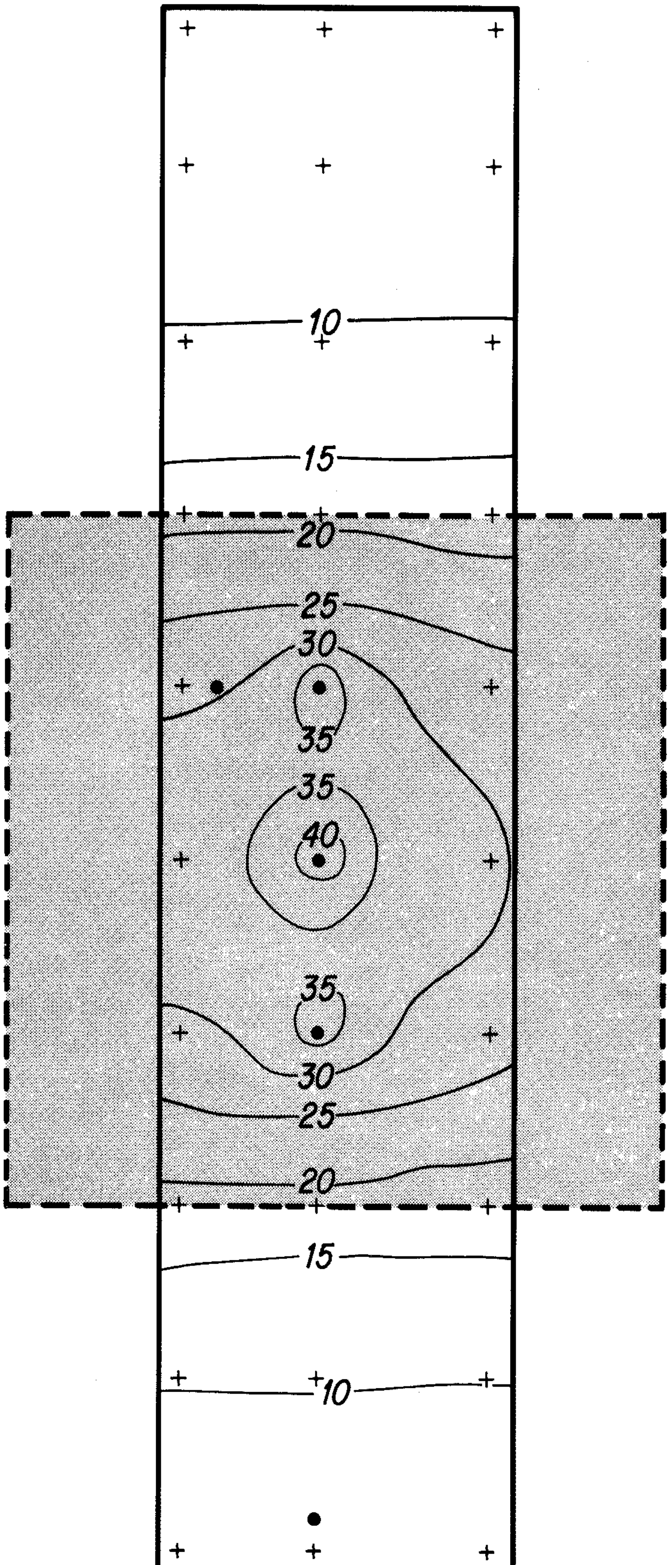
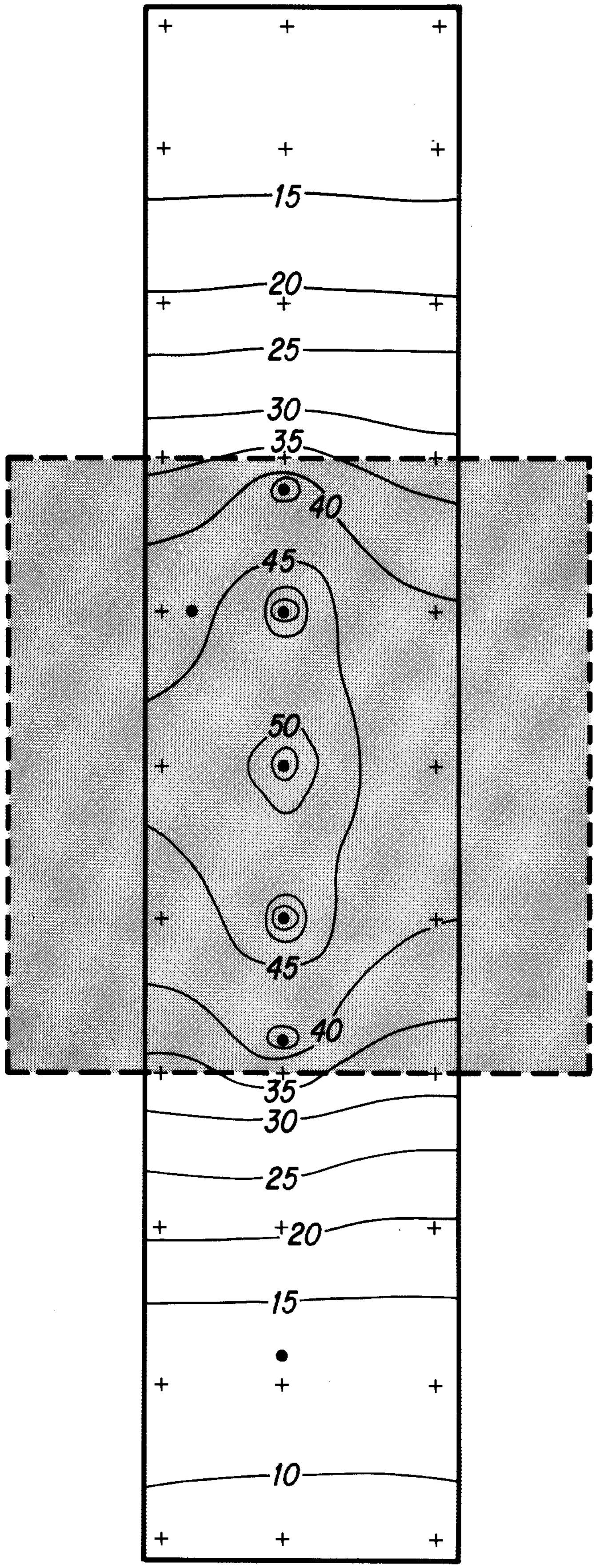

