UNITED STATES

DEPARTMENT OF THE INTERIOR

GEOLOGICAL SURVEY

GEOLOGY OF THE CARBONDALE COAL MINING AREA, GARFIELD AND PITKIN COUNTIES, COLORADO, AS RELATED TO SUBSURFACE HYDRAULIC MINING POTENTIAL

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Prepared in cooperation with the U.S. Bureau of Mines

Open-File Report 80-709

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GEOLOGY OF THE CARBONDALE COAL MINING AREA, GARFIELD, PITKIN, AND GUNNISON COUNTIES, COLORADO, AS RELATED TO SUBSURFACE HYDRAULIC MINING POTENTIAL

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INTRODUCTION

Some of the coal fields of Washington and Colorado contain large underground deposits of bituminous coal in beds pitching from 15 to 50 degrees. It is difficult to recover such coal at competitive mining costs because gradients of that magnitude place increasingly sharp limitations on the use of conventional mining machines. However, some of those deposits of pitching coal may be recovered by subsurface hydraulic mining methods, in which the pitch of the coal bed can be used to advantage.

In 1977, the Bureau of Mines initiated a cooperative program of work with the Geological Survey, by selecting five geographic areas of pitching coal for which additional geologic information was needed for an adequate appraisal of their hydraulic mining potential. The Geological Survey was to prepare a geologic report on each of those areas, in support of Bureau research on the technology of subsurface hydraulic mining. In order of priority assigned by the Bureau, the five areas are identified as follows:

- Area 1: Coal deposits, Wilkeson-Carbonado coal field, vicinity of Carbonado, Pierce County, Washington.
- Area 2: Thompson Creek coal deposits, Carbondale coal field, 8 miles southwest of Carbondale, Pitkin County, Colorado.
- Area 3: Coal deposits, Carbondale coal field, from vicinity of town of Marble, north along trend of coal-bearing rocks, to a point about 4 miles southwest of Glenwood Springs, Gunnison, Pitkin, and Garfield Counties, Colorado.
- Area 4: Coal deposits, Grand Hogback coal field, from point about 4 miles southwest of Glenwood Springs, north along trend of coal-bearing rocks, to vicinity of town of Meeker, Garfield and Rio Blanco Counties, Colorado.

Area 5: Coal deposits, Crested Butte coal field, vicinity of town of Crested Butte, Gunnison County, Colorado.

The four Colorado areas are contiguous (see fig. 1); Area 2, the Thompson Creek coal deposits, is a component part of the Carbondale coal field (Area 3). In compliance with the Bureau's request, however, a separate report was prepared for Area 2 (see Kent and Arndt, 1980)

This report is the second of a series of four reports on the four Colorado areas selected. The study area of this report is herein defined as the "Carbondale coal mining area", and Area 2 is included as the central part (see figs. 1 and 2).

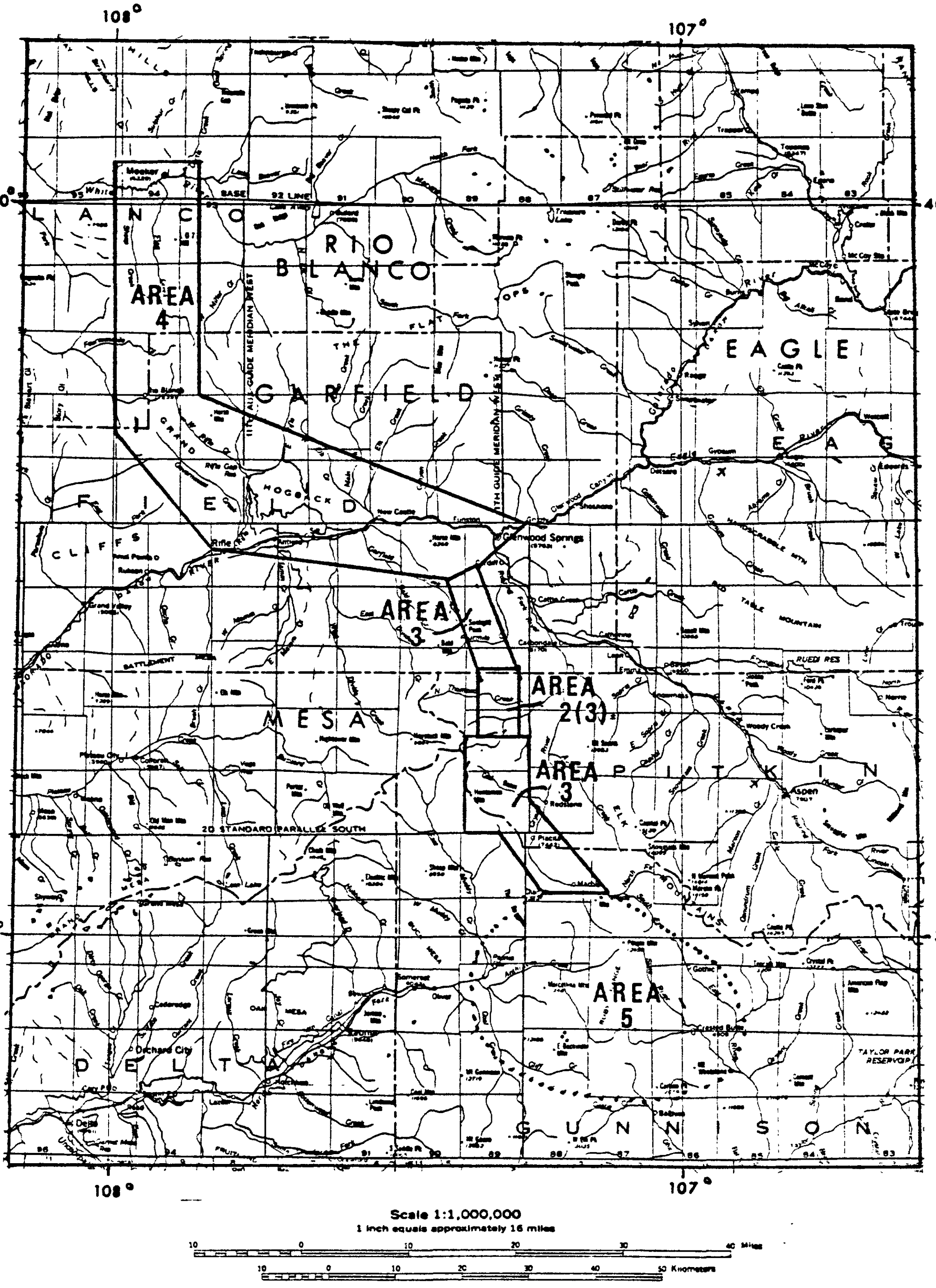

Figure 1.--Index map of part of western Colorado, showing locations of Areas 2, 3, 4, and 5. Area 2 is a component part of Area 3.

CARBONDALE COAL MINING AREA

Location

As defined here, the Carbondale coal mining area is a linear north-south trending area of about 104 square miles in Garfield, Pitkin, and Gunnison Counties, Colorado. The mining area is divided into northern, central, and southern parts (see fig. 2). The geologic maps also show the outline and division of the area (see figs. 5, 6 and 7).

Northern part

The northern part, an area of about 30 square miles, extends from secs. 4-5, T. 7 S., R. 89 W., in the Cattle Creek quadrangle (fig. 5), southward into secs. 21-23, T. 8 S., R. 89 W., in the Stony Ridge quadrangle (fig. 6). As a point of reference for location purposes, the "Sunlight Mine" on Fourmile Creek in sec. 34, T. 7 S., R. 89 W. (see fig. 5) is about 6 miles west of Carbondale, Colorado, and about 10 miles south of Glenwood Springs (see fig. 2).

Central part

The central part (i.e., the Thompson Creek coal mining area) is a rectangular area of about 14 square miles located within the Stony Ridge quadrangle (see fig. 6). The "Thompson Creek Mine" sites on North Thompson Creek in secs. 34-35, T. 8 S., R. 89 W. (see fig. 6) are about 8 miles southwest of Carbondale, and about 16 miles south of Glenwood Springs (see fig. 2).

Southern part

The southern part, an area of about 60 square miles, extends from secs. 13-16, T. 9 S., R. 89 W. in the Stony Ridge quadrangle, south and southeast to secs. 5-6, T. 11 S., R. 88 W. in the southeast corner of the Placita quadrangle (see fig. 7). The "Dutch Creek Mine" in sec. 17, T. 10 S., R. 89 W. (see fig. 7) is about 6 miles west of Redstone, Colorado, and about 25 miles south of Glenwood Springs (see fig. 2).

Surface-minerals land ownership

Surface and minerals land ownership in the Carbondale mining area and vicinity is indicated by patterns on the surface-minerals management map (see fig. 2).

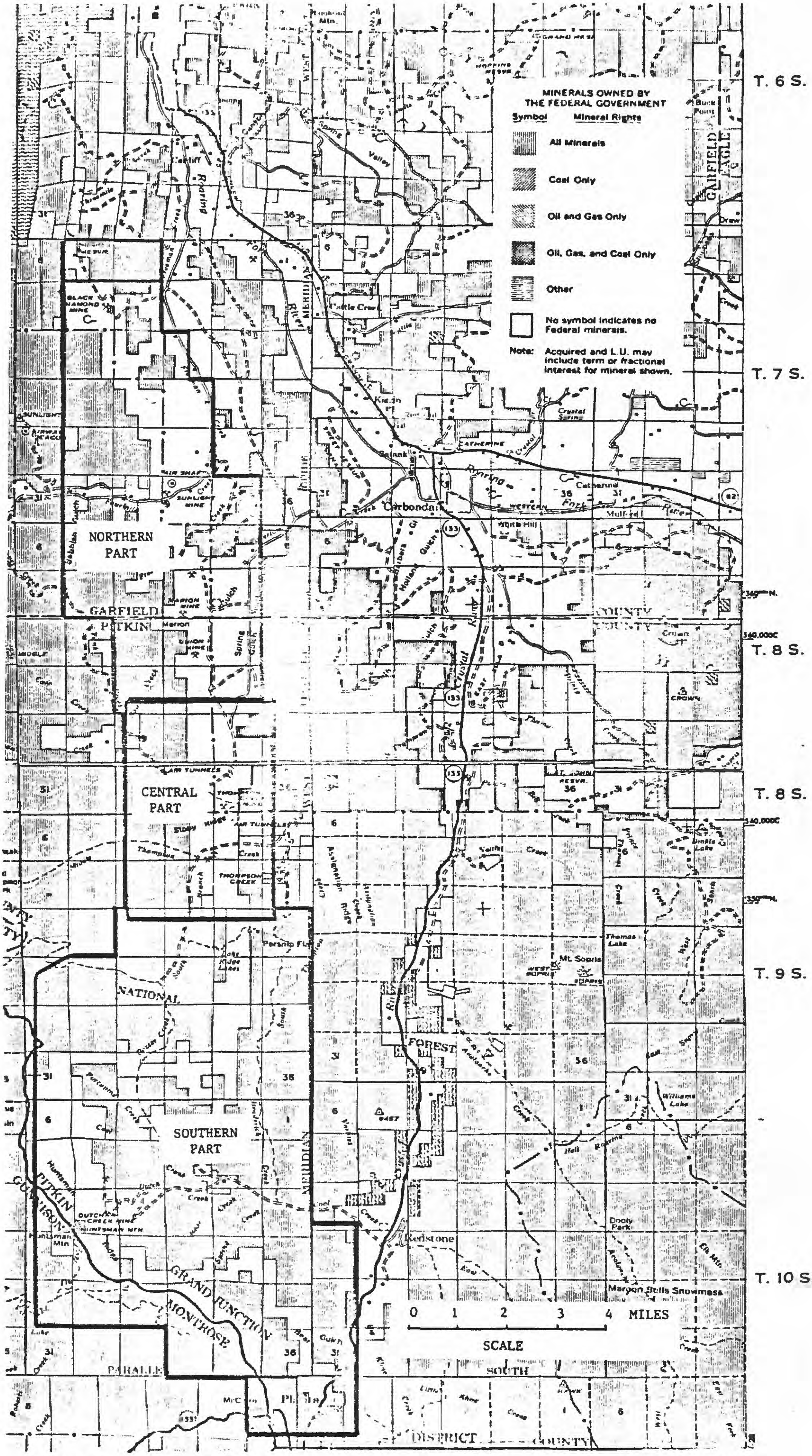

Figure 2.--Surface-minerals management map of Carbondale coal mining area and vicinity, assembled from U.S. Bureau of Land Management Surface-Minerals Management Quad NW-21 (Glenwood Springs, March, 1975) and Quad NW-27 (Paonia Resevoir, 1974).

Sections and parts of sections that are not patterned on the map (fig. 2) are patented lands (i.e., privately owned). Patented lands within the outline of the Carbondale mining area trend north-south and contain the belt of exposed coal-bearing rocks, and locations of coal mines are shown along that north-south trend; presumably, those lands are owned by the coal companies involved.

As of July, 1977, active mines within the Carbondale mining area included the "Sunlight Mine" in the northern part, the "Thompson Creek Mine(s)" in the central part, and the "Dutch Creek Mine(s)" in the southern part (see fig. 2).

Access

Northern part

Vehicle access to the northern half of the northern part of the Carbondale coal mining area is via an all-weather county road from Glenwood Springs (see figs. 2 and 5). Road-distance from Glenwood Springs to the Sunlight Mine is about 10 miles. A network of dirt roads between the Sunlight Mine and the "Marion Mine" to the south is shown on figure 2. As of 1977, however, those roads are not passable--even for 4-wheel drive vehicles.

Access from Glenwood Springs to the southern half of the northern part is via a road east of, and parallel to the Freeman Creek (Fourmile Creek) road (see figs. 2 and 5); this road joins the Edgerton Creek road in sec. 36, T. 7 S., R. 89 W.

Central part

Vehicle access to the central part (i.e., the Thompson Creek mining area) is via a light-duty road from Carbondale, Colorado. The route is shown on figure 2. Road distance from Carbondale to the mine sites on North Thompson Creek is about 10 miles. The bulk of the route is within incompetent shales of the Mancos Shale and Iles Formation (see fig. 5). Road maintenance for high-volume coal transport by truck is expected to be difficult, particularily along the north wall of North Thompson Creek.

Southern part

Limited access to the southern part of the Carbondale mining area is via a mine company road from Redstone, Colorado (see fig. 2). Road-distance from Redstone to the Dutch Creek Mines is about 6 miles.

Topography

Northern part

The topography of the northern part of the Carbondale mining area is indicated on the Cattle Creek and Stony Ridge topographic quadrangle maps (see figs. 5 and 6). Within the outlined area shown on those maps, altitudes above sea level range from about 6,500 feet on Fourmile Creek at the northeast edge of the outlined area, to about 10,150 feet at the west edge in sec. 29, T. 7 S., R. 89 W. (see fig. 5).

The relief of the northern part is about 3,650 feet; mean altitude is about 8,300 feet above sea level; and much of the western half is above 9,000 feet.

Central part

The topography of the central part of the Carbondale mining area is indicated on the Stony Ridge topographic quadrangle map (fig. 6). Within the rectangular outline shown on that map, altitudes above sea level range from about 7,200 feet on North Thompson Creek at the east edge of the outlined area, to 9,815 feet near the west-central edge.

The relief of the central part is about 2,600 feet; mean altitude is about 8,500 feet above sea level; and much of the western half is above 9,000 feet.

Southern part

The topography of the southern part of the Carbondale mining area is indicated on the Stony Ridge and Placita topographic quadrangle maps (see figs. 6 and 7). Within the outlined area shown on those maps, altitudes above sea level range from about 7,300 feet on the Crystal River at the southeast edge of the Placita quadrangle, to 11,852 feet in sec. 18, T. 10 S., R. 89 W. (see fig. 7).

The relief of the southern part is about 4,550 feet; mean altitude is about 9,580 feet above sea level; and much of the western half is above 9,800 feet.

Climate

In parts of western Colorado characterized by rugged topography, high relief, and abrupt differences in altitude, climate is a function of altitude rather than of latitude, and large variations in climatic conditions occur within short distances. Consequently, it is difficult to estimate the expected climate of the Carbondale coal mining area from weather records for weather stations in the vicinity. However, some indication of what to expect in terms of average yearly or monthly temperature, precipitation, and snow fall can be obtained from weather records for stations at Glenwood Springs and Crested Butte (see fig. 3). Those records are listed in Appendix A; they are summarized in the following paragraphs:

The weather station at Glenwood Springs, Colorado, at an altitude of 5,823 feet above sea level, is about 10 miles north of the northern part of the Carbondale coal mining area. At Glenwood Springs, normals for the period 1931-1960 show (a) the average temperature was 71°F in July, the warmest month of the year, and 25°F in January, the coldest month, (b) the average precipitation was maximum (1.80 inches) in January and minimum (1.19 inches) in June, and (c) the average annual precipitation was 18.03 inches. Average annual snowfall was 62.4 inches for the 26-year period prior to 1931 and 74.2 inches for the period 1931-1950, and maximum snowfall occurred during the months of December, January, and February. Normals for the period 1931-1960 show that the average annual temperature was 48°F and average monthly temperatures for November, December, January, February, and March were 36.4°, 27.4°, 24.8°, 29.7°, and 37.9°F, respectively, indicating that average monthly temperatures were below freezing for a 3-month period (December, January, and February).

Figure 5.--Diagrammatic relief map of part of western Colorado, showing locations of weather stations (Glenwood Springs and Crested Butte) with respect to Carbondale mining area.

The weather station at Crested Butte, at an altitude of about 8,900 feet above sea level, is about 25 miles southeast of the southern part of the Carbondale mining area. Weather records for Crested Butte contrast sharply with those for Glenwood Springs. At Crested Butte, normals for the period 1931-1960 show (a) the average temperature was $58^{o}F$ in July, the warmest month of the year, and $13.5^{o}F$ in January, the coldest month, (b) the average precipitation was maximum (2.68 inches) in January and minimum (1.31 inches) in June, and (c) the average annual precipitation was 23.00 inches. Average annual snowfall was 166.8 inches for the 19-year period prior to 1931 and 173.4 inches for the period 1931-1951, and maximum snowfall occurred during the months of December, January, February, and March. Normals for the period 1931-1960 show that the average annual temperature was $36^{o}F$; and average monthly temperatures, November-March, were 25.7^{o}, 16.8^{o}, 13.5^{o}, 16.3^{o}, and $22.6^{o}F$, respectively, indicating that average monthly temperatures were below freezing for a 5-month period (November-March).

Mean altitudes in the Carbondale coal mining area range from 8,300 to 9,580 feet. This range includes the altitude (8,900 feet) of the weather station at Crested Butte. On that basis, climatic conditions in the Carbondale coal mining area should be similar to those at Crested Butte, and climatic records for Crested Butte should provide a good indication of the climate expected for the Carbondale coal mining area.

Drainage

The Carbondale coal mining area is drained by small tributaries to the Roaring Fork and Crystal Rivers (see fig. 4).

As shown on figure 4, the northern part of the Carbondale mining area is drained by Fourmile and Freeman Creeks, and by Edgerton Creek and the South Branch. The central part of the area is drained by North and Middle Thompson Creeks. The southern part is drained by Porcupine and Dutch Creeks, which are tributaries of Coal Creek.

Drainage networks in the northern, central, and southern parts of the Carbondale mining area are shown in more detail on the Cattle Creek, Stony Ridge, and Placita topographic quadrangle maps, respectively (see figs. 5, 6 and 7).

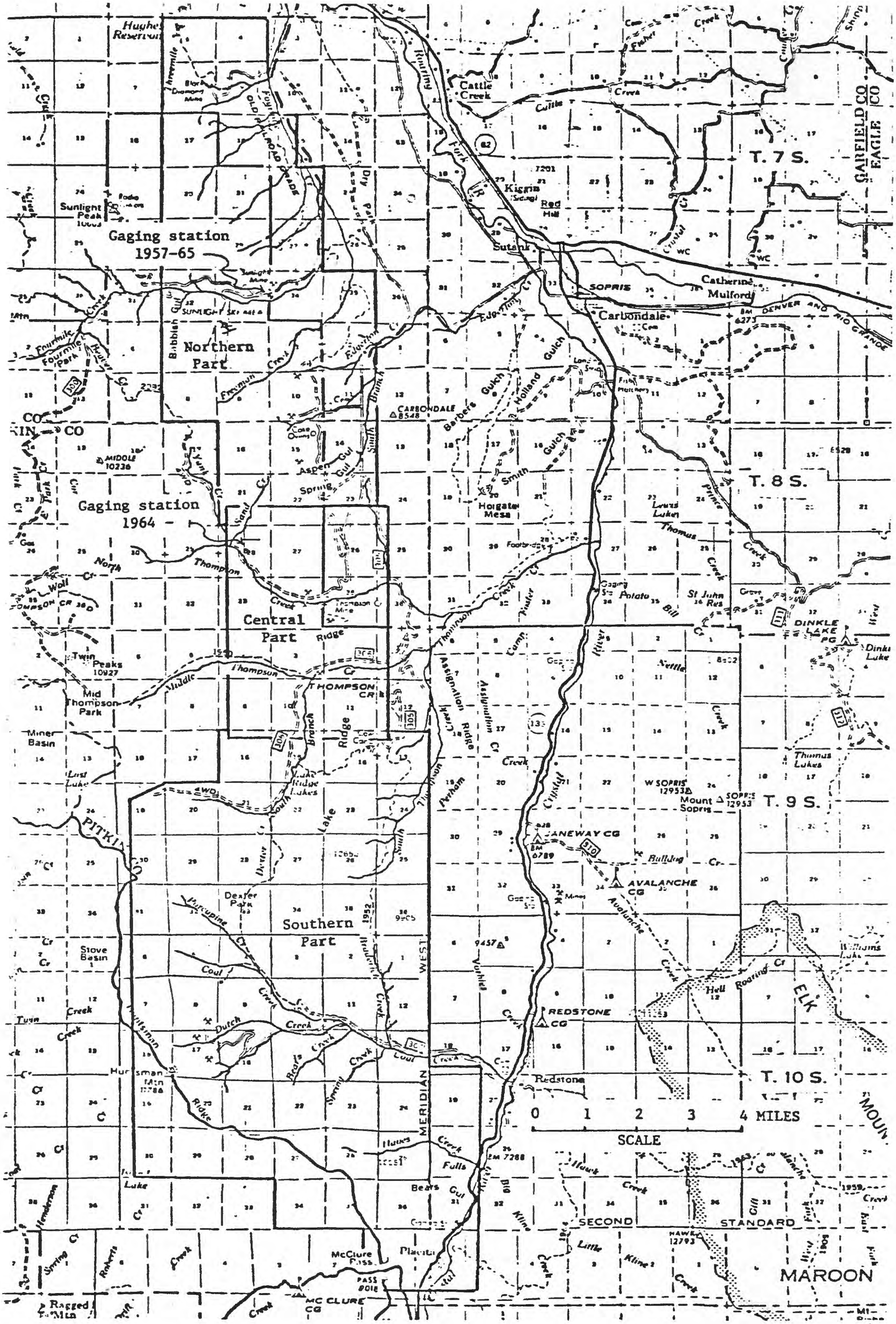

Figure 4.--Map showing drainage of the Carbondale coal mining area.

Surface water supply

Hydraulic mining requires a considerable amount of water. Although some of the water required can be conserved by recycling, the initial availability of an adequate water supply proximal to mining is an important measure of hydraulic mining potential.

Amounts of water potentially available from stream flow within the Carbondale coal mining area can be estimated from records for U.S. Geological Survey gaging stations that were installed on some of the larger creeks that drain the area. These records are listed in Appendix B. They are summarized in the following paragraphs.

Northern part

In 1957, a U.S. Geological Survey gaging station was installed on Fourmile Creek in sec. 34, T. 7 S., R. 89 W., near the Sunlight Mine (see figs. 4 and 5). The altitude of the gaging station is about 7,780 feet above sea level. Surface water supply records for that station were kept from October 1957 to September 1965. Records were discontinued in 1965. Records for water years 1961-65 are listed in Appendix B.

From records listed for Fourmile Creek, it can be derived that, for water years 1961-65, (a) the average total annual discharge was about 6,320 acre-feet, (b) the average discharge during a 2-month period, which included either April or June and always included May, accounted for about 80 percent of the average annual discharge, and (b) the average discharge during the month of May alone accounted for about 58 percent of the average annual discharge.

The daily rate of discharge during the months of August, September, and October is apt to be less than 1 cubic foot per second (i.e., less than 448 gpm), and freezing conditions prevail during 5 months of the year (November-March).

Central part

In 1963, a U.S. Geological Survey gaging station was installed on North Thompson Creek in sec. 28, T. 8 S., R. 89 W., about 0.5 miles downstream from Yank Creek (see figs. 4 and 6). The altitude of the gaging station is about 8,120 feet above sea level. Surface water supply records for that station have been kept since 1964. Records for water years 1964-70 are listed in Appendix B.

From records listed for North Thompson Creek, it can be derived that, for water years 1964-70, (a) the average total annual discharge was about 12,570 acre feet, (b) the average discharge during a 2-month period, which included either April or June and always included May, accounted for about 77 percent of the average annual discharge, and (b) the average discharge during the month of May alone accounted for about 47 percent of the average annual discharge.

The daily rate of discharge during the months of August, September, and October is apt to be less than 1 cubic foot per second (i.e. less than 448 gpm), and freezing conditions prevail during 5 months of the year (November-March).

Southern part

There are no gaging stations in the southern part of the Carbondale mining area. However, the area drained by Coal Creek is about the same as that drained by Fourmile Creek (see fig. 4). Amounts of stream flow in Coal Creek at a point in sec. 11, T. 10 S., R. 89 W. (see figs. 4 and 7) were probably comparable to those recorded at Fourmile Creek, for comparable water years.

Summary

Much of the stream flow in creeks draining the Carbondale mining area is derived from snow melt during April, May, and June. Stream flow during other months of the year is apt to be minimal. Consequently, if sustained hydraulic mining is contemplated and surface water is to be used for that purpose, storage of melt waters will be required in order to insure adequate water supply during dry months.

Yearly fluctuations in surface water supply during normally wet months are indicated by the following example: the winter of 1976-77 was characterized by unusually light snow fall. On May 20, 1977, stream flow on North Thompson Creek at the gaging station was reported to be about 25 cubic feet per second (13,220 gpm); this amount is about one-eighth the normal expected (see water records, North Thompson Creek, Appendix B).

Altitudes of mines, mine sites, and potentially minable coal beds within the Carbondale mining area are as much as 1,000 feet above stream levels, and many of those locations of coal are several miles distant from the nearest supply of surface water.

MINE-WATER SUPPLY

Many of the underground mines in the Carbondale coal mining area were in operation for many years in the past, and a few of those mines are still active; as a result, underground mine working are extensive in all parts of the mining area.

Surface waters enter such workings from up-dip recharge areas and creeks crossing the coal-bearing rocks, and surface waters percolate downward along permeable fault zones. As a consequence, considerable volumes of mine-water have accumulated in the mine workings. Such mine waters are potential sources of supply for subsurface hydraulic mining.

The underground mines on North Thompson Creek in the central part of the Carbondale mining area provide an example of the volume of stored mine waters involved. During the course of a brief field survey (May 19-24, 1977) it was learned that, during a dewatering operation in preparation for mine rehabilitation, about 150,000 gal/day was pumped from Mine No. 1, and normal overflow discharge continued at a rate of about 85,000 gal/day. The normal overflow discharge from Mine No. 3 was about 25,000 gal/day.

COAL GEOLOGY

Basic sources of geologic data, and acknowledgements

Geologic features of the Carbondale coal mining area were mapped on vertical aerial photos by U.S. Geological Survey geologists J. R. Donnell, E. R. Landis and W. E. Hallgarth, during 1952-53. A geologic map of the Carbondale area, on a planimetric base at a scale of 1:31,680, was published as an open file report by J. R. Donnell (1962).

The 1:24,000 scale geologic map(s) of the northern, central, and southern parts of the Carbondale area (figs. 5, 6 and 7) were constructed from the map prepared by Donnell (1962) by transferring the geologic data to the new topographic quadrangle map-bases. Contacts separating mapped stratigraphic units were not altered. Additional geologic information was provided by J. R. Donnell and E. R. Landis (oral commun.); their interest and support is hereby gratefully acknowledged.

Illustrated geologic data for the central part and the northern half of the southern part of the Carbondale area were derived principally from logs and records of drilling done in 1954-56. These logs and records belonged to the Anshutz Coal Company. We wish to thank Andrew Allen and Albert M. Keenan for permission to use some of that data, and for their interest in our work.

A comprehensive report by Collins (1976) on the coal deposits of the Carbondale, Grand Hogback, and southern Danforth Hills coal fields, eastern Piceance basin, Colorado, is a basic reference. The report includes detailed discussions of structural and igneous geology, stratigraphy of the Mesaverde Group (see fig. 8, this report), a proposed deltaic plain environment of deposition of coal-bearing rocks in Late Cretaceous time, analytic data on eastern Piceance basin coals, and geologic factors controlling coal quality. Also, Collins (1977) has recently published a detailed account of the coal geology of the Coal Basin area in the southern part of the Carbondale area (see fig. 7 for location).

Another basic reference is the report of Murray (1966) on the stratigraphy and structural geology of the Grand Hogback monocline, Colorado. This report emphasized regional structural aspects, but it also includes much geologic data on the Carbondale area.

A 1:250,000 scale preliminary geologic map of the Leadville 1° x 2° quadrangle, compiled by Tweto and others (1976), includes the Grand Hogback and Carbondale coal fields. This map also provides much general geologic information on the region, and includes data on the geologic ages of rock units and on the timing of events in past geologic time.

