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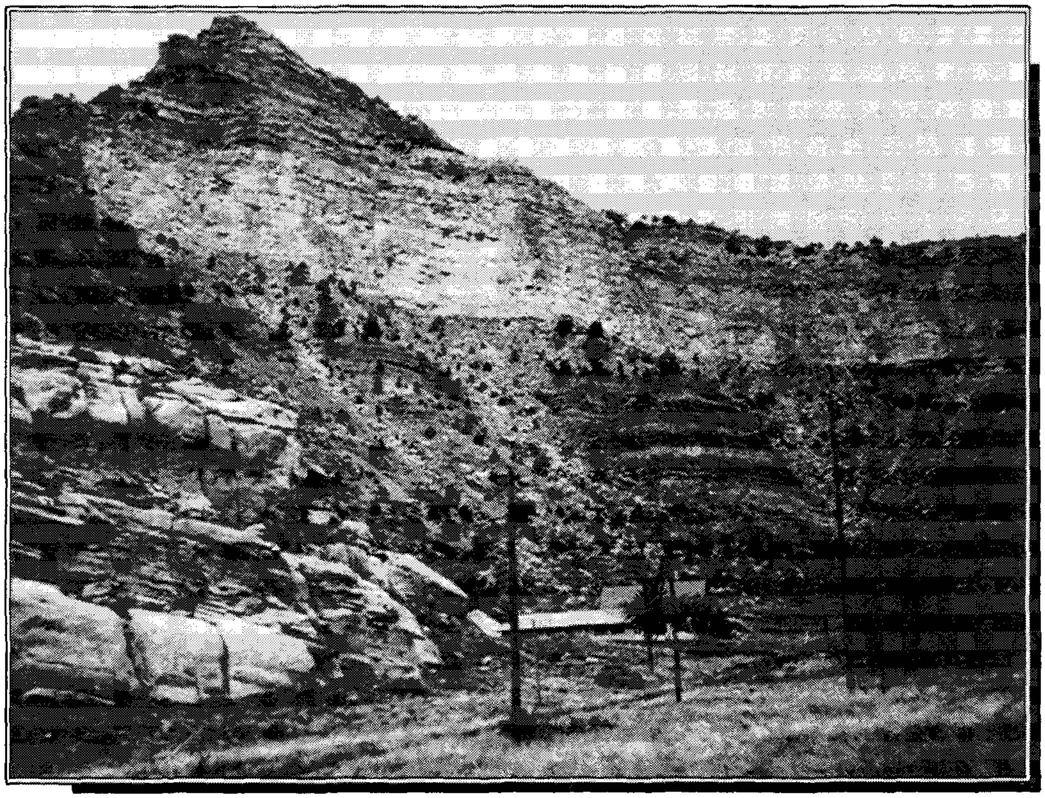
Longwall Gate Road Stability in Four Deep Western U.S. Coal Mines

By Lance R. Barron, Matthew J. DeMarco,
and Richard O. Kneisley



United States Department of the Interior

Bureau of Mines

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Cover: Massive overburden strata of Book Cliffs Coalfield at Sunnyside No. 1 Mine.

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Over the past decade, the U.S. Bureau of Mines (USBM) conducted longwall gate road stability studies at four mines in the Book Cliffs and Wasatch Plateau Coalfields of Utah. These operations are characterized by multiple-seam mining, abruptly varying cover depths to 914 m (3,000 ft), and massive rigid sandstone units in the main roof or floor. Such conditions encourage severe bumping, roof instability, and occasional floor heave problems. Various gate road configurations have been employed to alleviate these problems. Though the mines have comparable basic conditions, lithologies and qualities of the immediate roof, floor, and seam are different in each mine, and often vary in a single mine. Gate road systems which mitigate hazards in one mine may prove inappropriate in another or for different areas of the same mine.

This report relates the geology of the coalfields; describes the location, geologic setting, specific mining conditions, operating history, and the USBM field study and results or conclusions for each mine; summarizes the relative performance of the gate road systems, and emphasizes the need for site-specific geotechnical data to evaluate gate road conditions and performance.