Observation Wells
Water Table
Sandstone Source Bed
Shale
20 ft
P'2 = 0.001 gpd/sq.ft.
Channel Sandstone
T = 10,000 gpd/ft
S = 0.001
20 ft
1 Mile
Shale
P'1 = 0.001 gpd/sq.ft.
10 ft
Coal
Mine

Figure 13. Composite cones of depression for a 3 well system after 120 days of continuous pumping at 200 gpm per well (a) and for a 5 well system after 120 days (b).


10
15
20
25
30
35
35
40
35
30
25
20
15
10
5


15
20
25
30
35
40
45
50
45
40
35
30
25
20
15
10

lower source bed of about 43 feet after 120 days of pumping and would dewater the upper part of the aquifer locally. Drawdown corrections for aquifer dewatering were not made in these analyses. The leakage rate from the channel sandstone might now approach 7.56×10^4 gpd/sq. mile. Figures 14 and 15 also show that the influence of pumping would have extended to the limits of the aquifer in all directions and that drawdowns from 7.8 to 34 feet might result within the lower aquifer on adjacent landowners property. Water rights of these landowners would have to be considered or water provided to them from the dewatering operation if dewatering is to be carried to an extreme.

Figure 14. Selected time-drawdown plots for two observation wells influenced by three and by five wells pumping at the rate of 500 gpm each. The aquifer model for this example is given in Figure 10a and b and the location of observation wells in Figure 10b.