Stratigraphy

At least 17,000 feet of sedimentary rocks, ranging in age from Pennsylvanian to early eocene, is exposed in the Carbondale mining area and within a short distance to the east. Because the coal-bearing rocks were deposited in Late Cretaceous time and the stratigraphy pertinent to the coal geology of the area is related only to Upper Cretaceous and overlying lower Tertiary rocks, the underlying older rocks are not described here.

Stratigraphic nomenclature used by various authors for Upper Cretaceous and lower Tertiary rocks in the Carbondale mining area is summarized on figure 8 The nomenclature used by Collins (1976) is followed in this report because subdivision of the coal-bearing rocks of the Williams Fork Formation facilitates mapping at 1:24,000 scale (see figs. 5, 6, and 7) and allows for more precise definitions of the stratigraphic positions of coal zones and coal beds (see figs. 9, 10 and 11).

Some events of stratigraphic record, which lead to or were involved with the deposition and preservation of coal are noted in the following descriptions of Upper Cretaceous and lower Tertiary rocks in the Carbondale area.

Donnell (1962)	Warner (1964)	Tweto (1976)	Collins (1976)
Wasatch Formation	not studied	Wasatch Formation	Wasatch Formation
Ohio Creek Conglomerate	not studied	Ohio Creek Conglomerate	Ohio Creek Conglomerate
Mesaverde Formation: upper part	Mesaverde Formation	Mesaverde Formation: upper part	Mesaverde Group, Williams Fork Formation: upper, undifferentiated
Mesaverde Formation: upper part	Mesaverde Formation	Mesaverde Formation: upper part	Mesaverde Group, Williams Fork Formation: Paonia Shale Mbr
Mesaverde Formation, lower: upper ss zone	Mesaverde Formation	Mesaverde Formation, lower part: upper ss zone	Mesaverde Group, Williams Fork Formation, Bowie Shale Mbr: upper ss
Mesaverde Formation, lower: middle ss zone	Mesaverde Formation	Mesaverde Formation, lower part	Mesaverde Group, Williams Fork Formation, Bowie Shale Mbr
Mesaverde Formation, lower: lower ss zone	Rollins Sandstone Member	Mesaverde Formation, lower part: Rollins Sandstone Member	Mesaverde Group, Iles Formation: Rollins Sandstone Member
Mancos Shale	Mancos Shale	Mancos Shale	Mesaverde Group, Iles Formation
Cozzette Ss Mbr 1/	Cozzette Ss Mbr 2/	Mancos Shale	Mesaverde Group, Iles Formation: Cozzette Ss Mbr 1/
Mancos Shale	Mancos Shale	Mancos Shale	Mancos Shale

1/ -Of Price River Formation
2/ -Of Mesaverde Formation

Figure 8.--Stratigraphic nomenclature used by various authors for Upper Cretaceous and lower Tertiary rocks in the Carbondale area.

Mancos Shale

The Mancos Shale (Upper Cretaceous) includes deposits of dark-gray marine shale and minor amounts of limestone which are widely distributed throughout Colorado, New Mexico, and eastern Utah. The shales and limestones of the Mancos were deposited in a Late Cretaceous epicontinental sea.

Iles Formation

In the late stages of Mancos Shale deposition, eastward regression of the sea from northeastern Utah and northwestern Colorado was interrupted by many temporary westward transgressions as recorded by marine strata resting on nonmarine strata (Zapp, 1960, p. B246). Typically, each regressive-transgressive cycle resulted in deposition of intermixed marine and nonmarine siltstone and sandstone along a strand line (i.e., the line of maximum marine retreat or advance).

Regionally, the Iles Formation, as separated from the underlying Mancos Shale, includes strand line deposits of siltstone and sandstone (representing regressive phases) and intervening deposits of shale similar to the Mancos (representing transgressive phases). Collins (1976, p. 23) noted that the Iles Formation-Mancos Shale contact--"is everywhere placed at the bottom of the first significant sandstone unit in the Mancos."

Regionally, however, the strand line deposits tend to cross geologic time lines. Eastward regression seems to have been more pronounced in one place than in another, and the shore lines of the Late Cretaceous sea were no doubt irregular. As a result, stratigraphic positions of correlative strand line deposits seem to vary considerably, and the deposits of sandstone and siltstone commonly vary dramatically in thickness and prominence from place to place. Generally, however, such deposits tend to become thicker and more prominent to the west of the depositional strike, and additional strand line deposits are commonly recorded in more westward stratigraphic sections.

These characteristics of strand line deposits and the stratigraphic complexities they introduce are illustrated by Warner (1964, fig. 3). Warner's (1964, fig.3) illustration also emphasizes a significant aspect of strand deposits: where such deposits are thick and well developed, they apparently set the stage for subsequent deposition and preservation of coal.

In South Canyon Creek (see index map on fig. 9 for location), two lithic units consisting of sandstone and carbonaceous sediments and named the Cozzette and the Corcoran Sandstone Members are present and intertongue with the upper part of the Mancos Shale (see Warner, 1964, fig. 3). Those members were placed in the Iles Formation by Collins (1976); and following the guideline provided by Collins the Iles Formation-Mancos Shale contact should be placed at the base of the lower unit (i.e., the Corcoran) in that area.

In the northern and central parts of the Carbondale mining area, however, the Corcoran is poorly developed. At North Thompson Creek, for example, the Corcoran consists of shaly siltstone; to the south this siltstone grades laterally into Mancos Shale. In contrast, at North Thompson Creek the upper unit (the Cozzette) consists of a 20-foot sandstone about 100 feet above the Corcoran and about 900 feet below the Rollins Sandstone Member of the Iles. Accordingly, in the northern and central parts of the area the Iles Formation-Mancos Shale contact is placed at the base of the Cozzette (figs. 5 and 6).

Between Middle Thompson Creek (see fig. 6) and the north rim of Coal Basin to the south (see fig. 7), the Cozzette Sandstone Member intertongues with and is replaced by Mancos Shale. Accordingly, the Iles Formation is indicated on figure 7 as being only its topmost member (i.e., the Rollins Sandstone Member). It should be noted, however, that the exposures in the Coal Basin area are to the west of the depositional strike of the rocks in the Middle Thompson Creek. Therefore, to the west, the Cozzette might be present at the surface, and the overlying shales might be silty.

These possibilities are supported by Collins (1977, p. 366) and Warner (1964, p. 1099); they noted the occurence of 40-50 feet of silty shale overlain by 6 feet of silty sandstone, about 1,000 feet below the base of the Rollins, near the Dutch Creek Mine in sec. 16, T. 10 S., R. 89 W. (see fig. 7 for location). Collins (1977, p. 366) identified those rocks as being equivalent to the Cozzette, and on that basis he placed the Iles Formation-Mancos Shale contact at the base of that lithic unit (see Collins, 1977, fig. 1, p. 364). Collins (1977, p. 366) also noted that the sequence of shale above the Cozzette is silty and becomes progressively sandier upward toward the base of the Rollins Sandstone Member.

Rollins Sandstone Member--The Rollins Sandstone Member includes intermixed marine and nonmarine sandstone and siltstone deposited during a regressive phase. The Rollins is continuous throughout the Carbondale mining area (see fig. 9). According to the usage of Collins (1976), the Rollins is the topmost member of the Iles. Accumulation of the Rollins Sandstone Member set the stage for subsequent deposition and preservation of coal, as represented by the lower zone of the Williams Fork Formation (see fig. 9).

Williams Fork Formation

Murray (1966, p. 81-82) noted that the Williams Fork Formation was named by Hancock (1925) to identify about 1,000 feet of alternating Upper Cretaceous sandstone, sandy shale, and coal beds exposed at the type locality near the confluence of the Williams Fork and Yampa Rivers, about 30 miles north of Meeker, Colorado. In that area, the Williams Fork Formation is underlain by the Iles Formation (which also contains coal-bearing rocks); and the Trout Creek Sandstone Member is the topmost member of the Iles. The Trout Creek Sandstone and the Rollins Sandstone Members may be equivalent (Warner, 1964, p. 1099). (Coal-bearing rocks of the Iles are not present in the Carbondale area).

From the type locality, the Williams Fork Formation thickens to the south. In the central part of the Carbondale area as measured along North Thompson Creek, the total thickness of the Williams Fork Formation is 3,915 feet (see Collins, 1976, table 1, p. 22).

Bowie Shale Member--The Bowie Shale Member of the Williams Fork is shown on figures 5, 6, and 7. The member is 700-1,200 feet thick. The lowermost part of the member contains the lower coal zone of this report (see fig. 9). The middle sandstone bed of the Bowie is overlain by a middle coal zone which is present only in the southern part of the Carbondale area. the upper sandstone bed is the topmost unit of the member (see fig. 9).

In the Carbondale area, the Bowie contains a variety of rocks in addition to shale and coal. Both the middle and upper sandstone beds of the Bowie are continuous throughout the area and are suggestive of strand line deposits. Rock types contained in the Bowie are repeated vertically in crude cyclic sequences, and some of the coal beds are laterally discontinuous. Variations in strigraphic levels and intervals between coals are common (see fig. 9). Such features are characteristic of deltaic deposits.

Paonia Shale Member.--In the Coal Basin area (see fig. 7) the Paonia Shale Member consists of 600 feet of nonmarine sandstones, siltstones, shales, and coal beds overlying the Bowie Shale Member (Collins, 1976, p. 28). Collins (1979, p. 28) noted that the Paonia is difficult to trace north of Coal Basin, but he felt that it could be traced as far north as North Thompson Creek. On that basis, the Paonia was mapped as a 600-foot-unit in the Thompson Creek mining area (i.e., the central part of the Carbondale area) but the contact with the overlying upper part of the Williams Fork Formation is indefinite (see fig. 6). The lower part of the unit contains the upper coal zone of this report (see fig.9).

In most of the northern part of the Carbondale area (see fig. 5), rocks that might elsewhere be assigned to the Paonia Member of the Williams Fork Formation are included with those of the upper part of the Williams Fork to form a map unit about 2,900 feet thick. Also, in most of the southern part of the Carbondale area (see fig. 7), the rocks of the Paonia are not differentiated from those of the upper part of the Williams Fork, because structural complexities make the upper contact of the Paonia difficult to trace. However, the upper coal zone of this report maintains the same stratigraphic position with respect to the underlying upper sandstone bed of the Bowie throughout the Carbondale area (see fig. 9).

As compared to the lower coal zone, the lateral discontinuity of coal beds in the upper coal zone, and variations in stratigraphic levels and intervals between coals, are more pronounced (compare figs. 10 and 11). Rocks containing the upper coal zone probably are deltaic deposits.

Upper part of Williams Fork.--The upper part of the Williams Fork Formation includes nonmarine deposits of sandstone, siltstone, sandy shale, mudstone, shale, and thin coal. On figure 6, the upper part of the Williams Fork is separated from the underlying Paonia and the upper part is about 2,300 feet thick; whereas, on figures 5 and 7 the Paonia is included with the upper part to form a map unit about 2,900 feet thick.

The upper part of the Williams Fork is only about 1,800 feet thick in areas south and southwest of the Carbondale area. The difference in thickness (about 1,300 feet) is perhaps best explained by pre-Ohio Creek erosion.

Ohio Creek Conglomerate

An erosional unconformity separates the Ohio Creek Conglomerate from the underlying upper part of the Williams Fork Formation (Hanks, 1962, p. 145).

The Ohio Creek Conglomerate (Paleocene) is exposed discontinuously between Fourmile Creek in the northern part of the Carbondale area and the southern part of the area. At North Thompson Creek, for example, the Ohio Creek consists of about 90 feet of sandstone containing distinctive chert, quartzite, and quartz pebbles. In the Ragged-Chair Mountain area south of the Carbondale area, the Ohio Creek is as much as 200 feet thick (Hanks, 1962, p. 145).

In the Carbondale area, however, the Ohio Creek Conglomerate is not differentiated from the overlying Wasatch Formation (see figs. 5, 6, and 7).

Wasatch Formation

The Wasatch Formation (Paleocene and Eocene) conformably overlies the Ohio Creek. The Wasatch consists of nonmarine deposits of conglomerate, sandstone, siltstone, mudstone, and shale. In the field, the rocks of the Wasatch are distinguished by their bright varicolored appearance.

A complete section of the Wasatch is not present in the Carbondale mining area, because the upper part has been removed by erosion. However, a minimum of 4,000 feet of Wasatch is exposed along Fourmile Creek in the northern part of the Carbondale area. About 7 miles to the northwest, in sec. 17, T. 7 S., R. 90 W., the thickness of the combined Wasatch and Ohio Creek is at least 5,100 feet (see Murray, 1966, table 5, p. 93).

Igneous Rocks

Within the Carbondale mining area there was considerable igneous activity during the Tertiary Period.

Much of the northernmost quarter of the Carbondale area is covered by a basalt flow which is as much as 400 feet thick (see fig. 5). The alkali basalt, of Pliocene and Miocene age (Tweto and others, 1976), occurs in flat-lying layers resting on steeply-dipping Upper Cretaceous beds.

Surficial deposits that include abundant basalt boulders occur throughout the northern and central parts of the area (see figs. 5 and 6). These deposits are most common at higher elevations in the western half of the Carbondale area. Donnell (1962) mapped those deposits as remnants of the Tertiary basalt flow. Tweto and others (1976) subsequently interpreted them as high-level alluvium of Pleistocene and/or Pliocene age, and that identification is used on figures 5 and 6.

Intrusive igneous rocks, exposed as small plugs or dikes, occur in many places within the southern part of the Carbondale area (see fig. 7). The intrusive rocks are of Oligocene age (Tweto and others, 1976). The plugs consist of quartz monzonite or grandiorite, whereas the dikes are dacitic or andesitic in composition. Some of the dikes cut the coal-bearing rocks in the Coal Basin area, and the coal has been naturally coked along their margins (Collins, 1977, p. 365).

Oil and gas test holes drilled in secs. 9 and 10, T. 10 S., R. 89 W. (see fig. 7) penetrated as much as 1,500 feet of quartz monzonite at a depth of about 4,200 feet. This subsurface intrusive in the Coal Basin area is probably a laccolith; overlying strata were domed by the laccolith, to form the Coal Basin anticline (Collins, 1977, p. 363).

Structure

In the Carbondale mining area, the dominant structural feature is the Grand Hogback monocline which forms the eastern margin of the Piceance basin. Strata exposed in the Carbondale area form the uppermost part of this westward dipping monoclinal fold.

In the northern and central parts of the area, coal beds and associated rocks dip westward at 30 to 50 degrees (see figs. 5 and 6). To the west, the dip of overlying strata gradually decreases. Along the western margin of the central part of the area, the dip averages about 15° W. (see fig. 6). The combination of dip and increasing topographic elevations causes coal bed A to be under more than 1,500 feet of overburden within one-half mile west of the line of outcrop (see figs. 5 and 6). At the west margin of the northern and central parts and to the west, coal bed A is covered by about 3,000 feet of overlying rocks.

In the central part of the area, structure contours drawn on the base of coal bed A show that coal bed A decends below 6,000 feet at the west margin. About 3 miles to the west of the margin, elevation control from oil and gas tests indicates that the bed reverses its westward dip and rises to about 6,000 feet above sea level. The configuration of the 6,000-foot structure contour suggests a shallow northwest-plunging syncline between the central part of the Carbondale area and the wells to the west (see fig. 6). This shallow structure is probably a northwest extension of the syncline north of Coal Basin (see fig. 7).

In the southern part of the Carbondale area and in the Coal Basin area, a cross-trending, northwest-plunging anticline interrupts the westward regional dip of the Grand Hogback monocline. Exposed coal-bearing strata dipping 10 to 20 degrees outline the anticlinal structure (see fig. 7). In areas to the west and southwest of the anticline, the pitch of overlying Tertiary strata increases and coal bed A lies under 1,500 feet of overburden within one-half mile of the outcrop.

Faulting

Faulting here refers to the process in which rocks are displaced by fracturing. A fault is a fracture or zone of fractures along which rocks have been displaced on one side relative to the other.

In the northern and central parts of the Carbondale area, faults cutting exposed coal-bearing rocks and involving detectable displacement were not observed during the surface geologic mapping. However, extensive vegetation and surficial cover may have obscured faults that break to the surface in those parts of the area.

Within the subsurface of the central part of the Carbondale area, two fracture systems cutting coal beds and associated rocks were encountered during past mining. These fracture systems are described and illustrated in detail in a report by Keenan and Carpenter (1960). Displacements along faults are generally less than 60 feet; the strike of one system is to the west and the strike of the other is to the north.

As shown on figures 6 and 7, numerous faults cut exposed coal-bearing rocks in the southern part of the Carbondale area. Faults in the northwest corner of Coal Basin have displacements ranging from 100 to 300 feet with strikes generally perpendicular to the strike of the coal-bearing rocks.

Collins (1977, p. 366) reported that faults seriously hampered mining in the Bear Creek mine (see fig. 7 for location).

As described and illustrated by Keenan and Carpenter (1960), faulting of pitching coal beds produces a complex geometry for mining. Applied geology is required in order to determine the extent, pattern, and frequency of faults. Even a few feet of coal-bed displacement can seriously disrupt mining operations, and if mining problems resulting from displacement and roof breakage are to be resolved successfully, geologic data on faulting must be incorporated into mining plans and operations

In addition to adding to mining costs, faulting may also influence coal quality. For example, coal in the immediate vicinity of faults has been found to have weaker coking characteristics (Keenan and Carpenter, 1960, p. 235). This adverse effect could be the result of oxidation of the fractured coal, owing to greater porosity and permeability of coal and associated rocks in fault zones.

Stratigraphic sections

Stratigraphic sections of Upper Cretaceous coal-bearing rocks in the Carbondale area and vicinity are shown on figure 9. The north-south line of sections (fig. 9) extends from South Canyon Creek to the vicinity of Placita (see index map on fig. 9). The top of the Rollins Sandstone Member of the Iles Formation is the datum for the line of sections.

The Rollins, and the middle and upper sandstone beds of the overlying Bowie Shale Member of the Williams Fork Formation, are continuous over the (30-mile) distance from South Canyon Creek to Placita, as also are the lower and upper coal zones (see fig. 9). The index map on figure 9 shows the locations of sections illustrated on figures 10, 11, 12 and 13.

The line of stratigraphic sections shown on figure 10 illustrates positions and correlations of coal beds A, B, C and D, in the lower coal zone, and the position of the Dutch Creek coal bed representing the middle coal zone. The middle coal zone is not present in the central and northern parts of the Carbondale area.

Stratigraphic sections shown on figure 11 illustrate positions and possible correlations of coal bed in the upper coal zone. The top of the upper sandstone bed of the Bowie Shale Member is the datum for that line of stratigraphic sections. The coal bed identified as the Sunshine is laterally continuous, but overlying coal beds are difficult to trace and correlate (see fig. 11)

Coal sections

Coal sections of the principal coal beds (A and B) of the lower coal zone are shown on figure 12, and those of the Sunshine, Anderson, and North Rim coal beds in the upper coal zone are shown on figure 13.

The lines of coal sections illustrate lateral variations in thickness of coals and lateral variations in the number, thickness, and spacing of non-coal and impure-coal partings in the coal beds.

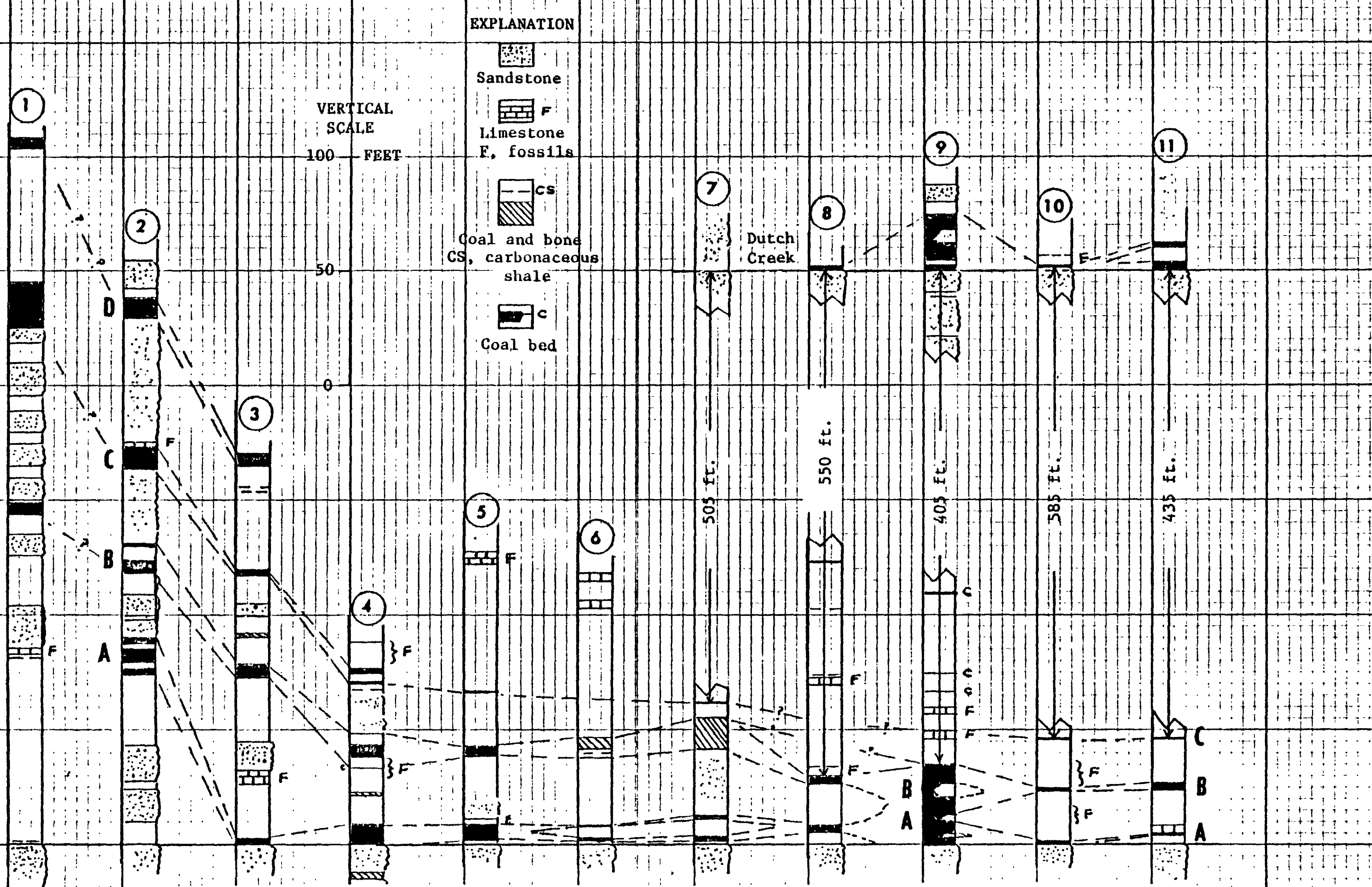

Figure 10 --Correlation of coal beds A,B,C and D in the lower coal zone, and of Dutch Creek coal bed in the middle coal zone, Carbondale coal mining area, Garfield and Pitkin Counties, Colorado. Datum for coal beds in lower coal zone is top of Rollins Sandstone Member of Iles Formation. Datum for Dutch Creek coal bed is top of middle sandstone bed of Bowie Shale Member of Williams Fork Formation. Locations of sections are shown on index map on figure 9.

BEE 10x10

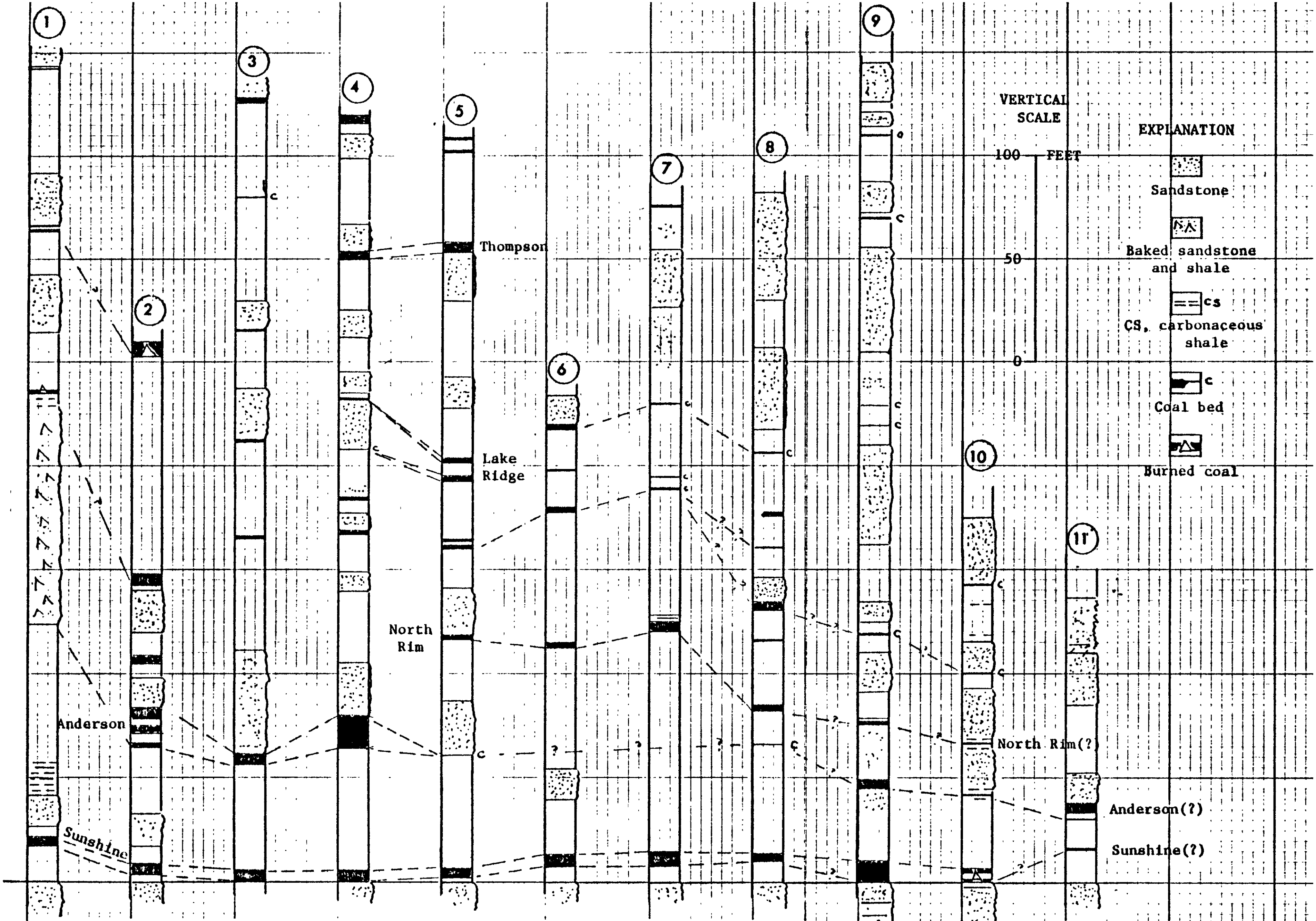

Figure 11. --Possible correlation of coal beds in the upper coal zone, Carbondale coal mining area, Garfield and Pitkin Counties, Colorado. Datum is top of upper sandstone bed of Bowie Shale Member of Williams Fork Formation. Locations of sections are shown on index map on figure 9.

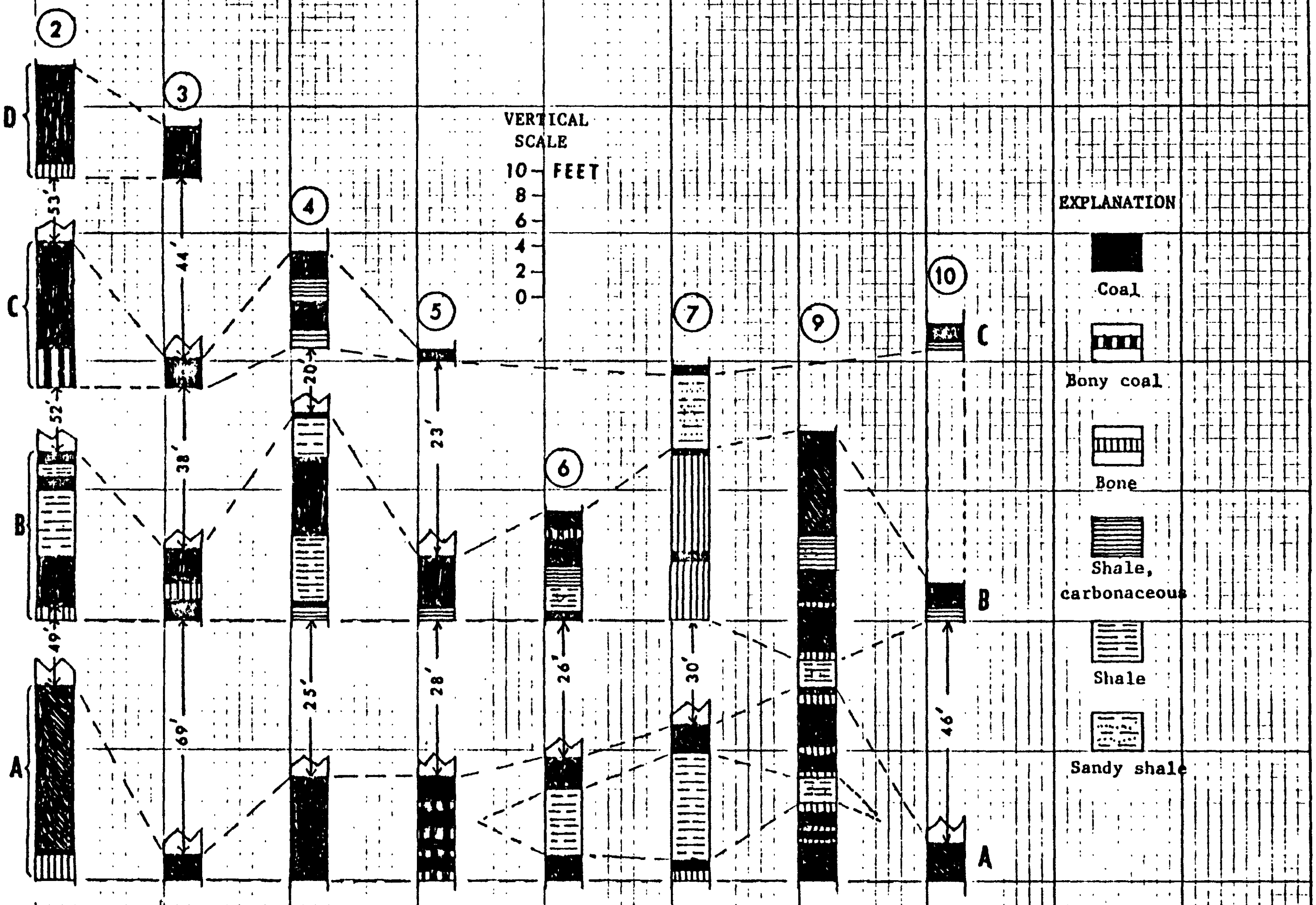

Figure 12. --North-south line of coal sections, coal beds A, B, C, and D, lower coal zone. Locations of sections are shown on index map on figure 7.