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UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

Metric Units

cm	centimeter	m/km	meter per kilometer
km	kilometer	mm	millimeter
kPa	kilopascal	MPa	megapascal
m	meter	Mt	million metric tons
min	minute	t	metric ton

U.S. Customary Units

ft	foot	psi	pound (force) per square inch
ft/mi	foot per mile	psi/ft	pound per square inch per foot
in	inch		

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LONGWALL GATE ROAD STABILITY IN FOUR DEEP WESTERN U.S. COAL MINES

By Lance R. Barron,[1] Matthew J. DeMarco,[1] and Richard O. Kneisley[1]

ABSTRACT

Over the past decade, the U.S. Bureau of Mines (USBM) has studied longwall gate road stability at four mines in the Book Cliffs and Wasatch Plateau Coalfields of Utah. These operations are characterized by multiple-seam mining, abruptly varying cover depths to 914 m (3,000 ft), and massive rigid sandstone units in the main roof or floor. Such conditions encourage severe bumping, roof instability, and occasional floor heave problems. Various gate road configurations have been employed to alleviate these problems. Although the mines have comparable basic conditions, the lithologies and qualities of the immediate roof, floor, and seam are different in each mine and often vary in a single mine. Gate road systems that mitigate hazards in one mine may prove inappropriate in another or for different areas of the same mine.

This report relates the geology of the coalfields; describes the location, geologic setting, specific mining conditions, operating history, and the USBM field study and results or conclusions for each mine; summarizes the relative performance of the gate road systems; and emphasizes the need for site-specific geotechnical data to evaluate gate road conditions and performance.

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INTRODUCTION

Longwall mining under deep cover in mountainous terrain presents some of the most difficult and challenging ground stability problems found in U.S. underground coal mines. Stress-induced and geology-controlled ground instabilities and hazards, such as pillar and face bumps, roof falls along the longwall faces and in the gate entries, unstable ribs and rib falls, and gate entry closure due to roof sag and/or floor heave, are common in mines operating under these conditions, and they become especially prevalent when thick, strong roof members overlie the mined coal seam.

During the past decade, as part of its ground control research program, particularly the effort to elevate the state of the art in longwall gate road design, the U.S. Bureau of Mines (USBM) has conducted gate road stability studies in four mines located in the Book Cliffs and Wasatch Plateau Coalfields of east-central Utah. Because of the rugged ridge-and-canyon topography of the area (figure 1), the dip of the coal seams away from the outcrops along the escarpment that bounds the district, and the nature and thickness of the strata overlying and underlying the coal seams, mines in these contiguous coalfields are characterized by cover depths of up to 914 m (3,000 ft), abrupt cover depth variation, massive sandstone overburden units, and highly variable roof and floor lithology and competency. Since two or more minable coal seams are present in most areas of the district, stress-concentration effects from overlying workings in multiple-seam operations often contribute to the severity of the adverse ground conditions and add to the difficulty and complexity of designing gate road systems for safe, efficient longwall mining.

Mining in the Book Cliffs and Wasatch Plateau Coalfields began during the latter 1800's. As mine workings advanced downdip from the escarpment outcrops and passed beneath the precipitous elevation rise of the plateau front, ground conditions became increasingly difficult and hazardous. Mountain bumps, the instantaneous, often catastrophic, release of massive amounts of strain energy during the failure of overstressed coal, roof, or floor (usually referred to as "bounces" in the district), were recognized as being prevalent in room-and-pillar workings of district mines as early as 1915 (*1*).[2] In 1918, bumps were also observed to occur in development headings (*2*). In 1932, Tomlinson stated (*3*):

> In the mines where the coal bed is being mined under heavy cover, bounces frequently occur. A "bounce" is the sudden breaking of the coal, often in large pieces, from the face or ribs. In fact, these bounces are the principal cause of accidents from falls in Utah mines.

Proper pillaring practices, such as maintaining long uniform pillar lines and minimizing the number and size of remnant pillars inby the gob lines, helped to reduce the frequency and severity of bumps in room-and-pillar operations. Roof bolting, first adopted in district mines during the early 1950's (*4*), alleviated the number and severity of roof falls resulting from bumps and enabled pillar-extraction operations in deep-cover areas previously deemed unminable. By the 1960's, nevertheless, bumps and roof falls in the deeper workings in the district had become so prevalent and severe that some of these operations had been limited to development-only mining (no pillar extraction), and a number of mines were closed or faced imminent closure. Numerous innovative ground support methods, such as yielding steel arches (*5*), backfilling (*6*), and concrete linings (*7*) were utilized in efforts to cope with the severe conditions. Although generally successful in mitigating particular ground hazards, they were too cost and labor intensive to enable continuation of minimally productive operations.