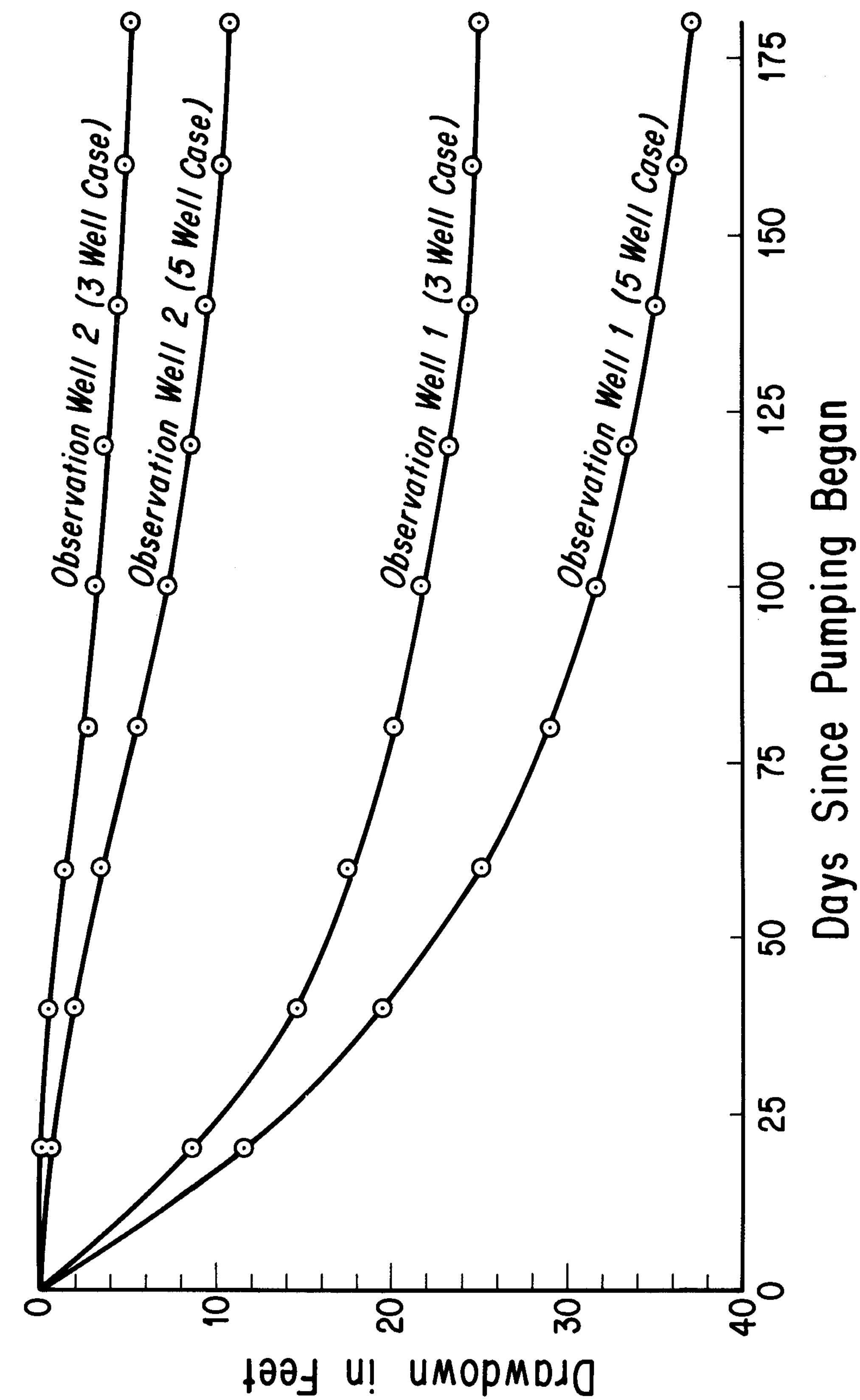
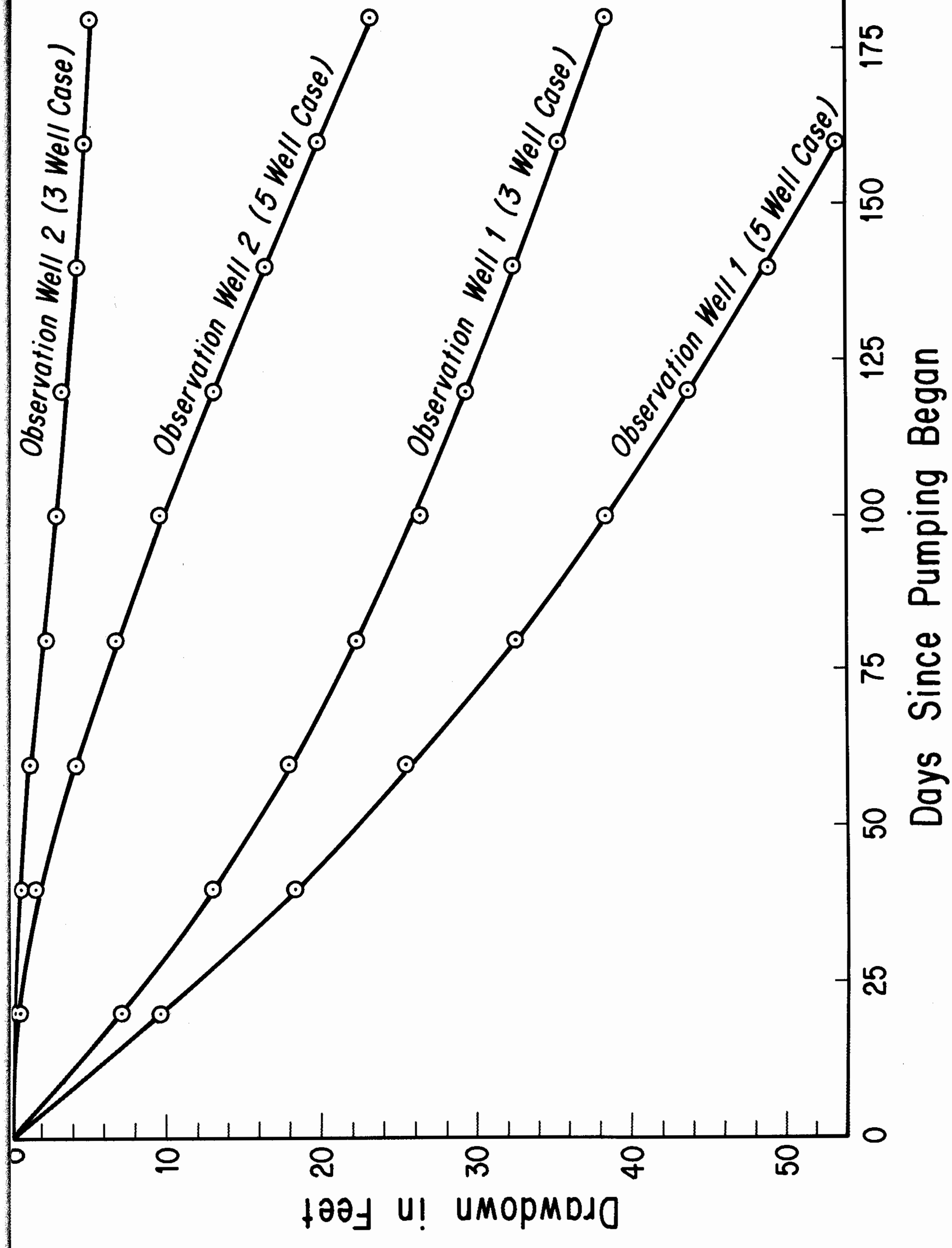