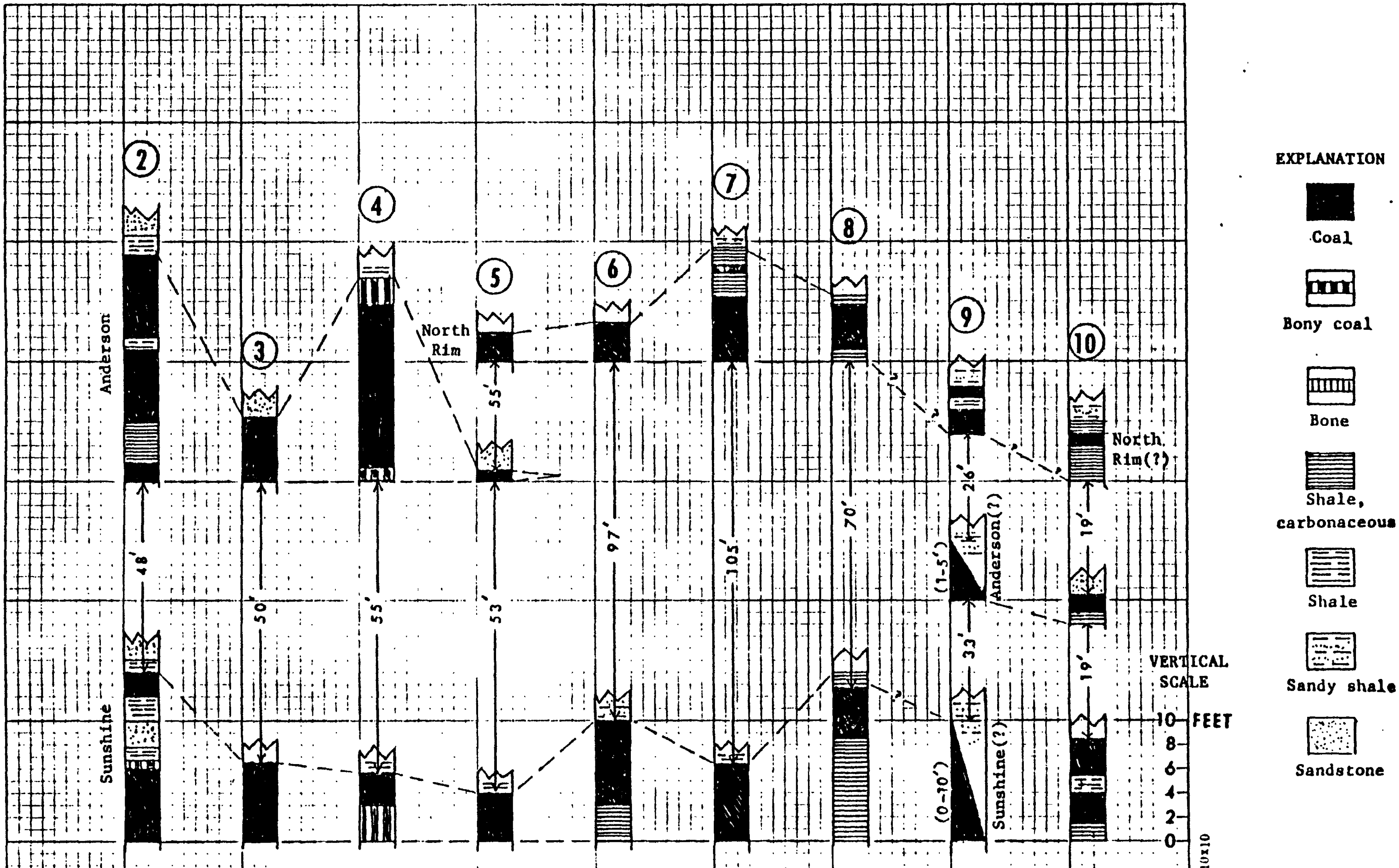

Figure 13. --North-south line of coal sections, Sunshine, Anderson and North Rim coal beds, upper coal zone. Locations of sections are shown on index map on figure 9.

MINING

Underground mining of coal in the Carbondale mining area dates from 1887. Total cumulative production of coal since that time is tabulated in Appendix C (see table C-3).

As of 1977, however, the Sunlight mine on Fourmile Creek is the only active mine in the northern part of the Carbondale area, and production from that mine is small. The Thompson Creek mines in the central part of the area were active from 1955 to 1966. Two of those mines (No. 1 and No. 3) were reopened in 1975; they are in the process of being rehabilitated and converted to longwall mining. The Dutch Creek Mine(s) in the southern part of the area are producing about one million tons/yr.

Coal resources

Estimates of original resources contained in coal beds more than 42 inches thick in the Carbondale area were made by Donnell (1962); they are summarized in Appendix C (see table C-4). Although those estimations do not conform to current Geological Survey standards, they are the most complete and accurate data available as of December 1, 1977

ANALYTICAL DATA

Analytical data on the chemical and physical properties of coal samples from the Carbondale mining area are tabulated in Appendix C (see table C-1). The coal beds sampled and analyzed include correlative beds in the lower and upper coal zones and the Dutch Creek coal bed representing the middle coal zone in the southern part of the Carbondale area.

The analytical data tabulated in table C-1 (see Appendix C) are grouped according to localities 1 through 8 from north to south, respectively. These localities are shown on figure 14, where the line of localities is approximately parallel to the strike of the coal beds.

This arrangement of analytical data emphasizes that chemical and physical properties of correlative coals exhibit progressive and pronounced changes over relatively short distances along the strike of the coal beds. Some of these changes are summarized in table 1. In table 1, locality 4 (near North Thompson Creek) serves as a reference point and distances to other localities are measured from that locality (see also fig. 14).

Free swelling index (FSI)

The ASTM test method (D 720-67) is used to determine the free swelling index (FSI) of a coal sample and to provide an approximate measure of its coking properties. The test does not provide a measure of coking quality.

In the test, a sample of coal is heated to 820^0 C in a standard covered crucible, and the size and shape of the resulting "coke button" is compared to standard outlines which have been assigned index numbers from 1 to 9 in order of increased swelling and relative strength of coking.

Coals in localities 2 and 3 have FSI's of from 0 to 1 1/2 (see table 1), indicating that those coals have virtually no coking properties. In contrast, coals in localities 4 and 5 have relatively strong coking properties, as indicated by FSI's of from 7 to 8 1/2; and coals in locality 6 have FSI's of 9 and 9+, indicative of the strongest coking coal. At locality 7, however, the southward increase in FSI's trend seems to be reversed: coals in locality 7 have FSI's of 4 and 4 1/2; and the FSI's of coals in locality 8 are probably less than 4.

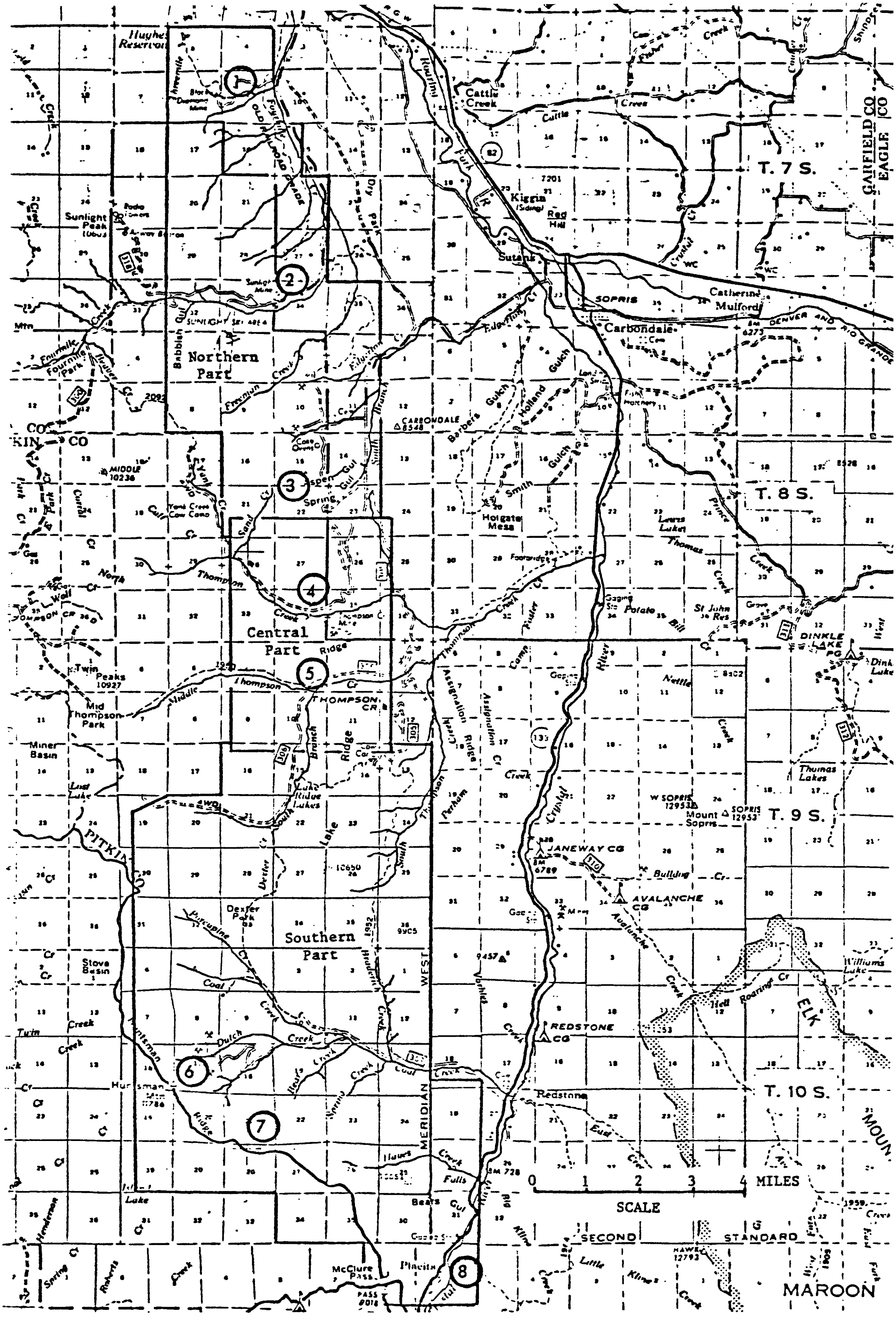

Figure 14.--Map showing localities of coal samples for which analytical data are available.

Table 1.--Summary of analytical data on coal samples from Carbondale mining area.

Locality[1/]	Mine area-coal bed	Distance from N. Thompson Cr. (loc. 4)	Coal rank	Dry coal, percent Vol.	F. C.	H	O_2	FSI	HGI
(1)	Diamond mine	10 mi N.							
	A bed		HvCb	36.2	54.6	5.4	19.7	-	-
	D bed		HvCb	35.9	49.4	5.9	21.5	-	-
(2)	Sunlight mine	5 mi N.							
	"D" bed		HvBb	42.6	54.0	5.5	12.5	1	47
(3)	Aspen Gulch	2.5 mi N.							
	"B" bed		HvBb	37.3	55.5	5.3	7.9	1½	
(4)	N. Thompson Cr	(0)							
	"A" bed		Mvb	30.8	60.3	5.2	6.4	7	89
	Anderson bed		HvAb	32.4	59.5	5.1	5.4	7	71
(5)	Middle Thompson Cr.	1.5 mi S.							
	Anderson bed		Mvb	27.0	59.5	4.9	3.5	8½	-
(6)	Dutch Creek	9 mi S.							
	B bed		Mvb	23.0	68.6	4.8	2.3	9	110
	Dutch Creek bed		Lvb	21.2	74.9	5.2	2.9	9	-
(7)	Bear Creek	10 mi S.							
	B bed		Mvb	26.1	70.9	4.9	4.9	4	-
(8)	Placita mine	13 mi S.							
	Sunshine bed		HvAb	33.8	57.4	5.3	7.6	-	-

1/ Localities are shown on figure 14. See Table C-1 for complete analysis figures.

The progressive increase in the FSI's of correlative coals (i.e., from locality 1 to 6) is accompanied by a decrease in percent-content of volatile matter and oxygen, an increase percent-content of fixed carbon, and a hydrogen content that remains about the same at all localities (see table 1). These changes illustrate some analytical guides outlined by Johnson (1952, p. 1) for separating coking coals from noncoking or weakly coking coals. According to the guides, and as a general rule, coals may be eliminated as possible coking fuels if (a) the oxygen content is greater than 11 percent, (b) the ratio of hydrogen to oxygen is less than 0.5, or (c) the ratio of fixed carbon to volatile matter is less than 1.3.

As applied to the coals at locality 1, for example, the analytical guides indicate that the coals are noncoking because their oxygen contents are considerably more than 11 percent, and their ratios of hydrogen to oxygen are considerably less than 0.5.

Hardgrove grindability index (HGI)

The ASTM standard test method for grindability of coal by the Hardgrove machine method (D 409-71) determines the relative grindability or ease of pulverization (i.e., <200 mesh) of a given coal sample, as compared to that of samples selected as standards. The specific purpose of the test is to determine relative ease of grinding finely enough to use as pulverized fuel.

In the test, initially sized sample fractions of 0.6 to 1.2 mm are used and 50 g of such sized particles are ground for 60 revolutions. The calculated weight of the material passing through a No. 200 sieve (particle sizes <0.075 mm) is then plotted against weights of <200 mesh materials corresponding to HGIs of standard reference samples (40, 60, 80, and 100). In this method of calculation the higher the resulting HGI of the given sample of coal, the easier the coal was to grind. Coals that are easiest to grind have HGIs on the order of 100 to 110.

Correlative coals at localities 2 and 3 have relatively low HGIs (47-52), indicating that samples of such coals were relatively hard to grind. Coals at locality 4 were much easier to grind as indicated by HGIs of 69 to 89. At locality 6 the coals have HGIs of 110, which indicates extreme ease of grindability.

As a physical property of coal, however, grindability is complex and somewhat ambiguous. For example, Yancey and Geer (1968, p. 1-3) describe grindability as "-a composite physical property embracing other specific properties such as hardness, strength, tenacity, and fracture." Even though the HGI test for grindability would not provide a measure of those specific properties, because the initial sample-size fractions used are rather minute (0.6 to 1.2 mm), this does not preclude associating HGI data with test data for other specific properties of coal.

Many workers have noted an empirical correlation between HGI and ASTM rank of coal. For example, Brown and Hiorns (1963, p. 148; and fig. 9) compiled HGI data on American, British, and Indian coals, and related HGI to rank as determined by percentage of volatiles. Results obtained indicate that coals containing 15-30 percent volatiles generally have HGIs >90, whereas coals containing 30-40 percent volatile exhibit HGIs from 90 to 40 respectively.

FSI, HGI, and content of volatile matter

Many workers have observed consistent relations between FSI, HGI, and percent of volatile matter. Strongly coking coals generally have relatively high HGI's and contain 20-35 percent volatiles, whereas weakly-coking coals generally have low HGI's and contain >35 percent volatiles. These relations are supported and illustrated by the analytic data for coals in the Carbondale mining area.

From work establishing these relations, Donahue and Leonard (1967, p. 365) concluded that it is possible to estimate the magnitude of one property if the other two properties are known. Their graphic illustrations (fig. 4, p. 363; fig. 7, p. 364) of the relationship of VM (within selected ranges of FSI) to coal hardness (i.e., grindability) are summarized on figure 15.

Analytic data for Carbondale coals are not complete for all samples and in cases either the FSI or the HGI was not determined. For such samples, the graphs shown on figure 15 provide a means of estimating the magnitude of either the FSI or HGI.

Coal hardness

Variations in hardness of coals have been cited as a factor affecting the productivity rates acheived in subsurface hydraulic mining. The property is discussed in that context in a later section of this report (see "Subsurface Hydraulic Mining;" "Coal Hardness").

Hardness as discussed in this section concerns the relations between FSI, HGI, and content of volatile matter (fig. 15). In documenting these relations, Donahue and Leonard (1967, p. 364) made the following statement:

> "Figs. 4 and 5 [cf Fig. 15] illustrate the relationship of VM, within selected ranges of FSI, to coal hardness. In figs. 4 and 7, HGI can be estimated from the measured values of VM and FSI. It is apparent from fig. 4 that for FSI values of 8 or less, decreases in VM are related to increasing coal softness (HGI). Generally, for any given VM content, increases in HGI are associated with increases in FSI. The hard, lower HGI coals, are generally weakly coking with FSI in the range of 1 to 1 1/2."

This quotation indicates that Donahue and Leonard consider (coal) hardness and grindability to be similar or comparable physical properties.

Hardness is also a specific property of coal. References commonly are made to the dynamic hardness of vitrain layers as measured by Vickers and Shore scleroscope methods. Such measurements provide a means of examining changes in that property according to rank of coal (Brown and Hiorns, 1963, p. 127). Yancey and Geer (1968, p. 1-25) compiled data on dynamic hardness of vitrain layers in coals of various types which are summarized as follows:

Type of coal	Volatiles (percent)	Shore hardness number1/	Vickers hardness numbers2/
Anthracite	6	122	54
Bituminous (noncaking)	41-42	107-109	24-29
Bituminous (caking)	40-44	105-107	22-26
Bituminous (coking)	21-36	44-65	9-16

1/ Shore number represents dynamic deformation by rebound.

2/ Vickers number shows static deformation, by size of indentation formed when load is applied to a steel ball or a diamond in contact with the coal.

The preceding tabulation indicates that vitrain layers in bituminous coking coals containing 21-36 percent volatile matter, are relatively soft. But the tabulation also implies that bituminous coals containing 37-40 percent volatiles probably will exhibit large (and presumably progressive) increases in hardness of vitrain layers. Variations in the hardness of coal containing 37-40 percent volatiles may actually be a function of the coking strengths of the coals. Vitrain layers of weakly coking coal may be relatively hard, whereas vitrain layers of rather strongly coking coal may be relatively soft (see fig. 15).

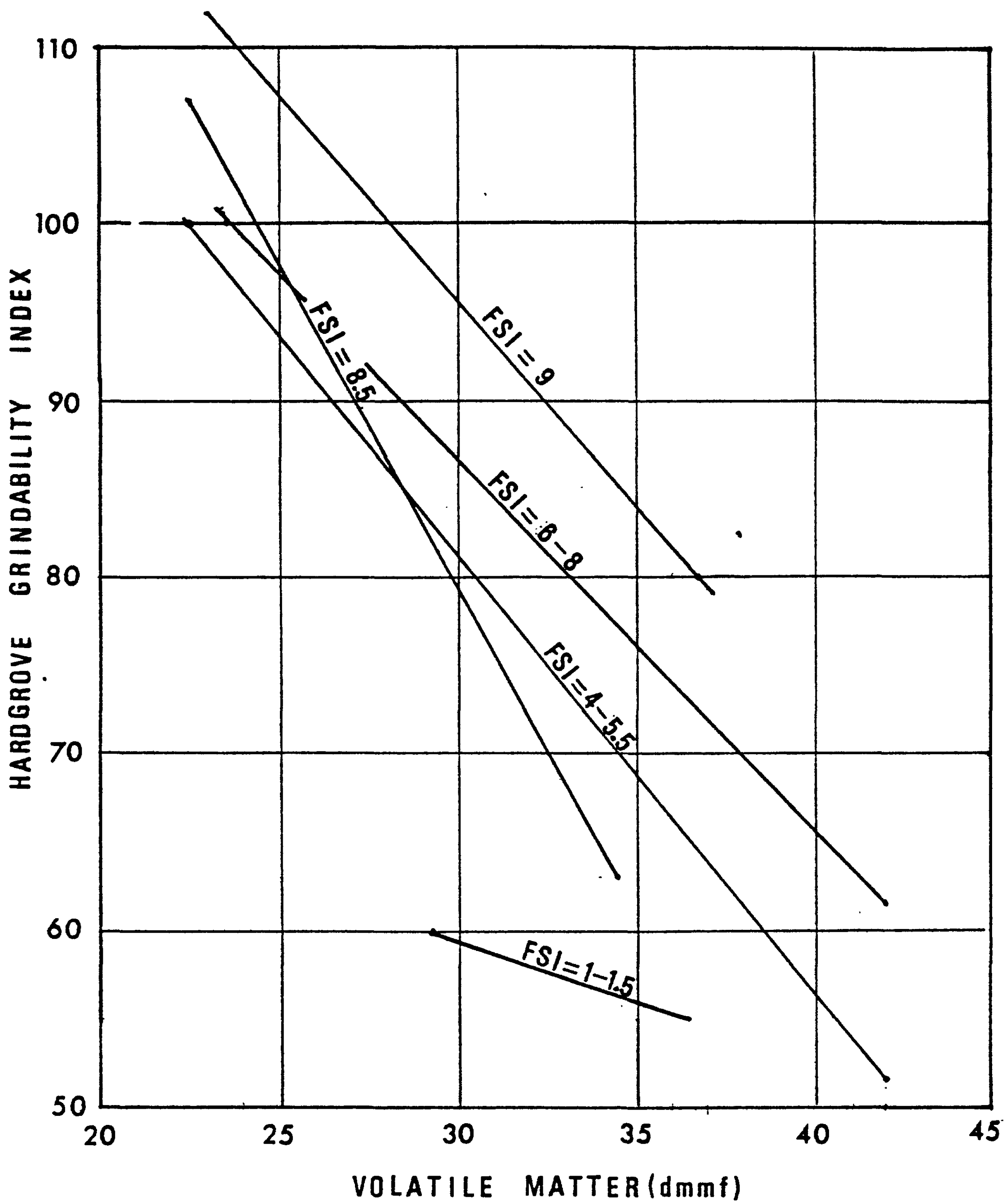

Figure 15.-- Empirical relationship of Hardgrove Grindability Index to percent-content of volatile matter for different levels of free swelling index (FSI). Source of data: Donahue and Leonard (1967, fig. 4, p. 363; fig. 5, p. 364).

With respect to hardness as a specific property, and as related to rank of coal and coking properties, Yancey and Geer (1968, p. 1-25) reported: "Hardness is related to rank and has a minimum value by various means of measurement at about 85 to 90 percent carbon." Brown and Hiorns (1963, p. 127) reported: "A minimum in hardness was also found at 90%C, and a maximum found at around 80%, supporting Van Krevelen's results."

COALIFICATION AND GEOTHERMAL METAMORPHISM

Deposits of Upper Cretaceous bituminous coal are widespread in the western United States and it must be assumed that, during Late Cretaceous time, swamps were extensive and conditions were optimum for the accumulation and preservation of coal-forming materials.

Deposition of coal and coal-bearing rocks commonly takes place in subsiding depositional basins. In the past, substantial thicknesses of overburden rocks commonly accumulated on any coal before a later geologic event arrested further deposition.

Coalification may be described as a long-term metamorphic process in which peat and anthracite are end-members of a series, and intervening ranks of coal represent stages of metamorphism. The processes of coalification involve pressure, heat, and geologic time. In the usual cycle of coal formation an accumulation of vegetable materials growing in a swamp is converted to a peat layer which is buried to increasing depths. As burial continues, overburden pressures, confining pressures, and heat increase. The increasing pressure, and heat, cause compaction and physical and chemical changes which alter the thickness, volume, structure, texture, and composition of the coal (i.e. metamorphism).

It is not within the scope of this report to attempt to classify pressure, heat, and geologic time as to their relative importance in the metamorphic process of coalification. It seems evident, however, that all three are required for coalification to be effective in reaching the rank of low-volatile bituminous coal. Nonetheless, the evidence at hand strongly suggests that coalification involves long-term geothermal metamorphism which may be described as a natural, thermal distillation of volatile substances, including oxygen, from buried coal, over long periods of geologic time.

Thermal metamorphism

The rank of coals in the central and southern parts of the Carbondale mining area locally is higher than the normal regional rank (see fig. 16; area 3). Chemical changes involved in these local advances in rank include (a) decreases in percent-content of moisture, volatile matter, and oxygen, and (b) increases in Btu value and in percent-content of fixed carbon (see table 3 and Appendix C).

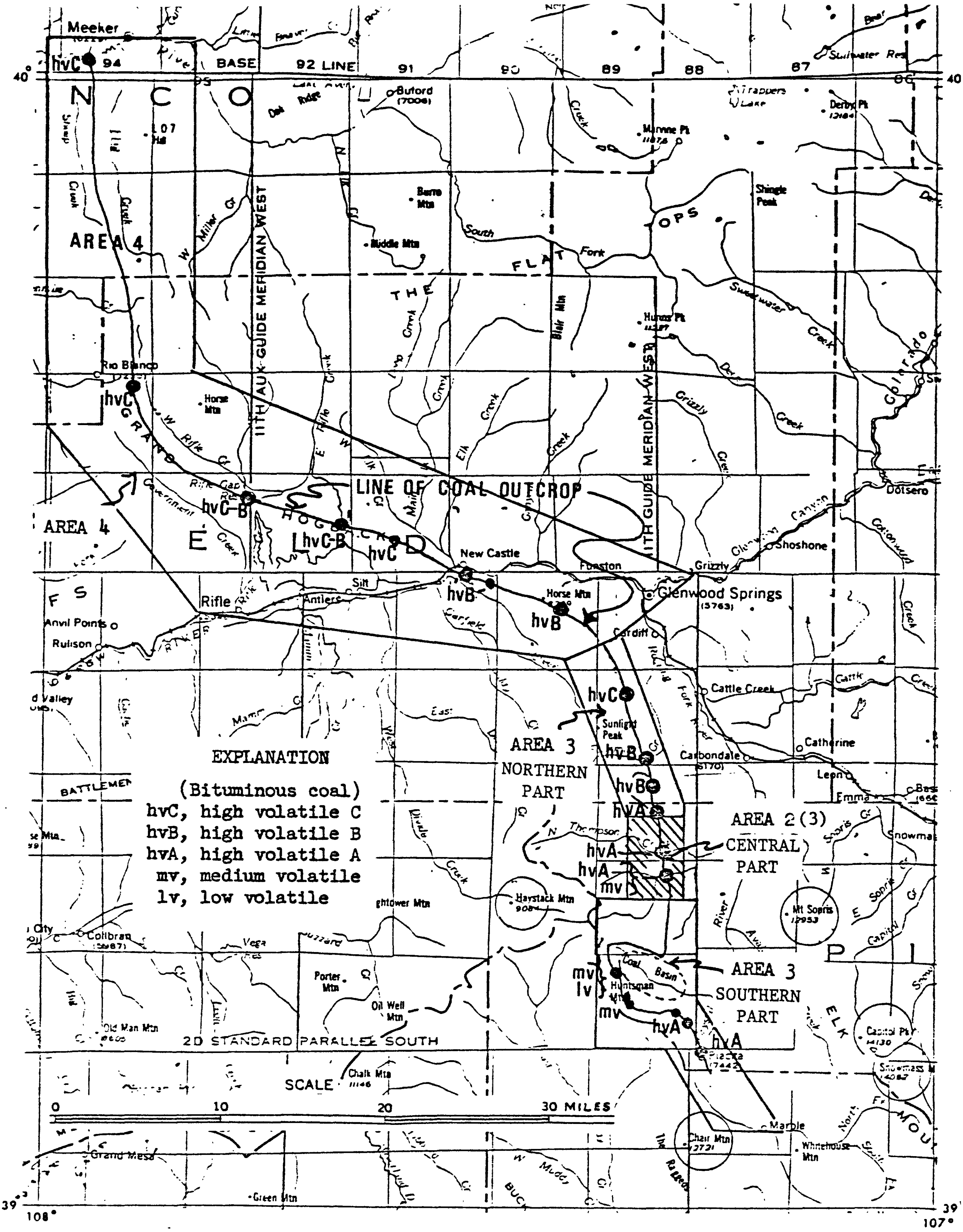

Figure 16.--Regional map showing localities and ASTM ranks of bituminous coal samples from Areas 2, 3 and 4

The most notable feature associated with these chemical changes is that most coals north of the central part of the Carbondale area are noncoking, whereas the coals in the central and southern parts are strongly coking. These changes are also accompanied by physical changes; for example, coals north of the central part have relatively low HGI's, whereas coals in the central part have relatively high HGI's and those in the southern part have extremely high HGI's. It should be emphasized that these chemical and physical changes occur over short distances along the strike of the coal beds (see table 3). This suggests that localized geologic events accelerated the coalification process.

Some regional and local geologic events germane to the coalification process, are as follows:

(1) Coals of the Carbondale area, and of the region shown on figure 16, were deposited in Late Cretaceous time about 75 million years ago. Subsequently, the lower zones of coals in the Carbondale area was buried by at least 9,000 feet of sediments of the Williams Fork and Wasatch Formations, and probably by an additional 3,000 feet or more of sediments of the Green River Formation of Eocene age, which have since been eroded off. Consequently, by Late Eocene time about 40 million years ago the lower zones of coals was buried by 9,000 to 12,000 feet of rocks, and the coalification process had been underway for about 35 million years.

(2) Beginning in Late Eocene time, about 45 million years ago, and continuing through Middle Miocene time (18 million years ago), large-scale vertical movements took place in the northwestern part of the region shown on figure 16. This structural movement resulted in the development of the White River uplift and the Grand Hogback monocline (Murray, 1966, p. 193), a prominent structural feature exposing some 25,000 feet of steeply-dipping rocks. Structural relief of the monocline ranges from about 10,000 feet near Placita, Colorado, to about 27,000 feet near Rio Blanco (Murray, 1966, p. 192). This regional orogenic event arrested further deposition. The coalification process had by then advanced to a regional rank presently indicated as HvC-HvB (see fig. 16).

(3) In Oligocene time 26-38 million years ago, and concurrent with the regional orogenic event described above, the rocks in the southwestern part of the region were intruded locally by igneous bodies of granodioritic and quartz monzonitic composition (Tweto and others, 1976). The temperature of comparable magma is estimated to have been about $800^{\circ}C$ (see Wahlstrom, 1950, p. 154).

The cooled intrusive bodies are now represented by prominent geographic features such as Haystack Mountain, Mt. Sopris, Chair Mountain, Snowmass Mountain, and Capitol Peak (see circled areas on fig. 16).

Subsurface laccolithic intrusions of quartz monzonite are also known to be present in the region; for example, beneath "Coal basin" (see fig. 16) where exploratory oil and gas wells penetrated at least 1,500 feet of quartz monzonite at depths of about 4,000 feet (Donnell, oral commun., 1977). The aeromagnetic map (fig. 17) of the region indicates that the buried intrusive under Coal Basin may be connected to the Mt. Sopris Stock.

(4) During Pliocene and Miocene time (8-23 million years ago), lava flows of alkali basalt were extruded in the area east of Glenwood Springs-Carbondale. Eroded remnants of the basalt flows overlie the coal-bearing rocks in the area southwest of Glenwood Springs. These basalt flows seem to be responsible for the area of relatively high magnetic intensity shown on the aeromagnetic map (fig. 17) in the vicinity of Glenwood Springs.

(5) Thermal springs developed subsequent to the Pliocene volcanism. Glenwood Springs and the area to the south and southeast of that town are well-known for numerous thermal springs (see fig. 18). Some of the hot springs are located in and adjacent to Oligocene-age intrusive bodies, and it would seem that their heat may be derived from nearby igneous rocks that are still cooling.

(6) Also shown on figure 18 are heat-flow contours. The contoured values indicate that, as measured from the surface, the flow of heat from the interior of the earth is still abnormally high in the eastern half of the region shown on figure 18. In this respect, the average rate of heat flow for the earth is 1.5×10^{-6} cal/cm^2/sec. (Pearl, 1972, p. 4), which for convenience is referred to as 1.5 heat flow units (HFU). Any measurement in excess of 1.5 HFU usually indicates a geographically restricted heat source at fairly shallow depths (Pearl, 1972, p. 12). As shown on figure 18, a 2.0 HFU-contour passes through the central part of the Carbondale mining area; higher rates of heat flow are indicated in the region to the east and southeast.

(7) A geothermal gradient map of part of western Colorado (fig. 19) shows that present-day geothermal gradients are steeper than normal (1.8^{o}f/100 ft) in an area including the central part of the Carbondale mining area and to the west. (The gradients were derived from annual surface temperatures and corrected bottom-hole temperature of oil and gas wells.)

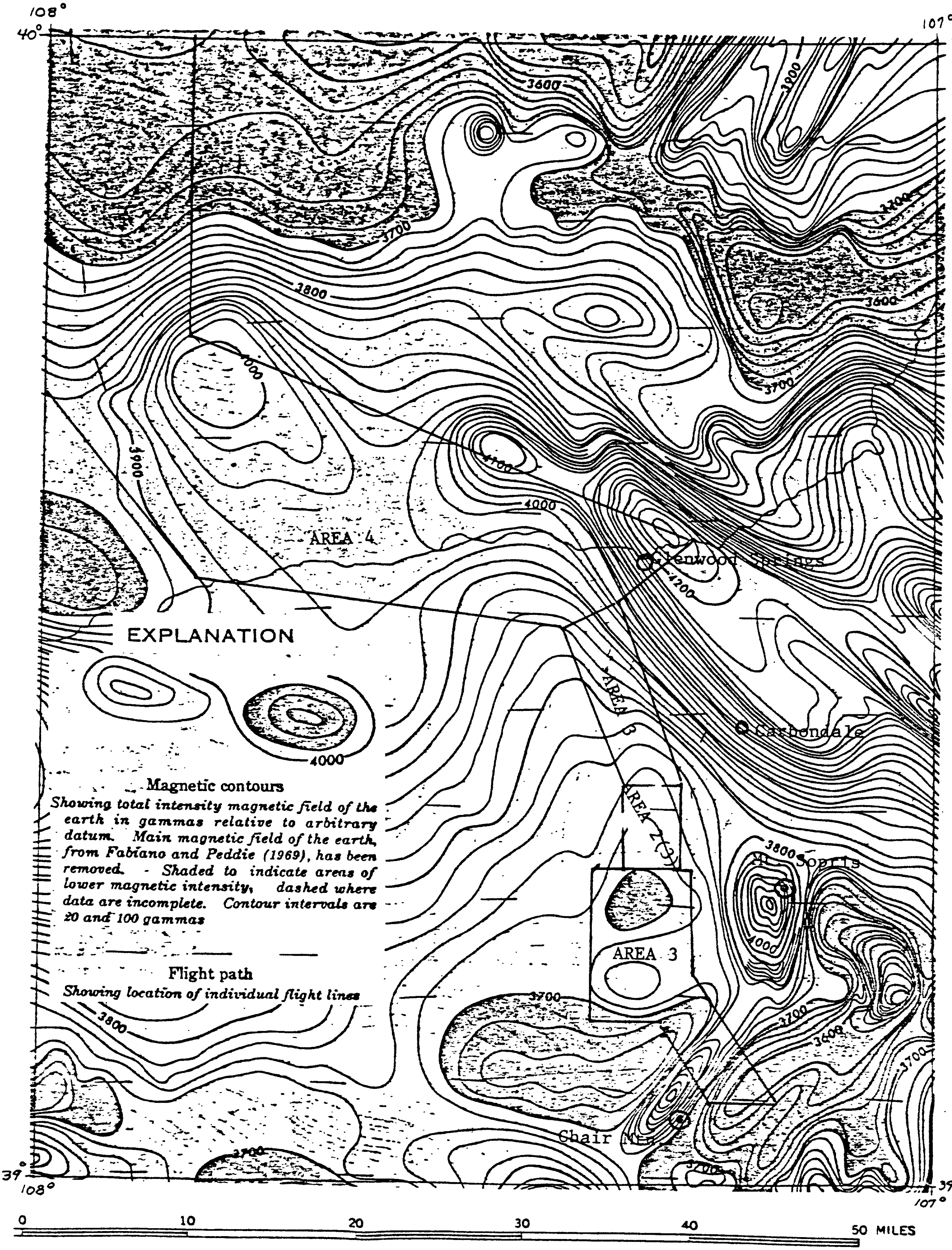

Figure 17.--Aeromagnetic map of part of western Colorado, showing locations of Areas 2, 3 and 4, after Zeitz and Kirby (1972).

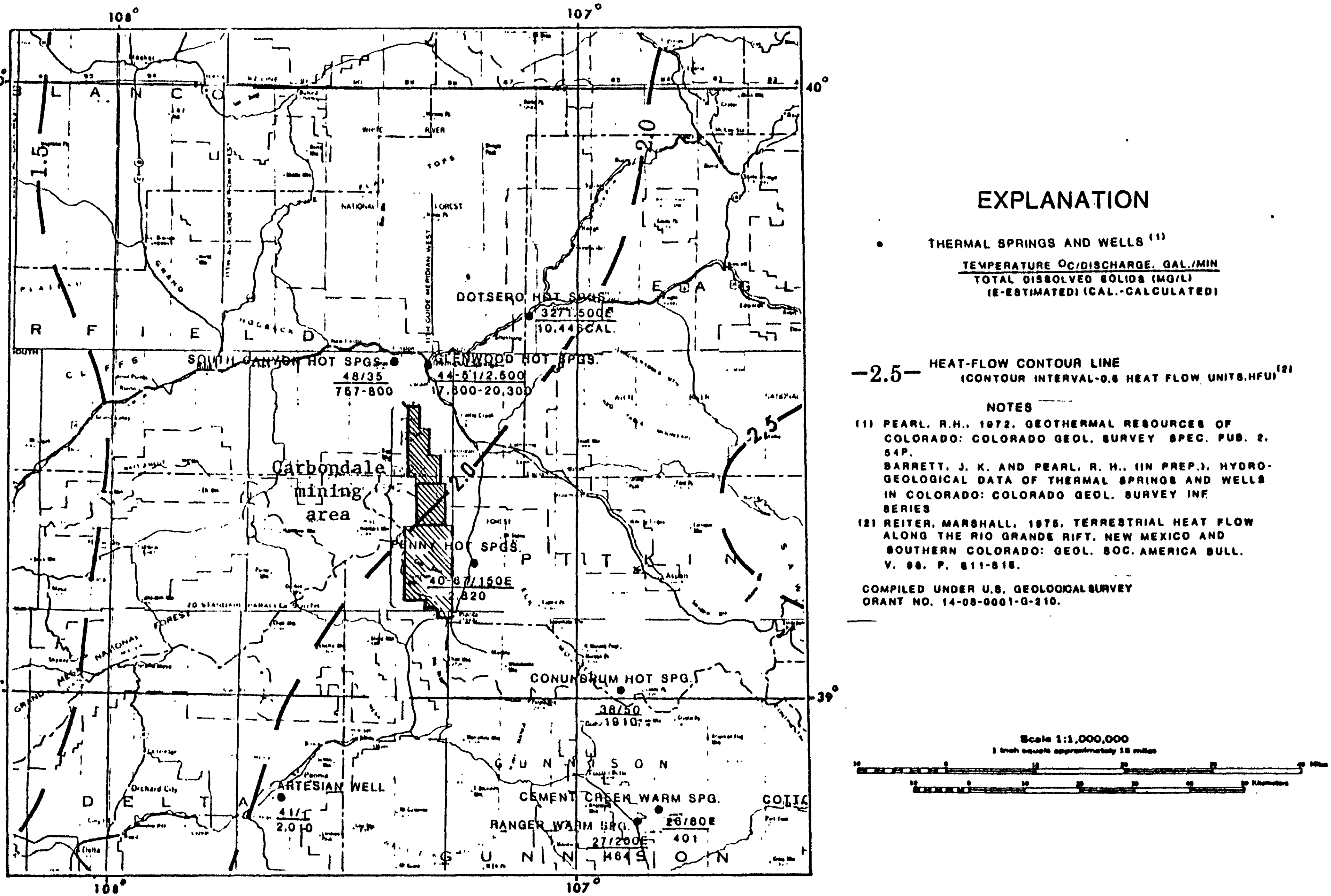

Figure 18.--Map showing Thermal springs, wells and heat-flow contours in part of western Colorado. Data source: Barrett, J. K., Pearl, R. H., and Pennington, A. J., 1976, Map showing Thermal springs, wells and heat-flow contours in Colorado: Colorado Geol. Survey, Information series 4.

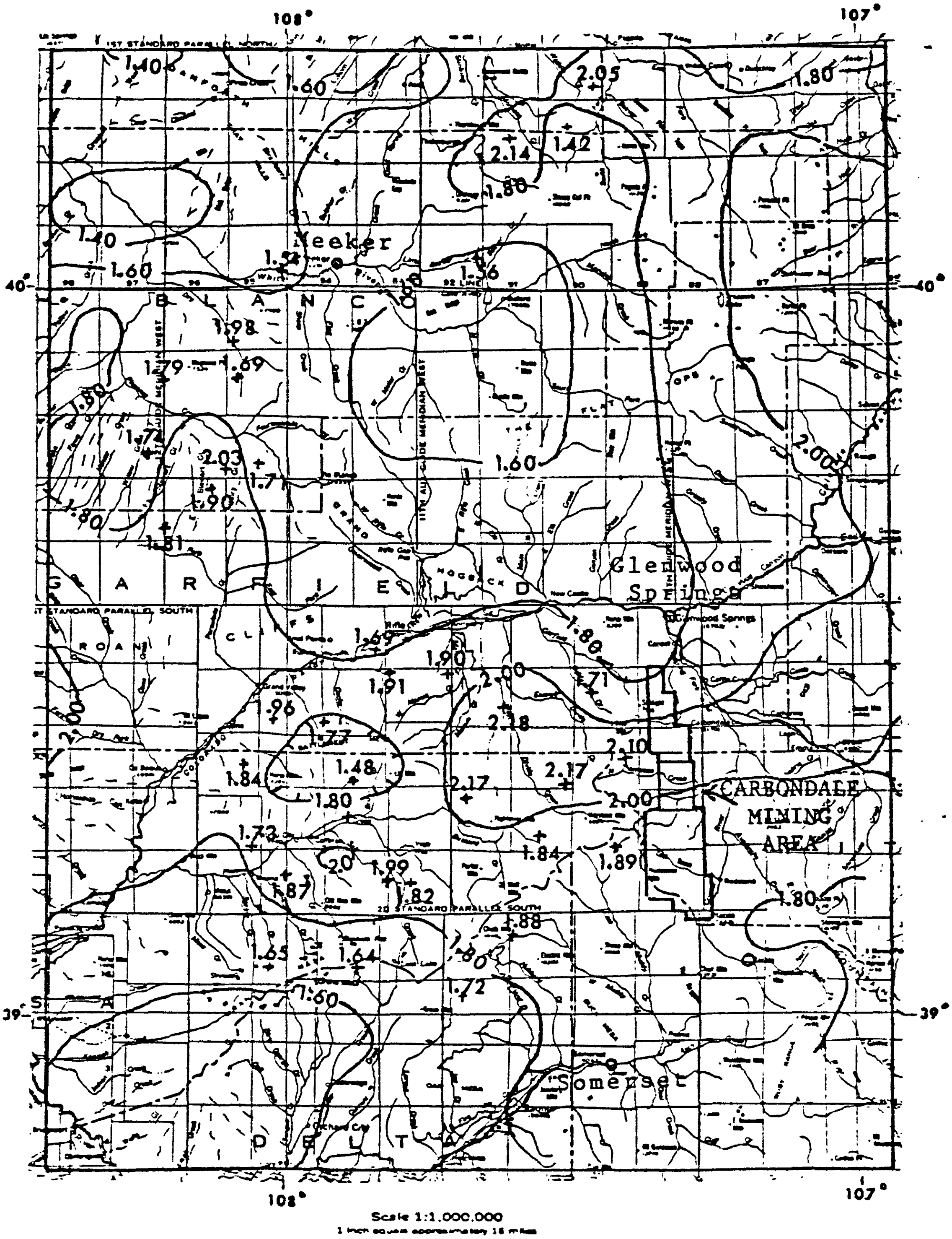

Figure 19.--Geothermal gradient map of part of western Colorado. Contour interval, 0.20°F/100 feet. Data source: computer-generated (1977) Geothermal gradient of Utah-western Colorado, portfolio map area No. 19, Geothermal Survey of North America, American Association of Petroleum Geologists.

(8) Point-locations of geothermal gradients shown on figure 19 correspond to locations of oil and gas wells. Logs of some of these wells were obtained, and data for each well commonly included the bottom-hole temperature encountered. One of the highest subsurface temperatures recorded is in sec. 36, T. 8 S., R. 91 W., about 7 miles west of the central part of the Carbondale area, where instruments in the California Co. No. 1 Hurd Govt. well recorded a bottom-hole temperature of 290° F at a depth of about 12,600 feet. That location is about 3 miles northwest of Haystack Mountain (see figs. 16 and 19) where igneous rocks of Oligocene age are exposed (Tweto and others, 1976). The surface elevation of the No. 1 Hurd Govt. well, 9,815 feet, is stratigraphically near the top of the Williams Fork Formation (see fig. 20). As interpreted from electric logs, (a) about 50 feet of igneous rocks were penetrated from 3,770-3,820 feet, (b) the base of coal bed A was penetrated at a depth of about 4,400 feet (i.e., the elevation of the base of coal bed A is about 5,400 feet), and (c) the thickness of the Williams Fork Formation is probably about 4,600 feet, or about 700 feet thicker than measured along North Thompson Creek, about 8 miles to the east.

(9) As shown on figures 19 and 20, geothermal gradients in the area west of the Carbondale area and northwest of the Haystack Mountain intrusive are steeper than normal, whereas, to the southwest and south, gradients are less than normal (i.e., 1.8° F/100 ft). The steeper gradients suggest a geographically restricted heat source which is still effective, and which may have been more effective in past geologic time. In this respect, it is of interest to note that the above-normal geothermal gradients, and the Haystack Mountain intrusive, occur in association with anticlinal structures trending northwest in the vicinity of Haystack Mountain; the anticline northwest of Haystack Mountain exposes Upper Cretaceous rocks along the crestline (see fig. 20).

From the available evidence, it seems highly probable that local advances in rank of coal in the Carbondale mining area resulted from accelerated thermal metamorphism as evidenced by very steep geothermal gradients caused by localized emplacements of magma masses during and following Oligocene time.

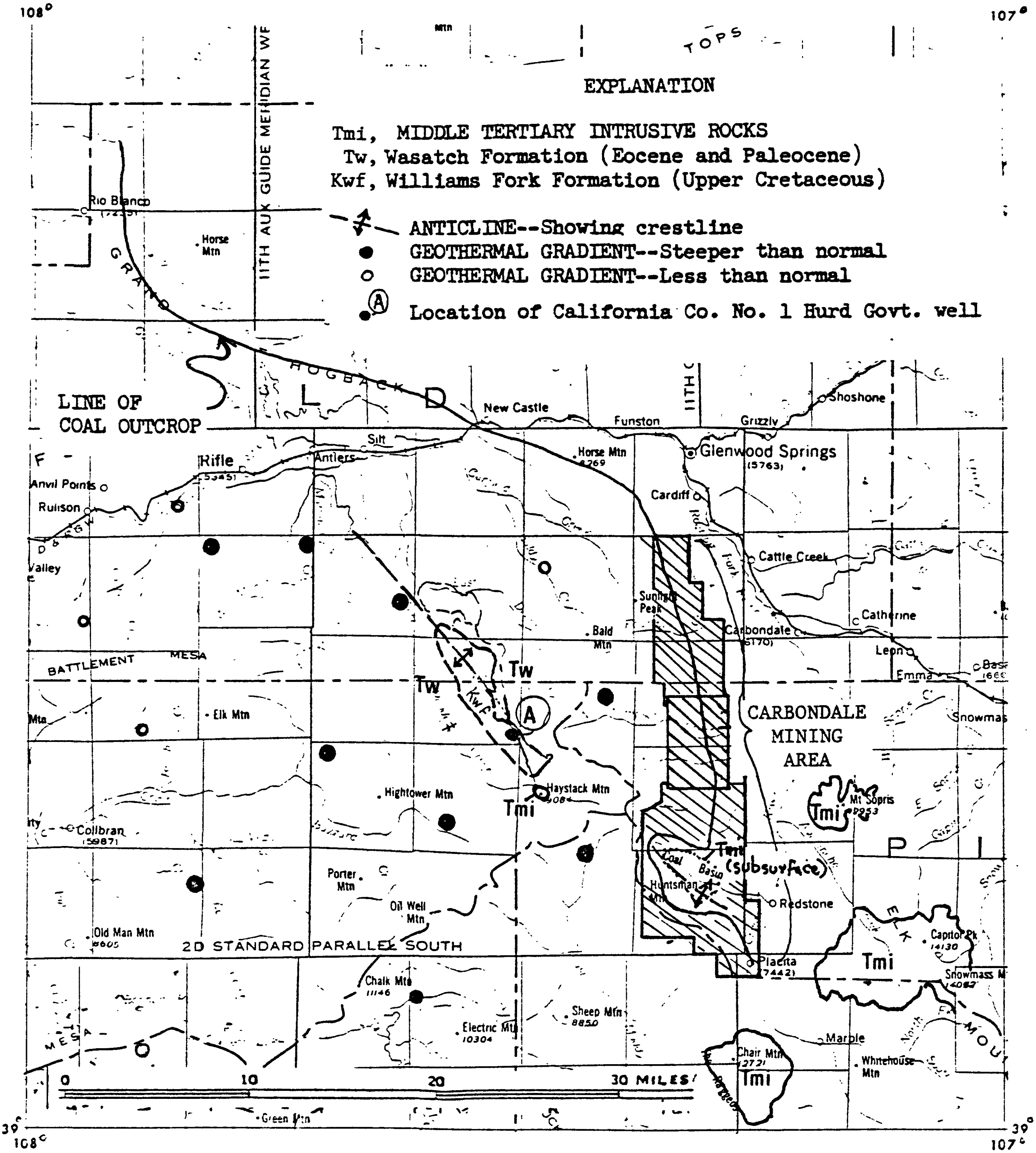

Figure 20.--Map showing areal relationships between anticlines, intrusive rocks, and geothermal gradients in the vicinity of the Carbondale mining area. Geology after Tweto (1976). As used here, Williams Fork Formation (Kwf) is equivalent to Mesaverde Formation.

The span of geologic time between intrusion of igneous bodies and completion of regional coalification (at a regional rank of HvB-HvC) is difficult to estimate. Emplacement of the intrusions which caused the steep geothermal gradients and accelerated the coalification process may have continued from Oligocene into Miocene time (±25 million years ago). Vertical uplifting may not have been appreciable in the southeastern part of the region (see fig. 20) until Late Miocene time (11 million years ago). These age differences suggest a time span on the order of 14-20 million years during which accelerated thermal metamorphism could have been effective. It has been reported that coals of the southern Ruhr Basin in Germany reached Lvb rank (22 percent volatile matter) in about 12 million years, implying a temperature of about 210^0 C at depths of burial of about 7,000 feet with a corresponding geothermal gradient of 1^0 C/12 meters (see Teichmüller, 1966, p. 151).

Overburden rocks of Late Cretaceous age thicken slightly westward and the total thicknesses of overlying Paleocene and Eocene rocks are unknown because of erosion. It seems likely that emplacement of the Oligocene intrusives may have inhibited further deposition, because the intrusives are exposed in the cores of anticlines and because structural relief of the Grand Hogback monocline is nearly three times greater near Rio Blanco than it is near Placita, Colorado. The geographic position of the depositional axis of the overburden rocks is unknown, but is thought to have been some distance west of the Carbondale area. There is no evidence for local advances in rank (beyond the regional HvB-HvC values) having been caused by increased static pressure due to thicker overburden. In fact, the HvB rank of coals in the Somerset area corresponds to the regional rank; the Somerset area is about 20 miles southwest of the Carbondale area (see fig. 1).

SUBSURFACE HYDRAULIC MINING

During the period 1958-1962, the Bureau of Mines conducted five field experiments with subsurface hydraulic mining of bituminous and anthracite coals using high-water pressure monitor jets. In chronological order, the field trials involved the following coal beds.

1) Pittsburgh coal bed, near West Lebanon, Pennsylvania.
2) Bottom Red Ash (anthracite) coal, near Sugar Notch, Pennsylvania.
3) Roslyn No. 5 coal bed, Roslyn No. 9 mine, near Roslyn, Washington.
4) Coal bed "A", Thompson Creek Mine, near Carbondale, Colorado.
5) Roslyn No. 5 coal bed, Roslyn No. 10 mine, near Roslyn, Washington.

Some of the data on the coal beds mined, types of monitor jets used, production rates achieved, and difficulties encountered during the field trials are summarized in table 2. Typical sections of the coal beds mined hydraulically are shown on figure 21.

Subsurface hydraulic mining of coal has been practiced for some time in Russia, Poland, and Canada. Most of the research on hydraulic mining methods and practical experience in such mining has been gained from operations in those countries. A comprehensive four-volume report on the worldwide status of subsurface hydraulic mining was prepared by Cooley (1975) under contract with the U.S. Bureau of Mines. He listed (1975, v. 1, tables 9-1 and 9-2) some United States areas of pitching coal seams thought to have hydraulic mining potential. His tables include the areas listed in the introduction to this report.

Mechanisms of coal failure in hydraulic mining

The mechanisms of coal failure when mined hydraulically need brief description here, as such information serves as background to discussions of physical properties of coal that are thought to be measures of hydraulic mining potential. As described by Cooley (1975, v. 1, p. 5-4), the mechanisms are as follows:

Table 2.--Summary of deta on coal teds on coal beds mined, jet monitors used, and results and experience obteined during field trials of subsurface hydreulic mining conducted by the U.S. Bureau of Mines.

Field Trial	Coal bed; description [1/]	Rank and coking strength	Pitch of bed	HGI	Friability,[2/] percent	Jet monitor used	Max. production rates development mining	Max. production rates pillar mining	Remarks	References
(1)	Pittsburgh coel bed; hard; pronounced face and butt cleats; numerous streake of impurities; hard binders	HvAb, moderate (FSI est 6)	flatlying	84	30 est	3/8"-3D nozzil 4000 psi 300 gpm	[1.6 tpm]		Coal bed 66" thick; 80' overburden	RI 5915; RI 7090; Palowitch (1964)
(2)	Bottom Red Ash coal bed; refuse bands of sandstone and sand slate	Anthrecite	0-20°	37	30	0.394"3D nozzil up to 5000 psi 300 gpm (hydraulic jumbo)	[0.83 - 0.99 tpm]		Coal bed 13' thick;450 ft overburden. Jet-cutting only 25 percent of totel face time. Bottom bench of coal harder to mine than other portions of the bed including the refuse bands.	Palowitch (1964) RI 6610 RI 7120
(3)	Roelyn No. 5 coel bed Roslyn No. 9 mine; veriable hardness; cleavage planes not pronounced	HvAb; week	34-38°		42	0.148" D nozzil 4500 psi 40 gpm (developmental mining: monitor attached to roofjack)	0.10 tpm (5.7 tons/ man-shift) in hard coal 0.51 tpm (9.0 tons/ man-shift in softer coel	0.74 tpm (16.6 tons/ man-shift 0.42-1.91 tpm (with varying roof pressure)	Coal bed 58" thick; overburden ave. 500 ft Productivity depended on coel "hardness". Hydraulic mining more productive than conventional, in pillar mining.	RI 6276 Palowitch (1964)
(4)	Coal bed "A", Thompson Creek mine; containing three pertings of verisble thickness.	Mvb; strong (FSI 7-8)	26°	71-83	Est. 70 Ave.	0.148" D nozzil 4500 psi 40 gpm (monitor attached to jeck)	[Range from 3.5 tons/ manshift in very herd coal to 46 tons/men-shift in soft coel. Ave: 23.3 tons/man-shift, 35% higher productivity than by conventional develop-ment mining. Ave. cutting rate: 0.39 tpm		Coal bed 101" thick: Est. 800 ft overburden Coal hardness varies but generally soft and friable Jet cutting only 31% of total face time "during 51 production shifts 2830 tons of coal wee mined driving a pair of reises 350 ft. up the pitch". (Palowitch, (1964, p. 72)	Palowitch (1964)
(5)	Roslyn No. 5, Roslyn No. 10 mine; parting in lower pt. of bed; cleavage planes not pronounced	HvAb; weak	42°		42	Raise mining: 1/4"D nozzil 4000 psi 128 gpm Room mining: 3/8" D nozzil 4000 psi 230 gpm (remote controlled monitor)	[4.7 tons/man-shift in raise mining compared with 7.5 tons/man-shift by conventional. Jet cutting rate: 0.73 tpm] [5.2 tons/man-shift in room mining, compared with 8.5 tons/man-shift by convantional. Jet cutting rate: 0.55 tpm]		Coal bed 64" thick: 900 ft overburden Roslyn No. 5 coal harder in No. 10 mine than No. 5 coal in No. 9 mine. Jet cutting, raise mining, only 12% of total face time. Remote controlled jet cutting, room mining, only 19% of total face time; trouble with single-acting roof jecks	RI 6685 Palowitch (1964)

1/ Typical sections of the coal beds mined are shown on figure 21 this report.

2/ Friability data from Yancy (1932) USBM Tech. Paper 512.