Observation Well 2 (3 Well Case)
Observation Well 2 (5 Well Case)
Observation Well 1 (3 Well Case)
Observation Well 1 (5 Well Case)
Days Since Pumping Began
0
25
50
75
100
125
150
175
Drawdown in Feet
0
10
20
30
40

Figure 15. Selected time-drawdown plots for two observation wells influenced by three and by five wells pumping at the rate of 200 gpm each. The aquifer model for this example is given in Figure 12a and b and the location of observation wells in Figure 10b.


Observation Well 2 (3 Well Case)
Observation Well 2 (5 Well Case)
Observation Well 1 (3 Well Case)
Observation Well 1 (5 Well Case)
Drawdown in Feet
0
10
20
30
40
50
Days Since Pumping Began
0
25
50
75
100
125
150
175

COST OF PUMPING GROUND WATER

The economic feasibility of dewatering deep coal mines to prevent mine drainage formation will depend upon a variety of factors some of which are hydrogeologic in nature; others are concerned with the cost of designing, drilling equipping and pumping high capacity water wells over a prolonged period. The cost of dewatering schemes should be compared to the cost of collecting and treating polluted waters that otherwise would have entered the mine. These costs are time dependent but for purposes of illustration hypothetical figures are presented to give order of magnitude estimates. Cost estimates of constructing and operating water wells were based on analyses presented by Seaber and Hollyday (1965).

The cost of designing dewatering wells is over and above the cost of the original coal and hydrogeologic exploration programs referred to earlier. Factors to be considered included number and spacing of wells; well depth and diameter; casing requirements; depth to static and pumping levels; pump working horsepower; test pumping, motor shaft, column, pump and strainer; fixed equipment costs; contingencies; engineering and administration costs. These represent initial costs. Annual costs considered by Seaber and Hollyday (1965) inlcude annual payments to retire initial costs, annual power, and maintenance costs.

Hypothetical wells capable of yielding 500 gpm are assumed to average 400 feet in depth, 8 inches in diameter, contain 40 feet of 8 inch diameter surface casing, have a 20 foot static water level and a 200 foot pumping level to achieve the desired drawdown.

One exploratory well (which could later be converted to an observation well) for every production well was assumed to be necessary by

Seaber and Hollyday to locate wells capable of yielding 500 gpm. If fracture trace intersection are used for well site location this is a conservative average. Seaber and Hollyday's 1964 average price for an 8 inch diameter well in shale is $1,800 and $200 for the casing. A 24 hour pumping test for aquifer analysis and designing the deep well turbine pumping unit was considered adequate and could cost $500. Motor, column, shaft, pump and strainer costs for a unit capable of delivering 500 gpm from a 200 foot pumping level might approach $4,500. Estimated cost of land and rights of way is $1,000 per well, $1,500 per pump house, wiring metering, piping, etc., $1,500 per well. Contingencies were taken as 10 per cent of the estimated equipment and construction costs and engineering and administration as 15 per cent of total construction costs and contingencies.

Annual costs were computed by Seaber and Hollyday (1965) as a single end-of-year payment to cover interest on the initial cost and payments to a depreciation fund. Annual power costs were based on a 24 hour a day use, and by assuming 75 per cent wire to motor efficiency. Annual maintenance costs were estimated as 4 per cent of the cost of equipment and the well's life was assumed to be 25 years, or the life of the mine. This assumes that the pump would never be replaced.

The total annual cost in dollars per million gallons of water a day delivered to the surface for wells with poor yields (75 gpm each) might approach $19,000 per year, $10,000 per year for wells with 230 gpm yields and $6,900 per year for wells with up to 760 gpm yields.

A more up to date estimate should be made in view of rapidly rising prices during the last 5 years. Fixed costs might have risen at a 3 to 4% rate annually, whereas, annual costs may have risen at a still faster rate. Based on a 1965 rate it is reasonable to assume that a dewatering system pumping 2.17 mgd may cost from $15,000 to $25,000 per year depending upon whether a 3 or 6 well system is required. A 4 per cent annual increase in prices, was allowed for to arrive at these figures.

COST OF TREATING COAL MINE DRAINAGE

Published figures on the cost of treating coal mine drainage are few and incomplete. Dr. Harold Lovell, Head of Mine Drainage Research Section at the Pennsylvania State University indicates that reported figures tend to underestimate these costs because taxes, depreciation, amortization, and maintenance figures, sludge disposal costs, etc., are not always included, hence the coal industry is paying more for treatment than they are credited for.

Costs are highly variable reflecting volumes of water treated, duration that mine waters are to be treated, variation in concentration in the pollution load, reagent costs, etc. Dr. Lovell points out that treatment costs should be on a per ton of coal produced basis to allow for meaningful comparison. For example, some mine operators may have a large volume of water to treat on a daily basis but when viewed on a tonnage of coal produced basis their problem is far less serious when compared to the mine operator with small production and equally large volumes of water.

Viewed on a long term basis and considering a variety of published data, Dr. Lovell indicates that treatment costs may range from $.05 to $1.25 per 1,000 gallons of mine drainage treated. The lower figure may be for mine waters with about 2 to 3 ppm of iron and up to 20 ppm of acidity, and the upper figure is for mine water with 500 to 700 ppm iron and 2,000 to 5,000 ppm acidity. An average figure today may be $.40 per 1,000 gallons for water with 100 ppm iron, 500 ppm acidity and using a hydrated lime treatment.

Dr. Lovell's research on mine drainage treatment should provide a more complete picture of actual costs to the industry for various treatment schemes using modular treatment units and a variety of concentrations of pollution constituents. He considers the above figures as conservative. It should be clear, nevertheless, that in the hypothetical example cited earlier a considerable savings may be achieved through mine dewatering. It is conceivable that 2.17 mgd may be pumped in a dewatering scheme at a cost of $15,000 to $25,000. per year. The same water may have been treated at a cost of $108.50 to $272.50 per day depending upon the concentration of pollution constituents using Dr. Lovell's maximum and minimum figures. The minimum cost of treating 2.17 mgd of mine water may have been $39,600 per year, and a maximum of $990,000 per year and $316,800 per year for mine waters of average composition. Dewatering may have achieved savings of between $14,600 to $965,000 per year by comparison for this hypothetical case.