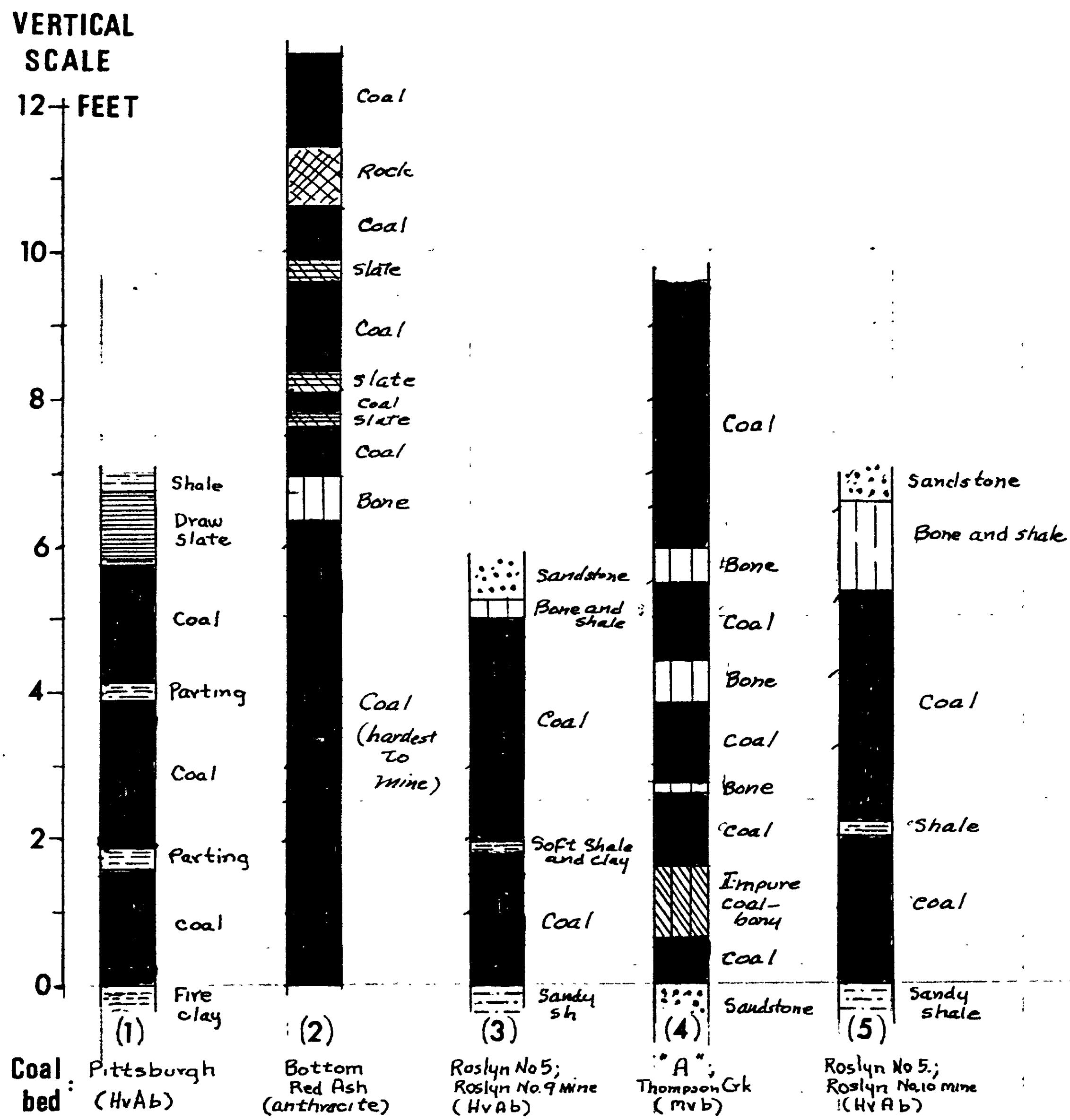

Figure 21.--**Typical sections of coal beds mined hydraulically in USBM field trials of subsurface hydraulic mining. Sources of data: section (1), USBM RI-7090; (2), RI-6610; (3), RI-6276; (4), this report; and (5), RI-6685.**

"When a high pressure jet is directed against coal, there are two main mechanisms of failure of the material. The first is granular erosion and the second is fracture (called hydrobreaking) under shear or tensile stresses aided by water permeation into pores and cracks. The erosion mechanism is quite inefficient because it breaks coal into very small grains with high specific energy because of the large surface area produced per unit volume. Hydrobreaking disintegrates coal into larger pieces with lower specific energy. Therefore coal disintegration by a continuous jet of small diameter (less than 5 mm) is best accomplished by cutting slots of minimum total volume with rapidly traversing jet, followed by breakage of the coal between slots by the action of water pressure, gravity, roof pressure stresses, mechanical methods, or combinations. Jet cutting of slots should be accomplished with a standoff distance from nozzle to coal of less than 100 nozzle diameters, if possible."

Coal properties that influence hydrobreaking production rates are described by Cooley (1975, v. 1, p. 5-5) as follows:

"The primary properties of coal affecting its ease of disintegration are the coal hardness (which is related to compressive strength) and its permeability to water (which is affected by porosity, joints, cleats, lamination planes, plasticity, the state of stress, and the direction of water permeation). In general, coals with an intermediate degree of metamorphism are easier to break than coals of low or high rank, but there are exceptions. Breakage of coal from a free surface is generally more effective if the jet is directed approximately parallel to the plane of cleats and laminations. If the jet is directed at 90° to the cleat plane, the breakage rate is decreased by 55 to 65 percent."

"As a first order approximation, it was initially found that the minimum jet pressure required to cut coal effectively should exceed a value of about one half the compressive strength or 50 times the Protod'yakonov hardness number F (which is approximately 1 percent of the unconfined compressive strength). However, exceptions to this rule were found and therefore a water infusion test method was developed in the USSR to measure the ability of coals to be broken by a monitor jet."...

Coal hardness

Cooley (1975, v. 1, p. 5-5) reports that, in Russian experience, hardness is a physical property of coal that affects its ease of disintegration and thereby influences hydrobreaking production rates in subsurface hydraulic mining. Variations in production rates achieved during United States field trials of hydraulic mining were also attributed to variations in coal hardness (see table 2, this report).

As a specific physical property of coal, hardness refers to the dynamic microhardness of vitrain layers; measurements are made by Vickers and Shore scleroscope methods; and such data commonly are used as a means of comparing changes in that property with different ranks of coal (Brown and Hiorns, 1963, p. 127). Yancey and Geer (1968, p. 1-25) compiled data on specific values for dynamic hardness of vitrain layers in coals of various types, their data are summarized as follows:

Type of coal	Volatiles (percent)	Shore hardness number 1/	Vickers hardness number 2/
Anthracite	6	122	54
Bituminous (noncaking)	21-42	107-109	24-59
Bituminous (caking)	40-44	105-107	22-26
Bituminous (coking)	21-36	44-65	9-16

1/ Shore number represents dynamic deformation by rebound.

2/ Vickers number shows static deformation, by size of identation formed when load is applied to steel ball or diamond in contact with the coal.

The tabulation of hardness of vitrain layers in coals indicates that, characteristically, Mvb and HvAb coking coals have minimum hardness, but the tabulation emphasizes that relatively slight changes in composition or in coking strength effect appreciable changes in the hardness of vitrain layers (which commonly comprise the bulk of the coal seam).

Hardness has also been related to carbon content of coal, in that minimum hardness is found at about 85 to 90 percent carbon and maximum hardness, at about 90 percent (Yancey and Geer, 1968, p. 1-25; Brown and Hiorns, 1963, p. 127). These findings emphasize that relatively slight changes in coal composition can result in rather drastic changes in hardness.

In Russian usage, the term "hardness" seems to be synonymous with (unconfined compressive) strength of coal as derived from measurements of strength in kg/cm^2. In this respect, Cooley (1975, v. 1, p. 3-2) defines hardness as follows:

> hardness--the resistance of a material to crushing. Usually Soviet data on coal and rock is reported in terms of the Protod'yakonov hardness "F," which is approximately 1 percent of the unconfined compressive strength in kg/cm^2.

However, the definition introduces some ambiguity as to (a) the specific properties of coal involved, (b) methods used in laboratory testing, and (c) the interpretive value of test results as a basis for predicting the hydraulic mining potential of the parent body of coal.

As a specific property, the (unconfined) compressive strength of coal has been tested in U.S. laboratories, and measurements commonly are expressed in lb/in^2 (psi). Equivalents to Protod'yakonov hardness "F" numbers might be derived through conversion of lb/in^2 to kg/cm^2 (e.g., multiply by 7.03×10^{-2}) wherein 1 psi is approximately equivalent to 0.07 kg/cm^2. But a quantitative approach of that nature, if applied in predicting hydraulic mining potential, is subject to qualification in both Russia and the United States. For example, in context with a discussion of variations in compressive strength of coal, Brown and Hiorns (1968, p. 128) refer to Protodyakonov (sic) as having shown "by empirical tests, that strength varies by as much as 30 percent over the width of a coal seam." In the United States, for example, it has been found that laboratory measurements of unconfined compressive strengths of coal samples do not relate directly to compressive strengths of the parent bodies of coal. In fact, results of laboratory testing show that (a) compressive strength increases markedly as the size of the sample tested decreases, and (b) compressive strength measured when the crushing load is applied across, or perpendicular to bedding planes is generally less than measured when load is applied along, or parallel to bedding planes (see Yancey and Geer, 1968, table 1-6). To illustrate increases in compressive strength with decreasing sample size, with load applied perpendicular to bedding planes, results obtained by Rice and Enzian (1929) are tabulated as follows:

Coal	Size of specimen	Compressive strength (lb/in^2)
Bituminous	54 in.	310
	30 in.	820
	12 x 12 x 18 in.	1,150
	10 in. - 12 in.	2,010
	7 in. - 8 in.	2,170
	2½ in. - 4 in.	2,490

Nonetheless, a general relationship between compressive strength and coal rank has been noted, in that minimum strength is commonly reached at 20 to 25 percent volatile matter (Yancey and Geer, 1968, p. 1-26).

The following quotation from Brown and Hiorns (1963, p. 130) is of interest here:

> "The variation of compressive strength with rank of British coals was given by Pomeroy and Foote, and a plot of strength against volatile matter shows the customary minimum at 20 to 25 percent dry, ash-free volatile matter (cf., fig. 4) for compression both perpendicular and parallel to the bedding planes [Reference]. They also obtained figures for cleat frequency for the nine coals tested, and a plot of their data shows that compressive strength falls off as cleat frequency increases."

The definition of hardness provided by Cooley (i.e., the resistance of a material to crushing) suggests that hardness is also a function of "grindability" which has been described as "a composite physical property embracing other specific properties such as hardness, strength, tenacity, and fracture" (Yancey and Geer, 1968, p. 1-31).

The ASTM standard test method most commonly used is the Hardgrove machine method (D 409-71) which determines the relative grindability or east of pulverization of a coal sample, as compared to samples of coals selected as standards; results are expressed as an HGI number which is inversely proportional to the east of pulverization of the sample tested. But again, the use of HGI's as a basis for predicting hydraculic mining potential is subject to qualification.

The ASTM standard test method (D 409-71) uses initial sample-size fractions of 0.6 to 1.2 mm. The purpose of the test is to determine the relative ease with which such minute size-fractions can be ground fine enough to use as pulverized fuel (i.e., <200 mesh). It should be noted that the dimensions of the initial sample-size fractions are considerably smaller than are the dimensions involved in, for example, cleat spacing--and it is evident that the HGI for a given coal sample is by no means a direct measure of overall strength, tenacity, or fractured condition of the parent block of coal.

With allowance for connotative differences between crushing versus pulverization, the HGI does provide a direct measure of the granular erosion potential of a coal to be mined hydraulically. But HGIs are not direct measures of hydrobreaking potential.

Granular erosion, however, is a very inefficient mechanism of coal failure (Cooley, 1975, v. 1, p. 5-4). Apparently, the degree of granular erosion achieved is somewhat incidental to the success, failure or effectiveness of hydraulic mining.

Unless hydrobreaking of coal at the face can be accomplished effectively by combined action of water pressure, gravity, roof pressure stresses, and mechanical methods, a hydraulic mining operation will not be successful.

Friability of coal

Friability of sized coal is the complement of size stability, and both are recognized as composite physical properties embracing specific properties such as toughness, elasticity, fracturing, and strength. These composite physical properties are expressed by the degree of breakage that occurs in mining, screening, and subsequent handling and, in testing, by the degree to which a given coal will shatter on impact. In these respects, friability and size stability are measures of hydrobreaking potential.

The ASTM drop shatter test method (D440-49), for example, determines the relative size stability of lump coal (size fractions up to 4 in. diameter). Results are expressed in percent, and friability (in percent) is equal to 100 minus size stability. In a similar way, the ASTM tumbler test method (D441-45) determines the relative friability of 1.06-1.50 inch size-fraction coal by tumbling for 1 hour, as a simulation of breakage on handling. After tumbling, weight-percentages of resulting size fractions are determined, factored, totaled, and subtracted from 100 to arrive at percent-expressions of friability. By this method, for example, coal determined to have a friability of 70 percent is considerably more friable than coal having 30 percent friability.

Specific data on friability of Carbondale coals were not found in the literature. However, some indications of relative friability of various coals in various parts of the Carbondale area were obtained from observations made in the field.

As observed in the Thompson Creek mines near North Thompson Creek, the friability of coal bed A seemed lower than that of coal bed A in other localities, whereas, the friability of coal bed B and that of the Anderson coal bed seemed consistently high. In the Dutch Creek mine in the southern part of the Carbondale area coal beds in the lower coal zone were judged to be extremely friable, as it was extremely difficult to obtain a 4-inch lump size from those coals (Landis; oral commun., 1977). In contrast, the friability of the Dutch Creek coal, representing the middle coal zone at that locality seemed relatively low.

The Bureau of Mines has made friability tests for various types of coals from various parts of the United States. For example, as reported in U.S. Bureau of Mines Technical Paper 512 (Yancey, 1932), some 235 friability determinations were made for coals from Washington, Pennsylvania, West Virginia, Maryland, Kentucky, Illinois, North Dakota, Montana, and Wyoming; the data reported are summarized fas follows:

Friability values of mine samples of coal

Rank of coal	Number of tests	Average friability (percent)	Range (percent)
Low-and medium-volatile bituminous	27	70.4	90.6 - 53-5
High-volatile bituminous	87	43.3	83.9 - 14.1 (splint)
Anthracite	36	33.2	51.6 - 24.0
Subbituminous A (<20 percent moisture)	40	30.2	56.1 - 20.7
Subbituminous B (<20 percent moisture)	29	20.3	31.1 - 11.5
Lignite	16	12.1	16.7 - 8.6

Methods and procedures for determining the friability values tabulated above are described in detail in U.S. Bureau of Mines Technical Paper 512. It should be noted, however, that (a) tumbling was for a 3-hour period rather than for the presently recognized ASTM standard of 1 hour, (b) initial size fractions used were 1.05-1.50 inch; and (c) the formula used to determine percent friability was as follows:

$$\text{Friability, percent} = 100 \times \frac{D - d}{D}, \text{ where:}$$

D = average diameter of pieces before test

d = average diameter of pieces after test

General relationships of friability to various aspects of coal composition were noted and reported by Yancey (1932, p. 37-39); these relationships include:

1) Coals having less than about 3 percent moisture content (a.r.) tend to have the highest friability.
2) Friability decreases as moisture content increases.
3) Friability increases with increasing Btu values for bituminous coal; on the average, the sharpest increase in friability occurs for coals having Btu values >14,000.

4) On the average, friability increases as fixed carbon content increases to 75 percent; coal containing 75 percent fixed carbon has about 70 percent friability.

Even though there is a general relationship between friability and rank of coal, coals of the same rank may vary widely in friability. For example, Yancey and Geer (1968, p. 1-31) report that Bureau of Mines tests have shown that "the high-volatile A bituminous coals of Alabama range in friability from a low of 17 percent to a high of 60 percent."

Yancey (1932, p. 35) noted that HvAb coal from the Pittsburgh coal bed in southwestern Pennsylvania has the well-known property of withstanding mining without excessive degradation, and the average friability of such coal is 31.1 percent. In contrast, Lvb coal from the Pocahontas No. 3 bed of West Virginia has friability as high as 70 percent, and it was difficult to obtain a sample of such coal of sufficient size for testing.

In connection with difficulties encountered in sampling the Pocahontas No. 3 coal for friability testing, Yancey, (1932, p. 35) quoted from field notes made by Thomas Hendricks, geologist, U.S. Geological Survey, as follows:

> "In sampling Pocahontas coal No. 3 at different places and noting its character at many points, I have come to the conclusion that it is very nearly impossible to secure lumps of the 1 inch to 1½ inch size used in the friability test for more than about 60 percent of a section of the coal bed. The development of vertical jointing and horizontal planes of parting is so great in parts of the bed as to make it extremely difficult to break out a small lump. However, some parts of a single section of the bed are sufficiently solid to yield small lumps. Thus it is apparent that the testing is on the less friable parts of the bed only, and that the remainder of the bed must be considerably more friable.
>
> "In addition to this difficulty, only about 60 to 70 percent of the coal extracted from a mine comes from advance workings. The remaining 30 to 40 percent is left as pillars. When extracted, this pillar coal is distinctly more friable than coal from advance workings. The increase in friability in pillar coal is not constant but depends on such factors as the size of the pillars, the amount of the overburden, the spacing of the pillars, and the age of the pillars as such. It would be extremely difficult to make a standard comparison between coal from a heading and the somewhat crushed coal of the pillars."

Qualitatively, friability of coal can be estimated from the lump size produced on mining, and in this connection Yancey (1932, p. 35) reports that, from observations in other coal fields, "mine operators know that they obtain less lump size from pillar coal than from advance workings."

As observed in Thompson Creek mine No. 1 in the south wall of North Thompson Creek, coal bed A exhibited some tendency to break into lump sizes on mining. In that locality the rank of coal bed A seems borderline between Mvb and HvAb (see fig. 22). To the north, coal bed A is expected to decrease in rank, and variations in friability noted for HvAb coal may be encountered. A Bureau of Mines hydraulic mining test was conducted in Thompson Creek mine No. 2 in the north wall of North Thompson Creek on coal bed A (see table 2). Production rates achieved varied considerably (see table 2; field trial no. 4); probably because of variations in friability, as well as in coal hardness.

Cleats and cleating

Cleats and cleating may be classified as physical properties of coal. The degree of cleat development in a coal bed is a measure of its hydrobreaking potential. For this reason a specific objective of the field surveys for this report was to observe cleats and cleating in the Carbondale coals. By way of background:

1) Cleating, as expressed by intersecting sets of vertical fractures cutting layered coal, is a form of jointing and cleats are universal features of bituminous coals just as joints are universal features of sedimentary rocks.

2) Cleat sets commonly intersect at approximately 90 degrees. Cleats belonging to one of these sets commonly are more planar and better developed than are those belonging to the other. In mining, cleats belonging to the better developed set commonly are referred to as "face cleats," as distinguished from "butt cleats" which identify the other set. As a term, (face) cleat dates back to the time when pick-and-shovel coal miners first recognized that bituminous coal was easiest to mine when the face of the coal was parallel to the direction of the better-developed cleat set because, when hit with a pick, the coal spalled off the face along lines parallel to that cleat direction.

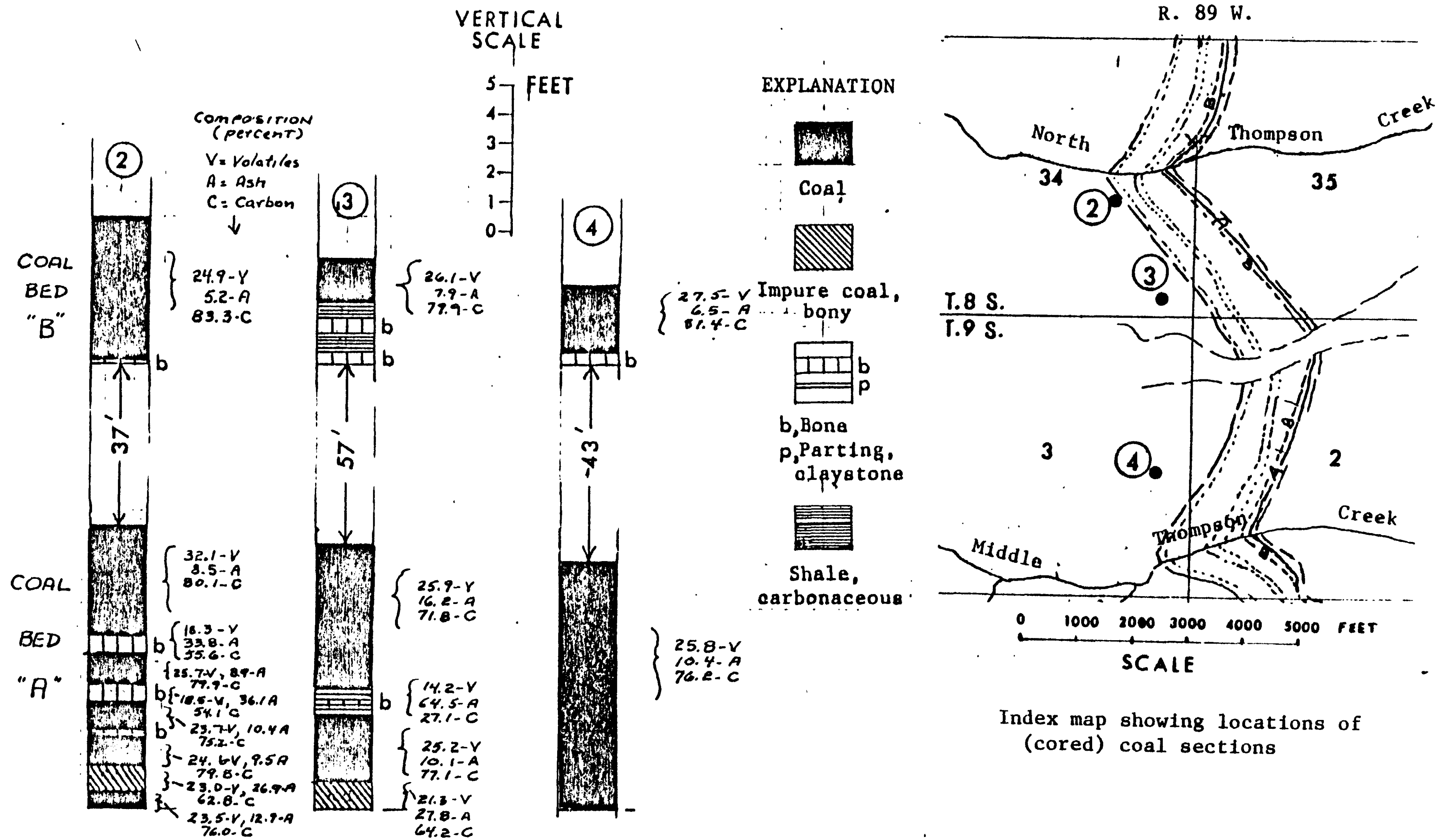

Figure 22.--Lateral (north - south) variations in composition of layers comprising coal beds A and B, lower coal zone, central part of Carbondale mining area. Analytical data obtained from cored coal samples.

3) In areas where the coal beds have been folded, and are now inclined, the direction of face cleats commonly is approximately perpendicular to fold axes, or perpendicular to the strike of a coal bed. Generally where a coal bed has been folded, the verticality of the cleating has been preserved, in other words the cleat-planes have also been inclined so as to remain perpendicular to the coal they cut. This relationship is commonly cited as evidence that cleating pre-dates folding.

4) Face cleats generally are most pronounced, most apparent, and most closely spaced as vertical planes cutting layers of bright coal.

These general features of cleats and cleating are characteristic of the coals in the Carbondale area. However, it was found that the degree of cleat development in the coals varies from one locality to another, and from one coal bed to another at a given locality.

In the central part of the Carbondale area, at a locality near North Thompson Creek (see fig. 6) cleating is well developed in both the Anderson coal bed (upper coal zone) and coal bed B (lower coal zone). At that same locality, however, cleating was not well developed in coal bed A (lower coal zone). Faces of coal bed A were studied in Thompson Creek Mine No. 1, and it was difficult to determine which cleat set (i.e. face or butt) was predominant.

In the southern part of the Carbondale area, in the Dutch Creek mine no. 2 (see fig. 7 for location) the degree of cleat development in the Dutch Creek coal bed is greater than that of coal bed A in the Thompson Creek mine, but the degree varies from place to place. Cleat development in coal beds A and B in other Dutch Creek mines was not studied by the authors; however, those coals were reported to have very high "friability" which is indicative of extensive cleating and fracturing.

Cleating in coals may be either a developed or a latent condition. Many of the cleats in coals are thought to be incipient prior to mining. Cleating and cleat development may be enhanced by intensive structural activity following the deposition of coal in a specified area (i.e. the southern part of the Carbondale area); enhancement of cleating would become apparent when mining commences.

The tendency for a coal to cleat is a measure of cleating potential. This tendency is manifested by cleat spacing in coals.

Cleat spacing

Quantitative determinations of cleat spacing and cleat development in various types of coal require careful examination under laboratory conditions. The only published report of data of that nature, found in the literature, concerned data obtained from systematic studies of face cleat frequency made for some bituminous coals in England. However, Dr. Francis T. C. Ting, Associate Professor of Geology, West Virginia University, Morgantown, West Virginia, is currently conducting research on the origin and spacing of cleats in coal beds. A recent visit with Ting provided opportunity for discussion of cleat spacing in context with hydrobreaking potential, and in subsequent correspondence he (written commun., 1977) shared some preliminary results of his research work.

With respect to some general aspects of cleat spacing in coals, Ting noted that although variations in cleat spacing are controlled by rank and petrographic composition of the coal, cleat frequency increases with increasing rank, reaching a maximum for Lvb coal; but within the same coal bed at the same mine face, dull coal layers tend to have fewer cleats than bright coal layers, and the number of cleats per a standard horizontal measurement varies from bed to bed and even within the same coal. More specifically, Ting (written commun., 1977) has found that Lvb coals have the most numerous cleats per a standard 10 centimeter measurement and that cleat spacing is on the order of 1 to 2 millimeters. Dull coal layers of an Mvc coal bed in West Virginia have approximately three cleats per 10 centimeters (i.e. cleat spacing is about 30 mm) whereas the bright coal layers contain more than 80 cleats per 10 centimeters (about 1 mm spacing). For Pittsburgh coal in West Virginia, a dull layer has 2-3 cleats per 10 centimeters whereas a bright coal layer at the same mine face has 75 cleats per 10 centimeters.

Of particular interest in Ting's (written commun., 1977) discovery of primary and secondary sets of <u>face</u> cleats. The following details some of his findings. Primary cleats usually cut through <u>all</u> coal layers, whereas secondary cleats either terminate at the contact of bright and dull layers or exhibit apparent offset (no lateral movement). The spacing of primary sets of cleats is uniform and more widely spaced than the spacing of secondary sets which are irregular. Generally coal tends to break along primary cleats into large blocks, with the size of the blocks being governed by petrographic composition and rank.

The degree of primary cleat development is a primary measure of hydrobreaking potential.

The research work of Ting supports, illustrates, and adds geographic dimension to results obtained from systematic studies of cleat frequency in the Yorkshire coal field in England. Data reported and illustrated by Williamson (1967; fig. 8.33 after Macrea and Lawson, 1954, fig. 2) are summarized as follows:

Cleat frequency and spacing in bituminous coals, Yorkshire coal field, England

		Number of cleats per horizontal foot		
Seam	Coal types 1/	Range	Average	Average cleat spacing
1	Bright coal (3)	8 - 48	19	0.6 in. (15 mm)
	Dull coal (1)	7	7	1.7 in. (43 mm)
	Inferior coal (2)	4 - 15	10	1.2 in. (30 mm)
2	Bright coal (1)	15 - 70	43	0.3 in. (8 mm)
	Cannel coal (1)	12	12	1.0 in. (25 mm)
3	Bright coal (1)	13 - 18	16	0.8 in. (19 mm)
	Inferior coal (3)	20 - 35	30	0.4 in. (10 mm)
4	Bright coal (3)	12 - 20	17	0.7 in. (18 mm)
	Interbanded coal (chiefly dull) (1)	9 - 13	11	1.1 in. (28 m)
5	Bright coal (1)	12 - 24	18	0.7 in. (17 mm)
	Interbanded coal (chiefly dull) (1)	10	10	1.2 in. (30 mm)
6	Bright coal (1)	12 - 24	18	0.7 in. (17 mm)
	Interbanded coal (chiefly dull) (1)	10	10	1.2 in. (30 mm)

1/ Number in parentheses following coal type refers to the number of layers of that type of coal for which cleat determinations were made.

HYDROBREAKING POTENTIAL

The hydrobreaking potential of Carbondale coals is expected to vary considerably. Noncoking coals in the northern part of the area are high-volatile C and B; coking coals in the central part of the area are high-volatile A and medium-volatile; and strongly coking coals in the southern part are medium and low-volatile. Each of those types or ranks of bituminous coal has properties which either promote or inhibit hydrobreaking.

Estimates of hydrobreaking potential are at best predictions of how effectively specified blocks or units of underground coal will break up under the cutting action of high-water-pressure monitor jets. In addition to properties of coal, a variety of other factors either promote or inhibit hydrobreaking. These factors include: mode of occurrence (i.e., virgin coal vs. pillar coal); degree of cleat development and fracturing; number, thickness, and spacing of non-coal partings within a coal bed; and the degree a specified block of coal is permeable to water. The cumulative effect of factors that promote hydrobreaking may predominate over those that are inhibitive, and the reverse is also true.

The hydrobreaking potential of (Carbondale) coals can be evaluated in a general way by (a) examining some of the factors listed in the preceding paragraph, (b) comparing the degree of hydrobreaking potential estimated before mining to the degree of actual hydrobreaking achieved during field trails of hydraulic mining, and (c) utilizing the results, experience, and information obtained during the field trials, as a general guide.

Statistically, medium and low-volatile coals are much softer, much easier to grind, and much more friable than are high-volatile C and B coals. High-volatile A coals have a wide range of hardness, grindability, and friability. On the basis of these two generalizaitons, the medium and low-volatile coals of the Carbondale area should have the highest hydrobreaking potential; the high-volatile C and B coals should have the lowest; and that of the high-volatile A coals cannot be estimated in the absence of specific analytical data.