Literature Cited

Bermes, B. S., 1960, An electric analog method for use in quantitative studies; U.S. Geol. Survey unpublished report.

Boulton, N. S., 1954, The drawdown of the water table under non-steady conditions near a pumped well in an unconfined formation; Proc. (British) Inst. Civil Engrs., Vol. 3, pt. 3.

Hantush, M. S., 1956, Analysis of data from pumping tests in leaky aquifers; Trans. Am. Geophysical Union Vol. 37 (6).

--------------, 1960, Modifications of the theory of leaky aquifers; Jour. Geophysical Res. Vol. 65 (11).

--------------, and C. E. Jacob, 1955, Non-steady radial flow in an infinite leaky aquifer; Trans. Am. Geophysical Union Vol. 36 (1).

Jacob, C. E., 1940, On the flow of water in an elastic artesian aquifer; Trans. Am. Geophysical Union, pt. 2.

--------------, 1946, Radial flow in a leaky artesian aquifer; Trans. Am. Geophysical Union, Vol. 27, (2) p. 198-205.

Lattman, L. H., and R. R. Parizek, 1964, Relationship between fracture traces and the occurrence of ground water in carbonate rocks; Jour. of Hydrology, Vol. 2, p. 73-96.

Millaman, J., and S. Seely, 1941, Electronics, McGraw Hill Book Co., New York.

Neuman, S. P., and P. A. Witherspoon, 1968, Theory of flow in aquicludes adjacent to slightly leaky aquifers; Water Resour. Res., Vol. 4, No. 1, p. 103.

--------------, and P. A. Witherspoon, 1969, Transient flow of ground water to wells in multiple-aquifer systems; Dept of Civil Engr., Inst. of Transportation and Traffic Engr., Pub. No. 69-1, Univ. of California, Berkeley, p. 1-182.

Parizek, R. R., 1969, Permeability development in folded and faulted carbonates, Pennsylvania; Trans. Am. Geophysical Union, Vol. 50 (4), p. 155.

Pindar, G. F., and J. Bredehoeft, 1968, Application of digital computer for aquifer evaluation; Water Resour. Res., Vol. 4, No. 5, p. 1069-1093.

--------------, 1970, An iterative digital model for aquifer evaluation; U. S. Geol. Survey open file report, p. 1-35.

Prickett, T. A., 1965, Type-curve solution to aquifer tests under water-table conditions, Ground Water, Vol. 3 (3), p. 1-10.

---------------, and C. G. Lonnquist, 1968, Comparison between analog and digital simulation techniques for aquifer evaluation; Int. Assoc. Scientific Hydrology; The use of Analog and Digital Computers in Hydrology, Symposium of Tuscon.

Remson, I., C. A. Appel, and R. A. Weber, 1965, Ground water models solved by digital computer; Journ. of Hydraulics Div. Proc. Am. Soc. Civil Engrs., Vol. 91, Hy 3.

Seaber, P. R., and E. F. Hollyday, 1965, An appraisal of the ground-water resources of the lower Susquehanna River basin (an interim report), U. S. Geol. Survey open file report, p. 1-75.

Siddiqui, S. H., and R. R. Parizek, 1969, Hydrogeologic factors influencing well-yields in folded and faulted carbonate rocks, central Pennsylvania; (Abstract) Trans. Am. Geophysical Union, Vol. 50 (4), p. 154.

Skibitzke, H. F., 1961, Electronic computers as an aid to the analysis of hydrologic problems; International Assoc. of Scientific Hydrology, Pub. 52.

Stallman, R. W., 1956, Numerical analysis of regional water levels to define aquifer hydrology; Trans. Am. Geophysical Union, Vol. 37.

Theis, C. V., 1935, The relationship between the lowering of a piezometric surface and the rate and duration of discharge of a well using ground-water storages; Trans. Am. Geophysical Union, 16th Ann. meeting, pt. 2.

Tyson, H. N., Jr., and E. M. Weber, 1964, Ground water management for the nations future-computer simulation of ground water basins; Journ. of Hydraulics Div. Proc. Am. Soc. Civil Engrs., Vol. 90, Hy 4.

Walton, W. C., 1960, Leaky artesian aquifer conditions in Illinois; Illinois State Water Survey Rept. of Invest. 39.

---------------, 1962, Selected analytical methods for well and aquifer evaluation, Bull. 49, Illinois State Water Survey, p. 1-81.