It should be realized, however, that analytical data are derived from samples of coal thought to be representative of an area or locality of coal occurrence. Lateral variations in type of coal are numerous and pronounced in the Carbondale area. Even if samples were obtained from a block of coal that is to be mined hydraulically, they represent only small parts of the block. Consequently, estimates of hydrobreaking potential based solely on analytical data, may not be realistic.

Underground coal before mining is confined and under overburden pressure directed across the bedding planes. During and after mining such pressure tends to focus on support pillars. The maximum pressure is estimable from the general rule that the static pressure of 1 foot of overburden is approximately 1 psi. As a result of such directed pressure, isolated blocks of pillar coal become more susceptible to coal failure. Incipient friability and cleating are enhanced; and the unconfined compressive strength of a block of coal logically should be minimal where the crushing load is applied across the bedding planes. Logically, then remaining blocks of pillar coal in mine workings in the carbondale area should have higher hydrobreaking potential than coal in virgin mine faces.

Production rates achieved during hydraulic mining of the Roslyn No. 5 coal bed in the Roslyn No. 9 mine near Roslyn, Washington, illustrate the difference in ease of mining pillar coal vs. virgin coal. The rate achieved in mining pillar coal was 16.6 tons/man shift, whereas the rate achieved in mining virgin coal varied from 5.7 tons/man shift in "hard" coal to 9.0 tons/man sift in softer coal (see table 2; field trial No. 3).

In that mine, the ASTM rank of Roslyn No. 5 coal is hvAb; the coal is weakly coking; variations in hardness were noted during hydraulic mining; and friability, as tested, was 42 percent (see table 2; field trial No. 3). Samples of coal from the No. 9 mine contained 38.5-39.9 percent volatiles and one of the coal samples had FSI of 3 (see Aresco, 1956b, RI 5270) suggesting an HGI of about 55, as estimated from the graphs shown on figure 15, this report. These properties do not favor hydrobreaking. Yet, in pillar mining, hydraulic methods were more productive than conventional methods (see table 2, field trial No. 3).

Coal bed A in Thompson Creek mine No. 2 in the north wall of North Thompson Creek (Carbondale coal mining area) was mined hydraulically by driving a pair of raises 350 feet up the pitch of the coal bed (see table 2 field trial No. 4). At that locality the rank of the coal is borderline between high-volatile A and medium volatile. The average production rate achieved (23.3 tons/man shift) was considerably greater than were those achieved during the other field trials. On that basis, it would seem that coal bed A has a high hydrobreaking potential. The analytical data available also seem reassuring: a sample of coal from Thompson Creek mine No. 2 contained 29.5 percent volatiles; the sample had an FSI of 8 and an HGI of 86 (see Appendix C; table C-1). These properties favor hydrobreaking.

It will be noted, however, that productivity rates achieved in hydraulic mining of coal bed A ranged from 3.5 tons/man shift to 46 tons/man shift. This extreme range was attributed to variations in coal "hardness" (see table 2, field trial No. 4). Such variations seem at odds with the statistical finding that strongly coking HvAb-mvb coals containing 21-36 percent volatiles generally exhibit minimum hardness of vitrain layers. This suggests that some other factors that promote variations in coal hardness were locally effective in Thompson Creek mine No. 2.

Keenan and Carpenter (1960) described and illustrated a composite system of faults encountered during mining in the Thompson Creen No. 2 mine. Their illustrations (1960, fig. 4 and 5) indicate that fault spacing is generally less than 100 feet. A change in the quality of coal was noted in the fault areas, and --"It has been the experience in these operations that the coal in the immediate vicinity of a fault has weaker coking characteristics" (Keenan and Carpenter, 1960, p. 235). Although no mention was made of faults having been encountered over the 350-foot distance of hydraulic raise-mining, it is logical to assume that some of the coal that was mined was in the immediate vicinity of faults. As noted in a previous section of this report (see "coal hardness"), vitrain layers of weakly coaking hvA coal may be relatively hard, whereas vitrain layers of strongly coking hvA coal may be relatively soft.

At the locality where coal bed A was mined hydraulically, the bed contains three non-coal partings of variable thickness (see table 2, field trial No. 4). Partings in coal bed A are localized in the immediate vicinity of North Thompson Creek; they are not present at a locality 1 mile to the south (see fig. 22). Where present, however, the partings would have made coal bed A harder to mine hydraulically. Owing to this circumstance, the full hydrobreaking potential of coal bed A may not have been realized during the USBM field trial.

During a brief field survey of the Thompson Creek mining area, some faces of coal beds A and B were studied in the Thompson Creek mine No. 1. Although mainly in the south wall, the No. 1 mine-complex extends to the north beneath North Thompson Creek, and is virtually contiguous with Thompson Creek mine No. 2.

The faces of coal bed A were examined with the expectation that the coal would be easily fractured, and would fall off each face when hit with a hammer. The faces of coal bed A resisted hammer attack, contrary to the theory of minimum hardness. On closer examination, it was observed that cleat development in coal bed A was notably poor, and the vitrain layers seemed relatively hard. In marked contrast, coal bed B proved to be highly fractured. Cleating in coal bed B was well developed and vertical layers of coal literally cascaded off all parts of the faces, when struck with a hammer.

Coal section No. 2, on figure 22, shows representative profiles of coal beds A and B in the immediate vicinity of North Thompson Creek. Analytical data on the percent-content of volatiles, ash, and carbon contained in the various component parts of each of those coal beds, are also shown on figure 22.

The analytical data on the coal-parts of coal beds A and B shown in coal section No. 2 do not seem to indicate any appreciable differences in composition (see fig. 22). As indicated by the profiles, however, coal bed A contains four partings, whereas coal bed B is virtually a single bench of coal. Evidently the partings in coal bed A interrupted cleat spacing and inhibited cleat development in the vitrain layers, whereas in coal bed B development of primary and secondary cleating proceeded without interruption, owing to the absence of partings. These differences would account for the marked difference in breakage observed.

Lateral variations in the number, thickness, and spacing of rock partings in correlative Carbondale coals are shown on figures 11, 12, and 22. The rock partings consist of shale, clay, slate, "draw slate," bone, or bony impure coal, which are interbedded with bright coal (i.e. vitrain). The partings would interrupt cleat spacing and cleat-development in the vitrain layers. The number, thickness, and spacing of partings in the coal beds would determine the degree the fracture-cleat pattern in bright coal is interrupted.

The mechanics of hydrobreaking strongly indicate that the number, thickness, and spacing of rock partings in a specified block of bituminous coal are primary factors of influence on the hydrobreaking potential of the block. Hydrobreaking occurs mainly along existing cleat and fracture planes when a high-water-pressure jet monitor is directed against a face of coal; and Cooley (1975, v. 1, p. 5-5) noted that :

> "Breakage of coal from a free face is generally more effective if the jet is directed approximately parallel to the plane of cleats and laminations. If the jet is directed at 90^0 to the cleat plane, the breakage rate is decreased by 55 to 65 percent [Ref. 668, p. 12]."

Therefore, it is logical to conclude that (a) the degree of primary cleat-development in a coal bed strongly influences the overall effectiveness of jet cutting-hydrobreakage, and (b) rock partings reduce hydrobreaking potential by inhibiting cleat development in the interbedded layers of coal. Primary cleats (i.e., those that cut vertically through all parts of a specified coal bed) are most apt to be well-developed in beds consisting of a single bench of bright coal.

It should not be too difficult to determine the number, thickness, and spacing of rock partings in a specified block of coal in advance of (hydraulic) mining. Specified blocks of bituminous coal in which partings are absent, or those in which only a few thin partings occur, should be most susceptible to hydrobreaking.

For a specified block of underground coal, the overall susceptability to breakage by monitor jets is a function of the degree the block is permeable to water. However, the block is composed of hetergeneous substances, and permeability is affected by a variety of factors which include porosity, friability, and cleat development; composition of rock partings and associated joint development; state of stress; and direction of water permeation. The permeability of a block of pillar coal might be evaluated in a general way in advance of mining, but that of virgin coal would be much more difficult to evaluate.

In Russia, water infusion tests of specified blocks of coal are made in advance of hydraulic mining, in order to determine degree of permeability. Results of such testing are also used to measure the ability of coals to be broken by a monitor jet (i.e. hydrobreaking potential), and to estimate the minimum jet pressure required to cut coal effectively. Advantages gained by water infusion testing, and the test method used, are described by Cooley (1975, v. 1, p. 5-5 and 6-6) in the following quotation:

> "As a first order approximation, it was initially found (Ref. 661) that the minimum jet pressure required to cut coal effectively should exceed a value of about one half the compressive strength or 50 times the Protod'yakonov hardness number F (which is approximately 1 percent of the unconfined compressive strength). However, exceptions to this rule were found and therefore a water infusion test method was developed in the USSR (REF. 450, later superseded by Ref. 66) to measure the ability of coals to be broken by a monitor jet. The method requires drilling a hole in the coal seam, installing a hydroseal (or packer), infusing water under pressure, and measuring the pressure required to fracture the seam and the pressure variation with time. The specific water absorption of the seam is determined and correlated with the monitor jet pressure required for effective breakage. Since water is infused in all directions into the seam, an average value of specific water absorption is obtained which allows for anisotropic permeability."

In addition, water infusion of blocks of (bituminous) coal in advance of hydraulic mining pre-loosens the coal, thereby making subsequent jet cutting-hydrobreakage more effective.

The U.S. Bureau of Mines conducted water infusion tests in the Bottom Red Ash bed of anthracite (see Malenka, 1968, RI-7120). The tests were conducted during the course of a field trial of hydraulic mining (see table 2, field trial No. 2). On figure 21, coal section No. 2 is a representative profile of the anthracite that was mined hydraulically.

The purpose of the test was to determine if water infusion would result in subsequent increases in production rates of hydraulic mining, as compared to the average rate of mining prior to infusion (see Malenka, 1968, p. 16-18). However, water infusion tests were terminated because no appreciable increase in production rates resulted, and water infusion apparently introduced hazards to safety. Malenka (1968, p. 18) summarized the results of infusion, as follows:

> "The mining of coal beds 13 to 15 feet thick has been hazardous owing to falls of top coal along ribs and face. Water infusion of the coal face apparently aggravated this hazardous condition in that falls of coal occurred with increasing frequency. This condition was most acute immediately following the undercutting of the coal face. The depth of the undercut was either 3 or 6 feet with a width of 7 or 11 feet depending on the number of mast setups employed. On one occasion a section of the coal face measuring 10 by 10 by 3 1/2 feet thick rolled out of the face and lodged against the hydraulic jumbo causing some delay. Considering this apparent hazard to safety and results obtained in mining, the water infusion tests were terminated."

As shown by the profile of the Bottom Red Ash bed of anthracite that was mined hydraulically, the upper part of the coal bed contains numerous refuse bands (i.e. rock partings) interbedded with coal, whereas the lower part consists of a bench of coal about 6 feet thick (see fig. 21; coal section No. 2). The refuse bands consisted of "sand rocks" and "sand slates" in which fracture lines were discernible (Buch, 1965, p. 5). In commenting on experience obtained during hydraulic mining of Bottom Red Ash coal, Buch (1965), p. 5) stated that--. "The bottom bench has cleavage planes in it, but in general this bench is denser and harder to mine than any of the overlying benches, including the refuse bands."

Characteristically, anthracite is a dense, hard, metamorphosed substance. Beds of anthracite tend to be structurally massive, and to have low porosity and permeability. It would seem that, in the category of metamorphic rocks a thick bench of anthracite has the <u>least</u> water infusion and hydrobreaking potential.

Actually, anthracite beds are comparable to bituminous coal beds only on the basis of thickness, and lithologic make-up (i.e., distribution of coal and rock in the bed). Relative differences in degree of actual hydrobreaking achieved for anthracite vs. rock partings are opposite from those for bituminous coal containing rock partings. For beds of similar thickness and (or) lithologic make-up, results obtained from water infusion testing of anthracite beds vs. bituminous coal beds should also be opposite, because (a) the effectiveness of water infusion depends on the degree of permeability of the block of coal tested, and (b) water infusion is a form of hydrobreaking.

Results and experiences obtained from water infusion testing of anthracite beds are not transferable to infusion testing of bituminous coal beds--except perhaps in an opposite way. Nonetheless, the technology developed for water infusion testing of blocks of anthracite prior to hydraulic mining, should be directly transferable to testing blocks of bituminous coal.

HYDRAULIC MINING POTENTIAL, CARBONDALE COALS

Successful hydraulic mining of the pitching coal deposits in the Carbondale area depends on nearness of water supplies; the hydrobreaking potential of the coals; and costs of mining, preparation, and transportation. The hydraulic mining potential of individual coal beds varies considerably from coal to coal and from place to place throughout the area. Under favorable conditions, hydraulic mining might be incorporated with conventional methods of mining.

Coal beds in the Carbondale area pitch from 10 to 50 degrees. Pitching coal deposits are favorable sites for hydraulic mining and initially appear to have high hydraulic mining potential, but this potential is relative: as the pitch of a coal bed increases, conventional mechanized mining becomes increasingly difficult; ultimately, either hydraulic mining or hand mining becomes the only viable mining method for recovering pitching coal.

Hydraulic mining of pitching coals at elevations above adjacent ground or creek levels, and above the ground-water table, is of course much easier than mining below such levels. However, as related to the economic potential of mining, such differences in elevation tend to place areal limits on parts of a coal deposit most easily mined hydraulically.

In all parts of the Carbondale area, availability of adequate water supplies for hydraulic mining is a matter of primary concern. Surface water supplies seem marginal to requirements for hydraulic mining, because surface water flow fluctuates drastically throughout the year, and ownership of surface water rights is uncertain. Seasonally, the minimum flow in Fourmile and North Thompson Creeks, for example, is less than 300 gallons per minute. Monitor jets that might be used require from 40 to 300 gallons per minute (see table 2).

Fortunately, considerable volumes of mine water have accumulated where mine workings are extensive. Such accumulations are distributed throughout the Carbondale area. Mine water supplies should be adequate for hydraulic mining, if stored and recycled. However, usage of such supplies would limit hydraulic mining to areas within or adjacent to existing mine workings.

Statistically, strongly coking medium and low-volatile bituminous coals are much softer, much easier to grind, and much more friable than are non-coking high-volatile C and B coals; and high volatile A coals which exhibit a wide range of coking strength also exhibit a wide range of hardness, grindability, and friability. On that basis, the hydrobreaking potential of coals in the Carbondale mining area is (a) high in the southern part of the area, (b) low in the northern part, and (c) variable, in the central part. However, the mechanics of hydrobreaking, and the variable production rates achieved during hydraulic mining tests, strongly indicate that the number, thickness, and spacing of rock partings in a specified block of bituminous coal are the primary factors that control the hydrobreaking potential of that block; hydrobreaking potential varies inversely, with those factors.

At places within the Carbondale mining area, it may be difficult to sustain large-scale hydraulic mining operations or to achieve high-volume production by that method alone because of locally inadequate water supplies. However, overall mining plans might incorporate hydraulic mining with longwall and (or) room-and-pillar mining operations, to insure more efficient long-term recovery of remaining resources of pitching coal and to provide a supplimental, additional, or alternative method of recovering coal.

Advantages of incorporating hydraulic mining include:

1) Small-scale hydraulic mining operations could be conducted in favorable parts of a mine or mine complex without interference to longwalling or room-and-pillar operations in other parts.

2) Some pillar coal could be recovered.

3) As compared to longwalling, hydraulic mining is much more flexible and adaptable to changes in mining conditions; consequently, some coal not otherwise recoverable may be recovered by hydraulic mining methods.

4) Coal beds B and Anderson, for example, have high hydrobreaking potential, but they are characterized by lateral variations in thickness, and the Anderson is locally discontinuous; hydraulic mining methods would be most adaptable to those conditions.

APPENDIX

Appendix A

Weather Records, Glenwood Springs weather station, El. 5,823

Type of record	Jan.	Feb.	Mar.	Apr.	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Annual
A. Temperature (°F)													
1. Normals for the period 1931-1960	24.8	29.7	37.9	47.6	56.3	64.3	71.1	69.0	61.6	51.0	36.4	27.4	48.1
2. Mean, record prior to 1931(26 years)--	23.0	28.1	37.0	45.5	53.6	61.4	66.8	65.7	58.3	47.8	36.0	24.3	45.6
3. Means, 1931-1951 (21 years)---------	24.0	29.7	38.1	48.1	56.6	64.0	71.3	69.3	61.8	51.1	36.6	27.5	48.2
B. Precipitation (in)													
1. Normals for the period 1931-1960---	1.80	1.75	1.53	1.90	1.36	1.19	1.36	1.67	1.41	1.40	1.21	1.45	18.03
2. Means, 1931-1952(22)-	1.91	1.94	1.72	2.04	1.45	1.30	1.49	1.74	1.46	1.39	1.23	1.61	19.28
3. Means, record prior to 1931 (28-33 years)	1.29	1.00	1.45	1.27	1.19	.81	1.31	1.65	1.34	1.17	.99	1.16	14.63
C. Mean Snowfall:													
1. Record prior to 1931 (26)	16.6	11.8	8.1	2.0	.4	.0	.0	.0	.4	1.7	5.6	15.8	62.4
2. 1931-1950	22.9	15.5	8.5	1.9	.3	T	.0	.0	.0	1.1	6.4	17.6	74.2

Weather Records, Crested Butte, El. 9 Station pre 1931-1945: 8,950
El, 9 Station 1946- 8,867

Type of Record	Jan.	Feb.	Mar.	Apr.	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Annual
A. Temperature (°F)													
1. Normals 1931-1960	13.5	16.3	22.6	33.4	44.2	52.3	58.0	56.9	50.1	40.3	25.7	16.8	35.8
2. Means, prior to 1931 (19)	12.2	15.2	22.8	32.5	43.4	52.3	57.0	54.8	46.8	35.4	24.0	18.2	34.6
3. Means, 1931-1950	12.7	16.5	22.2	33.6	44.2	51.9	57.8	57.0	49.9	40.2	25.5	16.7	35.7
B. Precipitation (in)													
1. Normals 1931-1960	2.68	2.56	2.36	1.73	1.31	1.43	1.95	2.27	1.66	1.43	1.52	2.10	23.00
2. Means, prior 1931 (20)	2.68	2.18	2.46	1.74	1.58	1.37	2.19	1.98	2.42	1.23	1.40	2.18	23.41
3. Means, 1931-1952	2.41	2.36	2.18	1.68	1.22	1.56	1.95	2.38	1.76	1.30	1.38	2.22	22.40
C. Snowfall													
1. Means, prior 1931 (19)	36.2	28.0	28.4	16.5	6.8	1.1	.0	.0	1.1	6.8	15.7	26.2	166.8
2. Means, 1931-1951	32.9	32.0	31.1	15.7	4.7	.5	T	T	1.0	6.3	18.6	30.6	173.4

9-0846. Fourmile Creek near Glenwood Springs, Colo.

Location.--Lat 39°24'00", long 107°19'25", in NW¼SW¼ sec.34, T.7 S., R.89 W., on left bank 100 ft downstream from National Forest boundary, 2.8 miles upstream from Freeman Creek, 6 miles west of Carbondale, 8 miles upstream from mouth, and 10 miles south of courthouse in Glenwood Springs.

Drainage area.--16.7 sq mi.

Records available.--October 1957 to September 1965 (discontinued).

Gage.--Water-stage recorder. Altitude of gage is 7,780 ft (from topographic map).

Average discharge.--8 years, 8.29 cfs (6,000 acre-ft per year).

Extremes.--Maximums and minimums (discharge in cubic feet per second, gage height in feet). 1/

Annual maximum discharge (*) and peak discharges above base (60 cfs), water years 1961-65

Date	Time	Discharge	Gage height	Date	Time	Discharge	Gage height	Date	Time	Discharge	Gage height
May 2, 1961	1800	82	4.35	May 11, 1962	1900	* 303	5.21	Apr. 22, 1965	1700	68	4.82
May 11, 1961	1730	* 131	4.61					May 4, 1965	1800	160	4.49
May 18, 1961	2000	79	4.18	Apr. 26, 1963	2100	* 35	3.93	May 17, 1965	2030	* 234	4.67
								June 12, 1965	0430	128	4.33
Apr. 28, 1962	2230	175	4.76	May 16, 1964	2000	* 267	5.13				

Annual minimum daily discharge, water years 1961-65

Water year	Date	Discharge	Water year	Date	Discharge
1961	Aug. 14, 15, 20, 22, 23, 1961	0.20	1964	Several days	0.20
1962	Sept. 3-19, 1962	.30	1965	Many days	.60
1963	Aug. 1-20, 1963	.10			

1957-65: Maximum discharge, 303 cfs May 11, 1962 (gage height, 5.21 ft), from rating curve extended above 100 cfs; minimum daily, 0.10 cfs Sept. 5, 7-11, 1959, Aug. 1-20, 1963.

Remarks.--Records good except those for winter periods, which are fair. No diversion above station.

DISCHARGE, IN CUBIC FEET PER SECOND, WATER YEAR OCTOBER 1960 TO SEPTEMBER 1961

DAY	OCT.	NOV.	DEC.	JAN.	FEB.	MAR.	APR.	MAY	JUNE	JULY	AUG.	SEPT.
1	.30	.60	.40	.40	.60	.40	.80	37	27	1.9	.60	.40
2	.30	.60	.40	.40	.60	.40	.80	50	25	1.5	2.0	.40
3	.30	.60	.50	.40	.60	.40	1.4	51	21	1.6	.90	.80
4	.30	.60	.50	.40	.60	.40	2.4	45	20	1.4	.60	1.1
5	.30	.60	.50	.50	.60	.40	2.7	29	18	1.2	.50	1.2
6	.30	.50	.50	.50	.60	.40	2.8	17	17	1.1	.40	1.5
7	.30	.50	.40	.40	.60	.40	2.4	13	15	1.1	.30	1.0
8	.30	.50	.40	.50	.60	.40	2.8	14	14	1.0	.30	.80
9	.30	.50	.50	.50	.50	.40	3.4	26	13	.80	.30	1.0
10	.40	.50	.50	.50	.50	.40	1.9	59	11	.80	.30	1.8
11	.40	.60	.40	.50	.50	.50	1.6	86	9.7	.80	.30	.70
12	.40	.60	.40	.60	.40	.50	1.7	81	8.7	.70	.30	.50
13	.30	.60	.40	.60	.40	.50	1.6	65	8.0	.60	.30	.40
14	.50	.60	.40	.60	.40	.60	1.2	46	7.0	.60	.20	.40
15	.60	.60	.40	.50	.40	.70	1.4	43	6.8	.60	.20	.40
16	.60	.50	.30	.50	.40	.70	1.4	47	5.9	.50	.30	.50
17	.60	.50	.40	.50	.40	.70	1.8	53	5.7	.50	.30	.80
18	.60	.50	.40	.50	.40	.70	3.4	62	5.2	.50	.30	.80
19	.60	.50	.40	.60	.40	.70	5.9	60	5.8	.50	.30	2.0
20	.60	.50	.40	.60	.40	.70	10	64	4.1	.50	.20	1.2
21	.50	.50	.40	.50	.40	.60	9.4	59	3.7	.60	.30	1.6
22	.50	.50	.40	.40	.50	.70	8.7	58	3.4	.60	.20	3.7
23	.50	.40	.40	.50	.50	.80	15	54	3.2	.50	.20	3.9
24	.50	.40	.40	.60	.50	1.0	9.0	47	2.8	.50	.30	4.6
25	.50	.40	.40	.60	.50	1.2	6.3	42	2.7	.50	.60	2.9
26	.50	.40	.40	.60	.40	1.0	4.3	41	2.3	.50	.40	3.4
27	.50	.50	.40	.60	.40	.80	4.1	38	2.2	.40	.40	2.8
28	.50	.40	.40	.60	.40	.70	4.3	37	2.1	.40	.30	2.6
29	.60	.30	.40	.60	------	.70	17	34	2.1	.40	.30	4.3
30	.50	.50	.40	.60	------	.70	30	35	2.0	.50	.30	4.1
31	.50	------	.40	.60	------	.80	------	31	------	.50	.40	------
TOTAL	13.90	15.30	12.90	16.20	13.50	19.30	161.50	1,432	273.6	23.60	12.60	50.80
MEAN	.45	.51	.42	.52	.48	.62	5.38	46.2	9.12	.76	.41	1.69
MAX	.60	.60	.50	.60	.60	1.2	30	86	27	1.9	2.0	4.6
MIN	.30	.30	.30	.40	.40	.40	.80	13	2.0	.40	.20	.40
AC-FT	28	30	26	32	27	38	320	2,840	543	47	25	101

CAL YR 1960: TOTAL 2,606.00 MEAN 7.12 MAX 88 MIN .20 AC-FT 5,170
WAT YR 1961: TOTAL 2,045.20 MEAN 5.60 MAX 86 MIN .20 AC-FT 4,060

1/ Cubic foot per second (cfs) is the rate of discharge representing a volume of 1 cubic foot passing a given point during 1 second and is equivalent to 7.48 gallons per second or 448.8 gallons per minute.

9-0846. Fourmile Creek near Glenwood Springs, Colo.--Continued

DISCHARGE, IN CUBIC FEET PER SECOND, WATER YEAR OCTOBER 1961 TO SEPTEMBER 1962

DAY	OCT.	NOV.	DEC.	JAN.	FEB.	MAR.	APR.	MAY	JUNE	JULY	AUG.	SEPT.
1	2.8	1.4	1.2	.90	1.0	1.0	2.6	64	46	9.1	1.6	.40
2	2.4	1.4	1.2	.90	1.0	1.2	2.8	61	48	7.8	1.3	.40
3	1.9	1.2	1.2	.90	1.0	1.2	3.0	74	47	7.2	1.1	.38
4	1.7	1.4	1.2	.90	1.0	1.2	3.2	107	47	6.4	1.1	.30
5	1.4	1.4	1.2	.90	1.0	1.0	3.2	145	47	6.1	1.1	.30
6	1.4	1.4	1.2	.90	1.0	1.0	3.5	175	46	5.6	.80	.30
7	1.3	1.6	1.2	.90	1.0	1.2	3.9	195	41	5.6	.80	.30
8	1.5	1.3	1.0	.90	1.0	1.4	3.7	182	37	5.8	.70	.30
9	4.8	1.3	1.0	.90	.90	1.4	3.7	200	34	5.0	.80	.30
10	4.1	1.3	1.0	.80	1.0	1.4	4.1	195	31	4.3	.90	.30
11	2.9	1.2	1.0	.80	1.2	1.3	4.4	195	31	3.9	.80	.30
12	2.7	1.3	1.0	.80	1.4	1.3	9.2	188	31	3.6	.80	.30
13	2.3	1.3	1.0	1.0	1.4	1.2	7.7	150	32	3.7	.90	.30
14	2.3	1.3	1.0	1.0	1.2	1.2	15	112	31	3.4	.80	.30
15	1.9	1.2	1.0	1.0	1.2	1.2	28	90	27	2.6	.60	.30
16	1.8	1.2	1.0	.90	1.3	1.2	55	73	26	2.4	.60	.30
17	1.7	1.2	1.0	.90	1.3	1.2	79	66	23	2.1	.60	.30
18	1.6	1.3	1.0	1.0	1.2	1.2	94	64	20	1.8	.60	.30
19	1.5	1.2	1.0	.90	1.2	1.2	103	58	18	1.8	.60	.30
20	1.4	1.2	1.0	1.0	1.3	1.2	115	61	18	1.8	.70	.50
21	1.4	1.2	1.0	1.2	1.3	1.2	188	57	18	1.6	.70	.60
22	1.5	1.2	1.0	1.2	1.3	1.2	95	54	17	1.3	.60	2.1
23	1.4	1.2	1.0	1.0	1.3	1.2	100	51	16	1.3	.60	.80
24	1.3	1.2	1.0	1.0	1.3	1.2	110	48	15	1.3	.60	.60
25	1.2	1.2	1.0	.90	1.3	1.4	114	48	13	1.3	.60	.50
26	1.2	1.2	1.0	1.0	1.3	1.7	118	46	12	1.5	.50	.40
27	1.8	1.2	1.0	1.0	1.2	2.3	92	49	11	1.6	.40	.40
28	1.6	1.2	1.0	1.0	1.0	2.4	105	46	10	1.6	.40	.50
29	1.6	1.2	1.0	1.0	——	2.4	128	49	9.7	1.6	.40	.60
30	1.6	1.2	1.0	1.0	——	2.4	76	45	10	1.2	.40	.60
31	1.4	——	.90	1.0	——	2.4	——	42	——	1.1	.40	——
TOTAL	58.7	38.1	32.30	29.50	32.60	44.0	1,578.0	2,990	812.7	105.2	22.60	13.50
MEAN	1.89	1.27	1.04	.95	1.16	1.42	52.6	96.5	27.1	3.39	.73	.45
MAX	4.8	1.6	1.2	1.2	1.4	2.4	128	200	48	9.1	1.6	2.1
MIN	1.2	1.2	.90	.80	.90	1.0	2.6	42	9.7	1.1	.40	.30
AC-FT	116	76	64	59	65	87	3,130	5,930	1,610	209	45	27

CAL YR 1961: TOTAL 2,132.20 MEAN 5.84 MAX 86 MIN .20 AC-FT 4,230
WAT YR 1962: TOTAL 5,737.20 MEAN 15.8 MAX 200 MIN .30 AC-FT 11,420

DISCHARGE, IN CUBIC FEET PER SECOND, WATER YEAR OCTOBER 1962 TO SEPTEMBER 1963

DAY	OCT.	NOV.	DEC.	JAN.	FEB.	MAR.	APR.	MAY	JUNE	JULY	AUG.	SEPT.
1	.40	.70	.70	.60	.80	.40	9.7	13	3.6	.20	.10	.20
2	.50	.70	.70	.60	.60	.40	6.4	16	3.4	.20	.10	.20
3	.40	.70	.70	.60	.60	.60	5.0	19	6.4	.20	.10	.20
4	.40	.70	.60	.60	.60	.60	4.6	19	5.8	.20	.10	.20
5	.40	.70	.70	.60	.60	.60	3.6	20	4.5	.20	.10	.20
6	.40	.70	.70	.60	.50	.60	5.0	18	3.7	.20	.10	.30
7	.40	.60	.70	.50	.50	.60	10	18	3.6	.20	.10	.40
8	.40	.60	.80	.50	.50	.60	12	18	3.2	.20	.10	.40
9	.40	.70	.80	.40	.50	.60	11	22	4.1	.30	.10	.30
10	.40	.70	.70	.40	.60	.60	11	20	3.9	1.8	.10	.30
11	.40	.70	.60	.40	.60	.60	9.1	18	2.9	.70	.10	.20
12	.40	.70	.60	.30	.60	.60	13	16	2.5	.60	.10	.20
13	.40	.70	.60	.30	.70	.60	18	13	2.5	.60	.10	.20
14	.40	.80	.60	.30	.80	.60	20	12	2.8	.40	.10	.60
15	.40	.80	.60	.40	.60	.60	15	11	2.9	.40	.10	.60
16	.40	.80	.60	.40	.40	.60	10	10	3.2	.30	.10	.40
17	1.0	.80	.60	.40	.40	.60	9.1	9.7	4.1	.30	.10	.40
18	.90	.80	.60	.40	.30	.60	8.1	8.8	3.8	.20	.10	.40
19	1.7	.80	.60	.40	.30	.60	6.4	8.4	4.0	.20	.10	.50
20	1.7	.80	.60	.40	.30	.60	4.1	7.8	3.2	.20	.10	.50
21	1.2	.80	.60	.40	.40	.70	6.1	7.0	2.4	.20	.20	.50
22	1.1	.80	.60	.40	.60	.90	5.8	6.7	1.6	.20	.20	.50
23	1.0	.80	.60	.40	.30	1.4	5.0	7.0	1.1	.40	.20	.40
24	1.0	.70	.60	.40	.40	1.6	9.7	6.7	.80	.60	.20	.40
25	1.6	.70	.60	.40	.40	1.5	15	6.1	.60	.40	.20	.30
26	.92	.70	.50	.50	.40	1.8	18	5.6	.60	.40	.20	.30
27	.96	.60	.60	.50	.40	2.4	18	4.8	.40	.30	.20	.20
28	[illegible]	[illegible]	.60	.50	.40	3.2	12	4.5	.30	.30	.20	.30
29	1.0	.70	.60	.60	——	4.8	10	4.3	.20	.20	.20	.30
30	1.0	.70	.60	.60	——	5.3	13	4.1	.20	.20	.20	.30
31	.80	——	.60	.60	——	6.6	——	3.7	——	.20	.20	——
TOTAL	22.80	22.00	19.70	14.40	13.90	41.60	305.7	358.2	82.30	11.20	4.20	10.40
MEAN	.74	.73	.64	.46	.50	1.34	10.2	11.6	2.74	.36	.14	.35
MAX	1.7	.80	.80	.60	.80	6.6	20	22	6.6	1.8	.20	.60
MIN	.40	.60	.60	.30	.30	.40	3.6	3.7	.20	.20	.10	.20
AC-FT	[illegible]	44	39	29	28	83	606	710	163	22	8.3	21

CAL YR 1962: TOTAL 5,692.60 MEAN 15.6 MAX 200 MIN .30 AC-FT 11,290
WAT YR 1963: TOTAL 906.40 MEAN 2.48 MAX 22 MIN .10 AC-FT 1,800

9-0846. Fourmile Creek near Glenwood Springs, Colo.--Continued

DISCHARGE, IN CUBIC FEET PER SECOND, WATER YEAR OCTOBER 1963 TO SEPTEMBER 1964

DAY	OCT.	NOV.	DEC.	JAN.	FEB.	MAR.	APR.	MAY	JUNE	JULY	AUG.	SEPT.
1	.30	.70	.50	.40	.30	.40	1.5	14	34	3.0	2.0	.90
2	.30	.70	.50	.40	.30	.40	1.5	12	32	2.8	1.8	.80
3	.20	.70	.40	.40	.30	.40	1.3	7.4	35	2.7	3.5	.70
4	.20	.90	.40	.40	.30	.40	1.2	6.6	33	2.6	3.9	.60
5	.20	1.0	.40	.40	.30	.40	1.1	5.2	30	2.5	3.8	.50
6	.20	.90	.50	.40	.30	.40	1.0	4.4	28	2.4	2.5	.40
7	.30	.90	.40	.40	.30	.40	1.0	4.4	27	2.2	2.0	.40
8	.20	.80	.40	.40	.30	.40	.90	4.4	31	2.0	1.8	.30
9	.20	.70	.40	.40	.30	.40	1.0	4.4	24	1.9	1.6	.30
10	.30	.70	.40	.40	.30	.40	1.1	8.0	20	1.9	1.4	.30
11	.20	.70	.40	.40	.40	.40	1.2	20	19	1.8	1.1	.30
12	.30	.60	.40	.40	.40	.40	1.3	34	18	1.6	1.5	.20
13	.30	.60	.40	.40	.40	.50	1.1	52	16	2.8	3.0	.20
14	.30	.60	.40	.40	.40	.50	1.3	75	14	3.5	2.0	.20
15	.20	.60	.40	.40	.40	.50	1.9	103	12	2.1	1.3	.40
16	.20	.60	.40	.40	.40	.50	2.5	133	11	2.8	1.5	.40
17	.30	.50	.40	.50	.40	.60	2.9	170	11	1.7	2.0	.40
18	.30	.50	.40	.50	.40	.60	2.8	117	12	2.0	1.5	.40
19	.30	.40	.40	.40	.40	.60	4.4	117	9.8	2.8	1.3	.40
20	.40	.40	.40	.40	.40	.50	4.6	112	9.1	2.2	1.9	.50
21	.50	.60	.40	.40	.40	.60	3.7	119	8.8	3.0	1.4	.30
22	.50	.60	.40	.40	.40	.70	4.1	117	11	2.6	1.5	.40
23	.40	.60	.40	.40	.40	.60	5.0	99	7.7	2.4	1.2	.50
24	.40	.60	.30	.40	.40	.50	11	88	6.6	2.0	.90	.60
25	.40	.60	.30	.40	.40	.50	13	81	5.7	1.8	.70	.70
26	.50	.60	.30	.40	.40	.50	9.4	81	4.8	1.6	.80	.80
27	.40	.60	.30	.40	.30	.50	5.7	78	5.0	1.4	2.0	.70
28	.40	.50	.40	.30	.30	.60	5.7	63	4.3	1.2	1.6	.70
29	.40	.50	.40	.30	.30	.90	6.6	53	4.8	1.0	1.2	.70
30	.40	.50	.30	.30	———	1.4	8.8	46	3.7	1.1	1.8	.70
31	.70	———	.30	.30	———	1.4	———	42	———	2.4	1.0	———
TOTAL	10.20	19.20	12.10	12.20	10.30	17.30	108.60	1,870.8	488.3	67.0	53.90	14.70
MEAN	.33	.64	.39	.39	.34	.56	3.62	60.3	16.3	2.16	1.74	.49
MAX	.70	1.0	.50	.50	.40	1.4	13	170	35	3.5	3.9	.90
MIN	.20	.40	.30	.30	.30	.40	.90	4.4	3.7	1.0	.70	.20
AC-FT	20	38	24	24	20	34	215	3,710	969	133	107	29

CAL YR 1963: TOTAL 883.40 MEAN 2.42 MAX 22 MIN .10 AC-FT 1,750
WAT YR 1964: TOTAL 2,404.60 MEAN 7.34 MAX 170 MIN .20 AC-FT 5,320

Note.--No gage-height record Dec. 30 to Feb. 3.

DISCHARGE, IN CUBIC FEET PER SECOND, WATER YEAR OCTOBER 1964 TO SEPTEMBER 1965

DAY	OCT.	NOV.	DEC.	JAN.	FEB.	MAR.	APR.	MAY	JUNE	JULY	AUG.	SEPT.
1	.50	.60	.50	.60	.50	.40	1.5	117	54	9.8	3.0	.80
2	.50	.70	.50	.60	.50	.40	1.8	105	48	9.1	2.3	.60
3	.60	.70	.50	.70	.50	.50	2.0	109	44	7.7	2.2	.70
4	.60	.60	.50	.70	.50	.60	2.0	105	43	7.8	2.0	.80
5	.60	.50	.50	.70	.50	.60	2.0	88	40	6.6	1.8	1.2
6	.50	.50	.50	.60	.50	.50	2.0	66	42	6.1	1.6	2.1
7	.50	.40	.50	.60	.50	.50	2.0	55	46	5.9	1.4	1.2
8	.50	.50	.50	.60	.50	.50	2.2	51	46	5.9	1.3	1.0
9	.50	.50	.50	.60	.50	.50	2.2	37	46	5.5	1.1	1.2
10	.60	.50	.50	.60	.50	.50	2.3	32	42	5.0	1.0	.90
11	.70	.50	.50	.60	.50	.50	2.2	37	54	5.5	.90	.80
12	.60	.60	.50	.60	.50	.50	2.2	49	98	6.1	.90	1.4
13	.60	.60	.40	.60	.50	.50	2.3	74	78	6.1	.90	1.0
14	.60	.60	.50	.50	.50	.50	2.2	75	68	4.6	.90	.60
15	.70	.60	.50	.50	.40	.50	2.6	66	61	4.1	.90	.50
16	.70	.60	.50	.50	.40	.60	3.0	88	55	3.7	1.0	.40
17	.80	.60	.50	.50	.40	.70	3.5	145	49	3.4	1.1	.40
18	.80	.60	.50	.50	.40	.70	4.3	140	44	3.7	1.7	1.9
19	.70	.60	.50	.50	.40	.70	5.5	158	39	4.1	2.0	3.6
20	.70	.60	.50	.50	.40	.70	7.7	128	35	3.2	1.6	2.8
21	.70	.60	.50	.50	.40	.70	14	140	31	2.9	1.2	2.3
22	.70	.60	.50	.50	.40	.70	24	135	27	2.7	1.0	2.3
23	.80	.60	.50	.50	.40	.70	28	110	23	2.8	1.1	3.2
24	.90	.60	.50	.50	.40	.70	30	84	22	3.7	1.2	3.2
25	.70	.60	.50	.50	.40	.70	30	66	20	4.1	.90	2.7
26	.70	.50	.50	.50	.40	.70	35	58	18	3.9	.80	2.2
27	.70	.60	.50	.50	.40	.70	28	48	15	2.8	.70	1.6
28	.70	.60	.50	.50	.40	.70	30	42	14	2.4	.70	1.6
29	.60	.60	.50	.50	———	.90	53	46	12	2.0	.80	3.2
30	.70	.50	.50	.50	———	1.0	103	50	10	1.8	1.0	2.6
31	.70	———	.60	.50	———	1.3	———	57	———	3.2	1.0	———
TOTAL	20.20	17.20	15.50	17.10	12.60	19.70	430.5	2,581	1,224	147.4	40.00	48.60
MEAN	.65	.57	.50	.55	.45	.64	14.4	83.3	40.8	4.75	1.29	1.62
MAX	.90	.70	.60	.70	.50	1.3	103	160	98	9.8	3.0	3.6
MIN	.50	.40	.40	.50	.40	.40	1.5	32	10	1.8	.70	.40
AC-FT	40	34	31	34	25	39	854	5,120	2,430	292	79	96

CAL YR 1964: TOTAL 2,696.00 MEAN 7.37 MAX 170 MIN .20 AC-FT 5,350
WAT YR 1965: TOTAL 4,573.80 MEAN 12.5 MAX 160 MIN .40 AC-FT 9,070

9-0828. North Thompson Creek near Carbondale, Colo.

Location.--Lat 39°19'47", long 107°19'58", in sec.28, T.8 S., R.89 W., on right bank 0.4 mile downstream from Yank Creek and 8½ miles southwest of Carbondale.

Drainage area.--26.8 sq mi.

Records available.--October 1963 to September 1965.

Gage.--Water-stage recorder. Altitude of gage is 8,120 ft (from topographic map).

Extremes.--Maximum and minimum discharges for the water years 1964-65 are contained in the following table:

Annual maximum discharge (*), peak discharges above base (160 cfs), and annual minimum discharge

Water year	Maximum				Minimum daily		
	Date	Time	Discharge (cfs)	Gage height (feet)	Date	Discharge (cfs)	Gage height (feet)
1964	May 17, 1964	1830	* 322	4.05	Sept. 9, 1964	0.30	-
1965	May 21, 1965	1830	292	3.87	-	-	-
	June 12, 1965	-	* 312	3.97			

1963-65: Maximum discharge, 322 cfs May 17, 1964 (gage height, 4.05 ft), from rating curve extended above 190 cfs; minimum daily, 0.30 cfs Sept. 9, 1964.

Remarks.--Records good except those for winter periods, which are poor. Transbasin diversions above station by Thompson Creek feeder ditch for irrigation in West Divide Creek basin.

DISCHARGE, IN CUBIC FEET PER SECOND, WATER YEAR OCTOBER 1963 TO SEPTEMBER 1964

DAY	OCT.	NOV.	DEC.	JAN.	FEB.	MAR.	APR.	MAY	JUNE	JULY	AUG.	SEPT.
1	1.[illegible]	2.2					2.2	22	100	13	.60	.80
2	.80	2.0					2.2	22	100	12	.70	.80
3	.70	2.2					2.0	19	116	10	6.4	.60
4	.60	2.6					2.0	17	107	9.0	2.2	.60
5	.80	3.0					2.0	12	95	8.0	2.5	.70
6	.60	2.6					2.0	12	92	7.2	1.0	.60
7	.80	2.4					2.0	14	95	6.8	.70	.50
8	.60	2.2				1.0	2.0	13	109	9.0	.60	.40
9	.60	2.0					2.2	14	83	8.0	.60	.30
10	.60	2.0					2.4	27	73	8.5	.70	.40
11	.60	2.0					2.4	33	68	6.8	.70	.40
12	.80	1.8					2.2	44	63	6.4	1.0	.40
13	.80	1.8					2.6	61	53	6.0	7.2	.40
14	.80	1.9					3.0	76	42	7.6	2.2	.50
15	.60	1.8	1.0	.80	.80		6.0	113	39	6.0	1.6	.60
16	.60	1.6					10	151	39	6.0	1.9	.80
17	.70	1.2					11	199	39	5.2	2.8	.40
18	.70	1.4					10	197	39	4.0	1.3	.40
19	.70	1.4					11	193	31	3.1	3.1	1.0
20	5.6	1.4					10	193	26	2.5	5.6	1.6
21	4.4	1.4					12	207	25	2.8	2.2	1.4
22	2.8	1.4					14	221	29	2.2	.90	1.2
23	1.9	1.2				1.6	22	219	23	2.2	.70	1.2
24	3.4	1.4					30	215	22	1.9	1.3	1.2
25	3.1	1.4					25	211	19	.80	3.1	1.2
26	1.9	1.4					22	203	17	.70	3.4	1.6
27	1.6	1.4					16	201	16	.50	6.4	1.8
28	1.[illegible]	1.4					17	175	17	.50	3.1	1.6
29	1.[illegible]	1.2					17	147	16	.50	1.3	1.6
30	1.3	1.[illegible]			------		17	125	15	.60	1.0	1.6
31	3.1	------			------		------	157	------	1.0	.90	------
TOTAL	43.5	52.7	31.0	24.80	23.20	40.6	281.2	3,456	1,608	158.80	67.70	26.40
MEAN	1.40	1.76	1.00	.80	.80	1.31	9.37	111	53.6	5.12	2.18	.88
MAX	5.6	3.0	-	-	-	-	30	221	116	13	7.2	1.8
MIN	.60	1.[illegible]	-	-	-	-	2.0	12	15	.50	.60	.30
AC-FT	86	105	61	49	46	81	558	6,850	3,190	315	134	52
(†)	0	0	0	0	0	0	0	0	455	90	0	0

CAL YR 1963: TOTAL MEAN MAX MIN AC-FT †
WAT YR 1964: TOTAL 5,813.90 MEAN 15.9 MAX 221 MIN .30 AC-FT 11,530 † 545

† Diversions, in acre-feet, by Thompson Creek feeder ditch; furnished by State engineer of Colorado.
Note.--No gage-height record Nov. 17 to Apr. 22.

9-0828. North Thompson Creek near Carbondale, Colo.--Continued

DISCHARGE, IN CUBIC FEET PER SECOND, WATER YEAR OCTOBER 1964 TO SEPTEMBER 1965

DAY	OCT.	NOV.	DEC.	JAN.	FEB.	MAR.	APR.	MAY	JUNE	JULY	AUG.	SEPT.
1	1.5	1.6					10	63	107	42	12	.80
2	1.2	1.9					11	90	105	41	10	.80
3	1.1	1.9					10	93	109	38	11	1.3
4	1.0	1.8					9.0	88	109	35	8.5	1.3
5	.90	1.8					8.0	88	109	34	7.2	2.2
6	.90	1.8					5.5	64	120	31	6.8	7.2
7	1.0	1.8					8.0	52	130	29	5.6	3.1
8	1.0	1.8				.80	5.5	50	140	30	5.2	5.2
9	1.0	1.8					4.0	41	180	29	4.6	6.8
10	1.0	1.8					5.0	37	160	26	4.0	3.4
11	1.3	1.8					4.0	36	200	29	4.4	3.4
12	1.0	1.6					3.7	40	250	50	3.7	6.8
13	1.0	1.6					3.4	60	200	44	3.7	5.6
14	1.0	1.6					4.0	71	180	32	3.1	2.5
15	1.0	1.6	1.4	1.0	.80		5.2	56	170	23	5.2	2.2
16	1.0	1.6					6.0	74	180	22	3.7	1.9
17	1.3	1.6					8.0	131	170	20	3.7	2.2
18	1.0	1.6					10	167	155	20	10	8.0
19	1.0	1.6					15	169	146	20	11	14
20	1.0	1.6					23	183	132	18	6.8	11
21	1.0	1.4					28	219	127	17	4.8	9.0
22	1.3	1.4				2.0	30	231	120	14	3.7	10
23	1.6	1.4					37	205	113	13	3.4	14
24	1.6	1.4					42	149	113	12	2.8	16
25	1.3	1.4					42	116	109	16	2.2	13
26	1.3	1.4					38	104	92	12	1.6	12
27	1.6	1.4					31	86	74	11	1.0	11
28	1.3	1.4					30	78	64	12	1.0	11
29	1.6	1.4			—		35	83	53	10	1.9	14
30	1.6	1.4			—		50	92	43	10	4.0	11
31	1.6	—			—		—	102	—	12	1.6	—
TOTAL	37.00	48.2	43.4	31.0	22.40	44.00	521.3	3,118	3,960	752	158.0	210.70
MEAN	1.19	1.61	1.40	1.00	.80	1.42	17.4	101	132	24.3	5.10	7.02
MAX	1.6	1.9	-	-	-	-	50	231	250	50	12	16
MIN	.90	1.4	-	-	-	-	3.4	36	43	10	1.0	.80
AC-FT	73	96	86	61	44	87	1,030	6,180	7,850	1,490	313	418
(†)	0	0	0	0	0	0	0	0	0	536	0	0

CAL YR 1964: TOTAL 5,815.30 MEAN 15.9 MAX 221 MIN .30 AC-FT 11,530 † 545
WAT YR 1965: TOTAL 8,948.00 MEAN 24.5 MAX 250 MIN - AC-FT 17,740 † 536

† Diversions, in acre-feet, by Thompson Creek feeder ditch; furnished by State engineer of Colorado.
Note.--No gage-height record Nov. 6 to Apr. 5.

09082800 NORTH THOMPSON CREEK NEAR CARBONDALE, COLO.

LOCATION.--Lat 39°19'47", long 107°19'58", in sec.28, T.8 S., R.89 W., Pitkin County, on right bank 0.4 mile downstream from Yank Creek and 8.5 miles southwest of Carbondale.

DRAINAGE AREA.--26.8 sq mi.

PERIOD OF RECORD.--October 1963 to September 1970.

GAGE.--Water-stage recorder. Altitude of gage is 8,120 ft (from topographic map).

AVERAGE DISCHARGE.--7 years, 17.4 cfs (12,610 acre-ft per year).

EXTREMES.--Maximums and minimums (discharge in cubic feet per second, gage height in feet).

Annual maximum discharge (*) and peak discharges above base (160 cfs), water years 1966-70

Date	Time	Disch.	G.H.	Date	Time	Disch.	G.H.	Date	Time	Disch.	G.H.
May 9, 1966	1630	*113	2.97	May 22, 1968	2030	251	3.64	May 2, 1969	2100	163	3.09
				May 31, 1968	2000	*302	3.85	July 21, 1969	1630	*217	3.11
July 31, 1967	1500	*150	3.22								
				Apr. 24, 1969	2100	169	3.16	May 22, 1970	2230	*365	4.00

Annual minimum daily discharge, water years 1966-70

Wtr yr	Date	Discharge	Wtr yr	Date	Discharge
1966	Sept.26-30, 1966	.40	1969	Sept. 3, 1969	.46
1967	Sept. 6, 7, 1967	.09	1970	Aug. 13, 31, 1970	1.0
1968	Nov. 29, 1967	.60			

Period of record: Maximum discharge, 365 cfs May 22, 1970 (gage height, 4.00 ft), from rating curve extended above 170 cfs; maximum gage height, 4.05 ft May 17, 1964; minimum daily discharge, 0.09 cfs Sept. 6, 7, 1967.

REMARKS.--Records fair except those for winter periods, which are poor. Transbasin diversions above station by Thompson Creek feeder ditch for irrigation in West Divide Creek basin.

DISCHARGE, IN CUBIC FEET PER SECOND, WATER YEAR OCTOBER 1965 TO SEPTEMBER 1966

DAY	OCT	NOV	DEC	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP
1	10	3.8	2.0	1.8	1.6	1.6	22	83	47	8.5	2.5	1.3
2	10	3.6	2.0	1.8	1.6	1.5	27	85	39	7.6	2.5	.96
3	8.0	3.4	2.0	1.8	1.6	1.4	24	85	32	6.0	6.4	.96
4	6.8	3.4	2.0	1.8	1.6	1.3	22	86	29	5.6	10	1.8
5	6.4	3.4	2.0	1.8	1.6	1.4	21	88	27	4.0	7.2	.84
6	5.6	3.2	2.0	1.8	1.6	1.6	20	83	24	4.4	5.2	.79
7	5.2	3.2	2.0	1.8	1.6	1.7	20	83	23	3.7	4.4	.76
8	5.2	3.0	2.0	1.8	1.6	1.7	22	88	25	3.4	3.7	.79
9	4.8	3.2	2.0	1.8	1.6	1.6	28	93	22	2.8	3.7	.79
10	4.6	3.0	2.0	1.8	1.6	1.7	31	90	21	2.8	3.7	.68
11	4.0	2.8	2.0	1.8	1.6	1.8	29	78	19	3.4	3.7	.68
12	3.7	2.8	2.0	1.8	1.6	2.2	26	71	18	4.0	5.2	.80
13	3.4	2.5	2.0	1.8	1.6	2.4	23	63	16	4.0	3.7	1.2
14	3.7	2.8	2.0	1.8	1.6	2.8	23	55	12	2.8	3.1	1.8
15	3.7	2.6	2.0	1.8	1.6	3.2	24	52	12	2.2	3.7	1.6
16	3.7	2.4	2.0	1.8	1.6	3.2	29	56	12	1.6	2.5	1.6
17	4.4	2.4	2.0	1.8	1.6	3.0	33	60	12	1.6	3.7	.96
18	7.6	2.6	2.0	1.8	1.6	2.6	34	61	10	1.9	3.1	[illegible]
19	7.2	2.6	2.0	1.8	1.6	2.6	32	60	9.5	2.5	3.4	[illegible]
20	7.2	2.4	2.0	1.8	1.6	2.9	30	56	9.0	2.2	5.6	[illegible]
21	6.8	2.2	2.0	1.8	1.6	2.9	31	58	9.5	1.3	4.8	.88
22	6.4	2.2	2.0	1.8	1.6	2.7	26	60	9.0	2.2	4.4	.70
23	6.0	2.2	2.0	1.8	1.6	2.5	25	58	7.2	2.5	3.7	.70
24	4.8	2.4	2.0	1.8	1.6	2.4	27	52	6.8	2.2	3.4	.68
25	4.2	2.6	2.0	1.8	1.6	3.0	34	50	6.0	2.2	2.5	.58
26	4.0	2.4	2.0	1.8	1.6	6.8	52	49	6.0	.80	2.5	.40
27	4.0	2.2	2.0	1.8	1.6	10	66	47	6.8	.80	1.9	.40
28	3.8	2.0	2.0	1.8	1.6	15	61	47	9.5	1.0	1.8	.40
29	3.8	2.0	2.0	1.8	------	20	63	47	8.5	.60	1.8	.40
30	3.8	2.0	2.0	1.8	------	19	74	47	9.5	.50	1.8	.40
31	3.8	------	2.0	1.8	------	20	------	55	------	.90	2.5	------
TOTAL	166.4	81.3	62.0	55.8	44.8	145.7	979	2,046	497.3	90.00	118.1	23.20
MEAN	5.37	2.71	2.00	1.80	1.60	4.70	32.6	66.0	16.6	2.90	3.81	.77
MAX	10	3.8	2.0	1.8	1.6	20	74	93	47	8.5	10	1.6
MIN	3.4	2.0	2.0	1.8	1.6	1.3	20	47	6.0	.50	1.8	.40
AC-FT	330	161	123	111	89	289	1,940	4,060	986	179	234	46
(†)	0	0	0	0	0	0	0	0	372	0	0	0

CAL YR 1965 TOTAL 9,127.10 MEAN 25.0 MAX 250 MIN .80 AC-FT 18,100 † 536
WTR YR 1966 TOTAL 4,309.60 MEAN 11.8 MAX 93 MIN .40 AC-FT 8,550 † 372

† DIVERSIONS, IN ACRE-FEET, BY THOMPSON CREEK FEEDER DITCH (FURNISHED BY STATE ENGINEER OF COLORADO).
NOTE.--NO GAGE-HEIGHT RECORD NOV. 26 TO MAR. 19.

09082800 NORTH THOMPSON CREEK NEAR CARBONDALE, COLO.--CONTINUED

DISCHARGE, IN CUBIC FEET PER SECOND, WATER YEAR OCTOBER 1966 TO SEPTEMBER 1967

DAY	OCT	NOV	DEC	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP
1	.80	1.0	1.8	1.2	1.8	2.4	9.0	30	33	6.5	5.5	3.4
2	1.0	1.9	1.6	1.2	1.8	2.6	10	26	32	5.5	5.0	.82
3	1.4	1.6	1.6	1.8	1.8	2.6	16	24	32	7.0	4.5	.55
4	1.8	1.4	1.4	1.0	1.8	2.4	20	22	34	7.0	5.0	.19
5	1.8	1.4	2.0	1.2	1.8	2.2	22	20	31	6.5	6.0	.18
6	2.0	1.3	1.6	1.0	1.6	2.2	21	19	30	5.5	6.5	.09
7	1.8	1.0	2.2	1.0	1.6	2.2	22	21	27	6.0	8.0	.09
8	1.8	1.0	2.0	.90	1.4	2.2	24	29	25	9.0	9.0	.64
9	1.8	1.4	1.6	.90	1.6	2.2	24	36	22	6.5	6.5	2.7
10	2.0	1.8	1.2	1.8	1.8	2.0	22	45	19	5.0	6.5	1.3
11	2.2	1.9	1.4	1.0	2.0	2.2	26	50	19	5.5	4.5	1.9
12	3.4	1.4	1.6	1.2	2.0	2.4	26	57	22	6.5	6.0	2.5
13	3.4	1.8	1.8	1.2	2.0	2.4	23	49	19	4.0	4.5	2.5
14	4.0	1.6	1.4	1.2	2.0	2.2	22	36	21	4.8	3.7	1.3
15	3.7	1.8	1.4	1.2	1.8	2.2	19	33	18	5.5	2.5	.73
16	4.0	2.0	1.4	1.2	1.8	2.6	18	37	22	6.5	2.5	.64
17	3.7	2.0	1.4	1.2	1.8	4.5	18	44	17	13	1.9	.64
18	2.2	2.0	1.4	1.0	1.8	7.0	21	52	14	10	1.3	.55
19	1.6	1.8	1.4	1.0	1.8	7.0	30	52	13	5.0	1.6	1.6
20	1.4	1.8	1.6	1.0	1.8	5.0	29	52	17	3.7	1.6	1.6
21	1.6	1.9	1.6	1.2	1.6	4.0	23	51	19	3.1	1.6	1.3
22	1.3	1.9	1.4	1.2	1.6	5.0	22	51	15	2.8	1.8	1.0
23	1.6	1.3	1.4	1.4	1.8	8.0	21	50	13	3.4	1.0	.91
24	1.3	1.4	1.2	1.6	2.0	12	18	48	10	3.7	.91	1.3
25	1.3	1.6	1.4	1.6	2.0	12	18	54	9.7	3.7	.82	1.9
26	1.3	1.4	1.4	1.6	2.0	11	17	69	9.8	2.5	.82	3.1
27	1.0	1.2	1.2	1.4	2.2	10	22	62	8.5	2.2	.82	2.8
28	1.8	1.2	1.0	1.4	2.2	8.0	32	56	8.0	1.9	.82	2.8
29	1.3	1.4	.80	1.4	------	10	43	46	7.0	1.0	.82	1.9
30	1.0	1.6	.80	1.6	------	9.8	39	41	6.5	.91	.91	1.6
31	1.0	------	1.8	2.0	------	9.8	------	35	------	7.1	2.8	------
TOTAL	59.70	47.0	44.20	38.00	51.2	158.5	677.8	1.297	572.7	160.51	106.92	41.95
MEAN	1.93	1.57	1.43	1.23	1.83	5.11	22.6	41.8	19.1	5.18	3.45	1.40
MAX	4.0	2.8	2.2	2.0	2.2	12	43	69	34	13	9.0	3.4
MIN	.80	1.0	.80	.90	1.4	2.0	9.0	19	6.5	.91	.82	.09
AC-FT	118	93	88	75	102	314	1.340	2.570	1.140	318	212	83
†	0	0	0	0	0	0	0	0	480	0	0	0

CAL YR 1966 TOTAL 4.150.80 MEAN 11.4 MAX 93 MIN .40 AC-FT 8.230 † 372
WTR YR 1967 TOTAL 3.254.68 MEAN 8.92 MAX 69 MIN .09 AC-FT 6.460 † 480

† DIVERSIONS, IN ACRE-FEET, BY THOMPSON CREEK FEEDER DITCH (FURNISHED BY STATE ENGINEER OF COLORADO).
NOTE.--NO GAGE-HEIGHT RECORD DEC. 8 TO MAR. 15.

DISCHARGE, IN CUBIC FEET PER SECOND, WATER YEAR OCTOBER 1967 TO SEPTEMBER 1968

DAY	OCT	NOV	DEC	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP
1	1.0	1.2	1.2	1.4	1.4	1.4	15	25	235	19	9.0	2.2
2	.91	1.0	1.0	1.6	1.2	1.0	15	27	227	18	10	2.2
3	1.0	.70	.80	1.6	1.0	1.1	12	30	205	16	18	3.1
4	1.0	1.0	.90	1.5	1.1	1.5	11	41	197	17	9.7	2.8
5	1.4	1.4	1.0	1.6	1.2	1.9	10	44	209	15	8.0	2.5
6	2.8	1.2	1.0	1.7	1.3	1.9	10	46	283	15	10	2.2
7	3.1	1.2	1.1	1.5	1.3	1.9	11	40	175	12	12	1.9
8	1.3	1.0	1.2	1.3	1.3	1.9	12	33	154	12	10	1.9
9	.91	1.1	1.3	1.2	1.2	1.7	15	38	129	12	9.0	1.6
10	.91	1.2	1.5	1.3	1.1	1.6	12	43	114	12	14	1.6
11	.82	1.0	1.8	1.4	1.1	1.8	12	50	104	11	8.0	1.6
12	.82	.80	1.9	1.4	1.1	1.9	13	54	102	9.7	7.5	1.6
13	.82	1.0	1.9	1.4	1.1	1.6	15	54	111	8.5	5.5	1.9
14	.82	1.1	1.8	1.3	1.2	2.1	14	45	111	8.0	9.8	2.5
15	.73	1.3	1.6	1.2	1.5	2.2	13	43	107	9.7	18	3.4
16	.82	1.4	1.7	1.2	1.2	2.1	13	44	105	8.0	6.0	2.2
17	.91	1.4	1.8	1.2	1.1	2.1	13	44	104	7.5	4.5	2.5
18	.91	1.3	1.6	1.2	1.1	2.1	14	56	100	7.5	3.7	2.2
19	.82	1.1	1.6	1.2	1.2	2.1	13	72	96	6.5	3.4	1.9
20	.91	1.2	1.8	1.3	1.3	2.2	12	98	98	7.0	3.1	1.9
21	.82	1.3	1.8	1.4	1.3	2.3	12	148	87	5.5	3.1	2.2
22	.82	1.3	1.4	1.5	1.5	2.5	11	219	63	5.0	3.1	1.8
23	.82	1.3	1.6	1.6	1.6	2.6	11	183	57	6.5	6.8	1.9
24	.80	1.3	1.8	1.6	1.6	2.8	10	135	53	9.7	4.0	1.9
25	.80	1.2	1.8	1.6	1.6	3.1	10	124	45	9.7	3.4	1.9
26	.80	1.1	1.7	1.5	1.6	3.5	10	126	36	9.7	3.1	1.9
27	1.0	1.0	1.7	1.4	1.6	3.4	11	152	28	10	3.1	1.9
28	1.3	.80	1.7	1.3	1.5	3.6	12	187	28	12	3.4	1.6
29	1.3	.60	1.6	1.1	1.5	4.4	14	219	26	9.7	3.1	1.6
30	1.0	.90	1.4	1.3	------	5.5	20	229	22	18	2.8	1.9
31	1.4	------	1.3	1.6	------	9.0	------	231	------	12	2.5	------
TOTAL	33.77	33.40	46.50	43.4	37.6	78.8	372	2.882	3.329	331.2	208.0	62.4
MEAN	1.09	1.11	1.50	1.40	1.30	2.54	12.4	93.0	111	10.7	6.71	2.08
MAX	3.1	1.4	1.9	1.7	1.6	9.0	20	231	235	19	18	3.4
MIN	.73	.60	.80	1.1	1.0	1.8	10	25	22	5.0	2.5	1.6
AC-FT	67	66	92	86	75	156	738	5.720	6.600	657	413	124
(†)	0	0	0	0	0	0	0	0	403	231	0	0

CAL YR 1967 TOTAL 3.217.45 MEAN 8.81 MAX 69 MIN .09 AC-FT 6.380 † 480
WTR YR 1968 TOTAL 7.458.07 MEAN 20.4 MAX 235 MIN .60 AC-FT 14.790 † 634

† DIVERSIONS, IN ACRE-FEET, BY THOMPSON CREEK FEEDER DITCH (FURNISHED BY STATE ENGINEER OF COLORADO).
NOTE.--NO GAGE-HEIGHT RECORD OCT. 31 TO APR. 1.

09082800 NORTH THOMPSON CREEK NEAR CARBONDALE, COLO.--CONTINUED

DISCHARGE, IN CUBIC FEET PER SECOND, WATER YEAR OCTOBER 1968 TO SEPTEMBER 1969

DAY	OCT	NOV	DEC	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP
1	1.9	1.9	2.0	1.8	2.0	2.6	7.0	103	39	10	4.5	.91
2	1.9	1.9	2.0	1.0	2.0	2.2	7.5	124	33	16	3.7	.87
3	1.9	1.6	1.8	1.8	2.0	2.1	8.0	124	28	15	3.4	.73
4	1.9	2.5	2.2	1.9	2.1	2.0	9.0	111	23	16	3.1	1.0
5	2.8	1.9	2.1	2.1	2.2	1.0	10	100	22	13	2.8	1.1
6	2.5	1.3	2.0	2.0	2.3	1.6	12	93	25	11	2.2	.91
7	2.5	2.0	1.8	1.8	2.2	1.6	14	109	26	11	1.9	.82
8	3.1	2.0	1.8	1.6	2.1	1.6	12	93	20	9.7	1.6	.91
9	3.4	2.0	1.8	1.4	2.0	1.6	16	89	16	9.0	1.6	3.6
10	3.6	1.9	2.0	1.5	1.9	1.6	20	91	15	8.5	2.2	3.7
11	3.4	1.8	2.1	1.6	1.9	1.6	28	86	18	7.5	4.0	2.7
12	2.8	2.0	1.8	1.7	1.8	1.6	37	80	18	7.0	3.1	1.6
13	2.8	2.1	1.6	1.8	1.9	1.6	38	89	18	6.5	5.0	1.1
14	2.5	2.1	1.8	1.9	2.0	1.6	46	93	20	7.0	2.5	1.6
15	2.5	2.0	2.0	2.0	2.1	1.7	43	79	24	7.0	1.6	2.2
16	3.1	1.8	2.0	1.8	2.0	2.0	36	71	18	6.0	1.3	3.4
17	2.2	1.8	2.0	1.8	1.9	2.1	31	66	17	7.0	1.3	3.4
18	1.9	1.8	1.8	1.6	1.8	2.1	36	66	19	10	2.2	2.2
19	2.8	2.0	1.6	1.8	1.9	2.1	44	66	15	15	3.4	1.9
20	2.8	2.0	1.0	2.0	2.1	2.8	72	60	12	18	3.7	1.6
21	2.8	2.8	2.0	2.0	2.2	2.1	93	76	11	19	3.4	2.8
22	2.5	2.0	1.8	2.0	2.3	2.3	109	68	9.7	9.0	2.2	6.5
23	2.5	2.0	1.5	1.9	2.4	2.2	124	63	9.0	10	1.3	4.5
24	1.9	2.0	1.7	1.6	2.4	2.1	129	51	46	12	1.3	2.8
25	2.2	2.0	2.0	1.9	2.3	2.0	109	49	39	10	1.0	2.5
26	2.2	1.6	2.0	2.1	2.2	2.1	68	46	39	8.0	.91	2.2
27	1.6	1.7	2.0	2.1	2.2	2.3	53	49	34	9.7	.91	1.9
28	1.3	1.9	1.6	2.0	2.3	3.0	52	49	29	9.7	1.3	1.6
29	1.6	1.6	2.0	1.8	———	4.0	63	45	23	9.0	5.0	1.6
30	1.9	1.7	2.0	1.6	———	5.0	80	44	21	7.5	4.5	1.9
31	2.2	———	2.0	1.8	———	6.0	———	44	———	6.0	1.9	———
TOTAL	74.8	56.9	58.6	56.5	58.5	70.0	1,404.5	2,383	686.7	326.1	78.82	64.70
MEAN	2.41	1.90	1.89	1.82	2.09	2.26	46.8	76.9	22.9	10.5	2.54	2.16
MAX	3.6	2.5	2.2	2.1	2.4	6.0	129	124	46	19	5.0	6.5
MIN	1.3	1.3	1.5	1.4	1.8	1.6	7.0	44	9.0	6.0	.91	.73
AC-FT	148	113	116	112	116	139	2,790	4,730	1,360	647	156	127
(†)	0	0	0	0	0	0	0	188	569	79	0	3

CAL YR 1968 TOTAL 7,534.70 MEAN 20.6 MAX 233 MIN 1.0 AC-FT 14,950 † 634
WTR YR 1969 TOTAL 5,318.62 MEAN 14.6 MAX 129 MIN .73 AC-FT 10,550 † 836

† DIVERSIONS, IN ACRE-FEET, BY THOMPSON CREEK FEEDER DITCH (FURNISHED BY STATE ENGINEER OF COLORADO).
NOTE.--NO GAGE-HEIGHT RECORD NOV. 7 TO MAR. 27.

DISCHARGE, IN CUBIC FEET PER SECOND, WATER YEAR OCTOBER 1969 TO SEPTEMBER 1970

DAY	OCT	NOV	DEC	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP
1	2.5	3.1	2.5	2.0	2.1	2.5	2.4	23	133	31	4.0	2.8
2	2.2	3.2	2.5	2.1	2.1	2.7	2.2	26	118	29	3.7	6.8
3	3.7	4.2	2.4	2.1	2.1	2.7	2.0	30	113	26	3.7	2.8
4	5.0	4.8	2.4	2.1	2.1	2.6	2.2	42	109	22	4.5	1.6
5	6.0	5.0	2.4	2.1	2.0	2.5	2.2	54	100	18	3.7	12
6	6.5	5.0	2.4	2.0	2.1	2.5	2.4	76	98	15	4.5	31
7	6.0	4.8	2.3	2.0	2.1	2.6	2.6	64	114	20	6.0	7.8
8	8.5	4.6	2.2	1.9	2.0	2.7	2.8	66	146	17	3.7	3.7
9	8.0	4.4	2.1	2.0	1.8	2.7	3.2	76	128	17	3.1	2.9
10	8.0	4.2	2.3	2.3	2.0	2.5	3.6	89	156	28	2.2	1.6
11	7.5	3.8	2.4	2.3	2.1	2.6	3.8	143	146	23	1.9	1.3
12	7.5	3.6	2.3	2.3	2.1	2.6	3.8	181	128	23	1.3	2.9
13	9.0	3.4	2.2	2.2	2.2	2.5	3.6	177	109	17	1.0	21
14	9.0	3.2	2.2	2.2	2.2	2.4	3.4	167	109	13	1.3	13
15	6.5	3.0	2.2	2.2	2.1	2.3	3.4	164	102	11	1.6	9.9
16	7.0	2.8	2.2	2.1	2.0	2.3	3.6	193	95	9.7	2.5	3.0
17	7.0	2.6	2.2	2.1	1.9	2.4	3.8	237	87	8.5	1.6	2.8
18	9.0	2.3	2.3	2.1	1.8	2.4	3.8	258	89	8.0	1.3	2.4
19	8.5	2.3	2.3	2.1	1.8	2.3	3.6	265	87	7.5	1.3	2.2
20	7.0	2.6	2.3	2.1	1.8	2.2	3.4	272	87	7.0	2.2	1.8
21	11	2.6	2.3	2.0	2.0	2.3	3.6	262	82	6.0	5.0	1.8
22	7.0	3.0	2.3	2.0	2.2	2.4	3.4	270	80	6.5	4.5	4.9
23	7.5	2.9	2.3	2.1	2.2	2.3	3.2	265	76	15	2.8	4.3
24	6.0	2.9	2.3	2.1	2.1	2.2	3.2	235	69	7.5	1.9	4.4
25	5.5	3.0	2.2	2.0	2.3	2.2	4.0	217	63	6.0	1.9	3.4
26	4.0	2.8	2.2	2.0	2.3	2.3	15	217	61	10	1.6	2.8
27	4.5	2.6	2.1	2.0	2.2	2.2	23	199	56	8.0	1.9	2.4
28	3.1	2.4	2.0	2.1	2.2	2.2	22	179	49	9.7	1.6	2.4
29	3.0	2.3	1.9	1.8	———	2.2	21	167	41	9.7	1.9	2.2
30	3.0	2.5	1.8	1.6	———	2.3	22	160	35	5.5	1.6	1.9
31	3.0	———	2.0	2.0	———	2.4	———	162	———	5.0	1.0	———
TOTAL	192.0	100.1	69.5	64.0	57.9	74.6	184.2	4,976	2,866	448.6	80.8	153.7
MEAN	6.19	3.34	2.24	2.06	2.07	2.41	6.14	161	95.5	14.5	2.61	5.12
MAX	11	5.0	2.5	2.3	2.3	2.7	23	272	156	31	6.0	31
MIN	2.2	2.3	1.8	1.6	1.8	2.2	2.0	23	35	5.0	1.0	1.3
AC-FT	381	199	138	127	115	148	365	9,870	5,680	890	160	320
(†)	0	0	0	0	0	0	0	0	0	181	0	3

CAL YR 1969 TOTAL 5,489.92 MEAN 15.0 MAX 129 MIN .73 AC-FT 10,890 † 836
WTR YR 1970 TOTAL 9,267.40 MEAN 25.4 MAX 272 MIN 1.0 AC-FT 18,380 † 181

† DIVERSIONS, IN ACRE-FEET, BY THOMPSON CREEK FEEDER DITCH (FURNISHED BY STATE ENGINEER OF COLORADO).
NOTE.--NO GAGE-HEIGHT RECORD NOV. 2 TO APR. 27.

APPENDIX C

Table C-1.--Analytical data on coal samples, Carbondale mining area

Locality[1/]	Mine-location Coal bed	Source	Size and kind of coal sample[2/]	Moisture (as rec'd)	Proximate[2/], percent Dry coal Vol.	F.C.	Ash	Ultimate[2/], percent Sulfur	Hydrogen	Carbon	Nitrogen	Oxygen	Calorific value (Btu) as received	dry	Ash softening temp., °F	Free Swelling Index[2/] (FSI)	Hardgrove Grindability Index (HGI)
(1)	Diamond mine																
	"A" bed	USBM (1942)		6.7	36.2	54.6	2.5	0.5	5.4	70.4	1.5	19.7	12,250	---	---	---	---
	"D" bed	USBM (1942)		10.7	35.9	49.4	4.0	1.0	5.9	66.0	1.7	21.5	11,660	---	---	---	---
(2)	Sunlight mine																
	D bed	RI 5221	1½"x½"-T	6.2	42.6	54.0	3.4	0.6	5.5	76.1	1.9	12.5	12,840	13,680	2,670	1	47
	D bed	RI 5270	3"x3/8"-T	4.4	42.2	54.0	3.8	0.9	5.5	76.1	1.9	12.5	13,010	13,600	2,650	---	48
	D bed	USBM-Bull.516	1½" lump-T	5.5	42.8	53.7	3.5	0.7	5.5	76.8	1.8	11.7	12,980	13,730	2,380	---	---
	D(?) bed	Collins (1976)	Stockpile	6.7	40.6	58.7	1.7	0.6	5.2	77.7	2.0	13.1	12,779	13,697	2,460	0	---
(3)	Aspen Gulch																
	"B"(?) bed	Collins (1976)	Mine sample	3.1	37.3	55.5	7.2	0.6	5.3	77.3	1.8	7.9	13,176	13,598	2,700	1½	---
	Marion																
	Sunshine(?) bed	RI 5270	1½" lump-T	3.8	40.2	56.1	3.7	0.5	5.4	77.9	1.8	10.7	13,530	14,060	2,160	---	---
		RI 5270	1½"x½"	3.9	39.9	55.0	5.1	0.5	---	---	---	---	13,260	13,810	2,230	---	52
		RI 5270	½"xo	4.2	38.9	54.6	6.5	0.6	---	---	---	---	12,970	13,540	2,420	2½	---
(4)	Thompson Cr. No.1																
	A bed	RI 5270	ROM	2.6	30.8	60.3	8.9	1.0	5.2	76.5	2.0	6.4	13,750	14,110	2,500	7	89
	A bed	RI 6086	ROM	3.5	30.3	55.6	14.1	1.2	4.9	72.8	1.9	5.1	12,830	13,300	2,870	8	83
	No. 2 mine																
	A(?) B(?) bed	RI 6086	ROM	3.1	29.5	58.2	12.3	1.1	---	---	---	---	13,220	13,640	2,470	8	86
	No. 3 mine																
	Anderson bed	RI 6086	ROM	2.3	34.9	57.5	7.6	0.5	---	---	---	---	13,900	14,230	2,420	8	63
	Anderson bed	RI 5270	ROM	3.6	32.4	59.5	8.1	0.7	5.1	79.0	1.7	5.4	13,590	14,100	2,440	7	71
(5)	Middle Thompson Cr.																
	"Prospect"	Collins (1976)		3.0	27.0	59.5	13.5	1.3	4.9	74.6	2.0	3.5	12,982	13,383	2,730	8½	---
	Anderson bed	Collins (1976)		5.1	33.6	57.4	9.0	0.6	5.3	77.1	2.0	6.1	13,150	13,857	2,495	8½	---
(6)	Dutch Creek B bed	IC 8497	ROM	4.9	23.0	68.6	8.4	0.6	4.8	81.9	2.0	2.3	13,780	14,480	2,250	9	110
	Dutch Creek No. 1																
	B bed	Collins (1976)	Mine sample	1.56	22.6	69.8	7.6	0.7	4.8	81.8	2.0	2.9	14,322	14,555	2,460	9+	---
	B bed	Collins (1976)	Mine sample	1.7	22.0	66.8	11.2	0.5	4.6	79.4	1.8	2.5	13,759	13,997	2,370	9+	---
	B bed	Collins (1976)	Mine sample	2.13	23.1	69.6	7.3	0.6	4.8	82.4	2.0	2.8	14,297	14,604	2,340	9+	---
	Dutch Creek No. 2																
	Dutch Creek bed	Collins (1976)	Mine sample	2.58	28.6	67.2	4.2	2.4	5.3	83.3	1.8	2.9	14,455	14,841	2,240	9	---
	Dutch Creek bed	Collins (1976)	Mine sample	3.08	21.2	74.9	3.9	0.6	5.2	85.5	2.0	2.9	14,697	15,167	2,300	9	---
	No. 3, L.S. Wood																
	B bed	Collins (1976)	Mine sample	4.0	22.6	69.6	7.8	0.8	4.9	83.3	1.9	1.5	13,940	14,521	2,370	9+	---
(7)	No. 4, Bear Creek																
	B bed	Collins (1976)	Mine sample	3.8	26.1	70.9	3.0	0.6	4.9	84.7	2.0	4.9	14,397	14,966	2,465	4	---
	B bed	Collins (1976)	Mine sample	5.1	24.5	69.3	6.2	0.5	4.7	81.8	2.0	5.0	13,661	14,395	2,500	4½	---
(8)	Placita mine(s)																
	Sunshine(?) bed	USBM TP 574	Mine sample	2.2	33.8	57.4	6.6	0.5	5.3	78.3	1.7	7.6	---	14,170	2,370	---	---
	Sunshine bed	USBM #865,175	Mine sample	2.5	31.4	60.4	5.7	0.5	5.5	79.1	1.9	7.3	---	14,110	---	---	---
	Coryell mine																
	Sunshine(?)	USBM TP 574	Mine sample	2.3	30.5	62.2	5.0	0.6	5.4	80.2	1.9	6.9	---	14,350	---	---	---

1/ Localities 1 through 8 are shown on figure 4, this report.

2/ For samples collected by Collins (1976): proximate and free swelling index (FSI) done by Coal Laboratory of Eastern Associated Coal Corp., Pittsburgh, C. W. Lype, Chief Chemist; ultimate and high temperature ash analyses by U.S. Bureau of Mines.

APPENDIX C

Table C-2.--ASTM classification of coals by rank

Class	Group	Fixed Carbon Limits, percent (Dry, Mineral-Matter-Free Basis)		Volatile Matter Limits, percent (Dry, Mineral-Matter-Free Basis)		Calorific Value Limits, Btu per pound (Moist,[b] Mineral-Matter-Free Basis)		Agglomerating Character
		Equal or Greater Than	Less Than	Greater Than	Equal or Less Than	Equal or Greater Than	Less Than	
I. Anthracitic	1. Meta-anthracite	98	...	...	2	...	...	nonagglomerating
	2. Anthracite	92	98	2	8	...	...	
	3. Semianthracite[c]	86	92	8	14	...	...	
II. Bituminous	1. Low volatile bituminous coal	78	86	14	22	...	...	Commonly agglomerating[e]
	2. Medium volatile bituminous coal	69	78	22	31	...	...	
	3. High volatile *A* bituminous coal	...	69	31	...	14 000[d]	...	
	4. High volatile *B* bituminous coal	...	...	...	...	13 000[d]	14 000	
	5. High volatile *C* bituminous coal	...	...	...	...	11 500	13 000	
						10 500	11 500	agglomerating
III. Subbituminous	1. Subbituminous *A* coal	...	...	...	...	10 500	11 500	nonagglomerating
	2. Subbituminous *B* coal	...	...	...	...	9 500	10 500	
	3. Subbituminous *C* coal	...	...	...	...	8 300	9 500	
IV. Lignitic	1. Lignite *A*	...	...	...	...	6 300	8 300	
	2. Lignite *B*	...	...	...	...	...	6 300	

[a] This classification does not include a few coals, principally nonbanded varieties, which have unusual physical and chemical properties and which come within the limits of fixed carbon or calorific value of the high-volatile bituminous and subbituminous ranks. All of these coals either contain less than 48 percent dry, mineral-matter-free fixed carbon or have more than 15,500 moist, mineral-matter-free British thermal units per pound.

[b] Moist refers to coal containing its natural inherent moisture but not including visible water on the surface of the coal.

[c] If agglomerating, classify in low-volatile group of the bituminous class.

[d] Coals having 69 percent or more fixed carbon on the dry, mineral-matter-free basis shall be classified according to fixed carbon, regardless of calorific value.

[e] It is recognized that there may be nonagglomerating varieties in these groups of the bituminous class, and there are notable exceptions in high volatile *C* bituminous group.

Table C-3.--Total production of coal mines, Carbondale mining area.

Data Source [1/]	Mine, location, operator (last or latest)	Coal bed (zone)	Opened	Closed	Total production (short tons)
(2)	Diamond Sec. 8, T. 7 S., R. 89 W.	A-B (lower)	1898	1951	122,031
(2)	Pocahontas #1 Sec. 27, T. 7 S., R. 89 W.	(lower)	1897	1907	175,136
(1)	Four Mile (Sunlight) Sec. 34, T. 7 S., R. 89 W. Four Mile Coal Co. Inc.	A, C and D	1888	1958 (now open)	642,031
(1)	Midland Sec. 33, 34, T. 7 S. Rocky Mtn. Fuel Co.	A, B, C, and D	1896	1919	1,286,376
(2)	Marion Sec. 10, T. 8 S., R. 89 W.	A Allen- Anderson	1885	1927	378,985
(2)	Union Sec. 15, T. 8 S., R. 89 W.	(lower)	1896	1902	142,765
(1)	Spring Gulch Sec. 15, 22,23,26, and 27 T. 8 S., R. 89 W. C.F.& I. Steel Corp.	Sunshine Allen Anderson (upper)	1887	1916	3,372,385
(1)	Thompson Creek No. 1 Sec. 34 and 35, T. 8 S., R. 89 W. Anschutz Coal Corp.	A and B	1951	1966 (reopened 1975)	1,079,166
(2)	Thompson Creek No. 2 Sec. 34, T. 8 S., R. 89 W. Anschutz Coal Corp.	A and B	1953	1966	226,893
(1)	Thompson Creek No. 3 Sec. 34, T. 8 S., R. 89 W. Anschutz Coal Corp.	Anderson	1955	1966 (reopened 1975)	672,206
(2)	Bear Creek Sec. 21, T. 10 S., R. 89 W. Mid Continent Coal & Coke Co.	B	1970	----	551,170
(2)	Coal Basin Sec. 5,6,7, and 8, T. 10 S., R. 89 W. Mid Continent Coal & Coke Co.	B	1900 1973	1908 ----	1,122,160
(2)	Dutch Creek No. 1 Sec. 17, T. 10 S., R. 89 W. Mid Continent Coal & Coke Co.	B (lower)	1956	----	5,605,080
(2)	Dutch Creek No. 2 Sec. 17, T. 10 S., R. 89 W. Mid Continent Coal & Coke Co.	Dutch Creek	1965	----	749,595
(2)	L. S. Wood Sec. 8, T. 10 S., R. 89 W. Mid Continent Coal & Coke Co.	"B"	1966	----	2,998,272
	Placita Sec. 6, T. 11 S., R. 89 W.	(upper)	1899	1926	151,367

1/ Sources of data:
(1) Jones and Murray (1976)
(2) Collins (1976)

Table C-4.--Estimated original reserves (in thousands of short tons) in coal beds more than 42 inches thick, Carbondale mining area 1/

Coal Zone	Location, coal bed	Overburden (ft) 0-1,000	1,000-2,000	2,000-3,000	Totals
	T. 7 S., R. 89 W., Garfield County				
Lower	Diamond	2,714	1,860	2,207	6,781
	Upper A	18,342	7,508	9,015	34,865
	A	12,396	4,476	3,345	20,219
	B	14,847	5,661	5,325	25,833
	C	22,853	9,263	8,812	41,028
	D	4,998	2,664	2,329	9,991
	Pocahontas 2/	18,570	6,899	7,256	32,725
	Total, lower zone	94,720	38,333	38,389	171,442
Upper	Sunshine	1,791	2,052	1,665	5,508
	Anderson	6,444	6,188	5,180	17,812
	Total upper zone	8,235	8,240	6,845	23,320
Township total		102,955	46,573	45,234	194,762
	T. 8 S., R. 89 W., Garfield and Pitkin Counties				
Lower	A	9,150	14,272	21,369	44,791
	B	6,405	7,977	12,606	26,988
	C	4,778	5,485	9,677	19,940
	D	4,648	5,619	5,722	15,989
	Total, lower zone	24,931	33,353	49,374	107,708
Upper	Sunshine	10,067	14,485	15,391	39,943
	Anderson	12,288	11,746	8,622	32,656
	Thompson	1,493	399	---	1,892
	Total, upper zone	23,848	26,630	24,013	74,491
Township total		48,829	59,983	73,387	182,199
	T. 9 S., R. 89 W., Pitkin County				
Lower	A	5,505	11,145	12,632	29,282
	Upper split, lower B	5,911	3,030	4,102	13,043
	Lower split, lower B	2,324	2,257	4,352	8,933
	Upper B	1,148	---	---	1,148
	Total, lower zone	14,888	16,432	21,086	52,406
Upper	Sunshine	17,534	22,283	15,228	55,045
	North Rim	4,799	4,350	---	9,149
	Lake Ridge	548	660	2,818	4,026
	Total, upper zone	22,681	27,293	18,046	68,220
Township total		37,769	43,725	39,132	120,626
	T. 10 S., R. 89 W., Pitkin and Gunnison Counties				
Lower	A	7,561	9,419	50,895	67,875
	Upper B	7,785	7,676	5,049	20,510
	B and Lower B	17,601	16,075	33,245	66,911
	C	7,549	8,603	---	16,152
	Total, lower zone	40,496	41,773	89,189	171,438
Middle	Huntsman (Dutch Creek)	17,353	53,351	68,427	139,131
	Total, middle zone	17,353	53,351	68,427	139,131
Upper	Sunshine	24,198	17,917	14,214	56,329
	Upper split, North Rim	68	---	---	68
	Total, upper zone	24,266	17,917	14,214	56,397
Township total		82,215	113,041	171,830	
Total, all townships		271,668	263,322	329,583	864,573

1/ Source of data: Donnell (1962). Resource estimations do not conform to present Geological Survey standards, but they are the most complete and accurate data available for the area as of December 1977.

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