During the early 1960's, longwall mining with powered hydraulic supports, developed in Europe, was introduced into the Sunnyside Mines in Utah as the mining system that offered the best prospects for safer, more productive, continued operation (*8*). This system proved highly successful in meeting Sunnyside's requirements and was adopted into several district mines during the 1970's and 1980's. Although longwall mining definitely alleviated the worst ground hazards, bumps occasionally occurred in longwall panels, both in the gate roads and at the production face, particularly when mining was beneath large remnant pillars in overlying workings (*9*). In response to this problem, the Sunnyside Mines began developing gate roads utilizing narrow yielding chain pillars, which could gradually crush under load rather than accumulate sufficient strain energy to catastrophically burst or bump (*10*). Also, the number of entries in each gate road was reduced to two in order to minimize exposed roof across the gate roads, the distance between the primary-load-carrying panel blocks, and the number of roof-fall-prone intersections (*11*).

Because utilizing two-entry yield-pillar gate roads generally alleviated bumps and roof falls at Sunnyside, a number of deep-cover western U.S. mines adopted this design. However, the December 1984 Wilberg Mine disaster, near Orangeville, UT, raised questions concerning the interpretation and definition of required safety when

[2]Italic numbers in parentheses refer to items in the list of references at the end of this report.

Figure 1

Ridge-and-canyon topography typical of Book Cliffs and Wasatch Plateau Coalfields.

the escapeway and ventilation benefits offered by a third entry are exchanged for improved ground stability in deep, bump-prone and/or multiple-seam mining. In 1985, a Mine Safety and Health Administration (MSHA) Task Force on Longwall Mining recommended that two-entry gate roads be permitted in these ground conditions on a case-by-case basis if operators complied with stringent requirements for escapeway maintenance, gas detection, and fire and explosion prevention and extinguishing (*12*); however, debate on the relative merits of these gate road systems currently continues.

The USBM has investigated two fundamental gate road pillar design approaches: nonyielding abutment pillar systems and yielding-pillar systems. The Analysis of Longwall Pillar Stability, or ALPS (*13*), empirical approach for sizing abutment pillars, developed by the USBM, has been shown to provide realistic evaluations of gate road performance and should provide consistently acceptable levels of design confidence with current refinements regarding intrinsic gate road geology and the use of artificial support systems.

Design of yield-pillar systems, however, is much more challenging, owing to their inherent complex, interdependent, multiple support components. Although there are several methods available for estimating yield-pillar width for various ground condition parameters, there is currently no complete yield-pillar gate road design methodology integrating pillar support and failure mechanics, roof and floor strata integrity and behavior, postextraction gob support capability, and artificial support. Even though quantitative performance data are somewhat limited, case history evaluations can provide an understanding of the successful application of these gate road designs.

Beginning in 1982, the USBM conducted a series of studies on longwall gate road stability and ground control in the Wilberg, Star Point No. 2, Castle Gate No. 3, and Sunnyside No. 1 Mines in east-central Utah. During these studies, various gate road design parameters, such as overall gate road design, gate road width, pillar sizes, entry widths, and support requirements, were investigated and correlated with intrinsic ground conditions, such as cover depth, geologic structure and stratigraphy, lithology and physical properties of the coal, roof, and floor, and loading attributable to nearby mined-out areas. The various gate road and gate pillar designs were then evaluated according to their relative success in meeting the operators' requirements for stability, safety, and longwall productivity. The results of these studies, in conjunction with the histories of the operators' efforts in developing the gate roads and mining the adjacent longwall panels, constitute an important background for developing gate road design criteria and procedures for future longwall operations.

GENERALIZED GEOLOGY OF BOOK CLIFFS AND WASATCH PLATEAU COALFIELDS

LOCATION AND PHYSIOGRAPHIC SETTING

The portions of the Book Cliffs and Wasatch Plateau Coalfields that include the Wilberg, Star Point No. 2, Castle Gate No. 3, and Sunnyside No. 1 Mines are located along the escarpments of the Book Cliffs and Wasatch Plateau (figure 2) near the city of Price and the towns of Orangeville, Wattis, Helper, and Sunnyside in east-central Utah (figure 3). The Book Cliffs and Wasatch Plateau lie along the northwestern margin of the Colorado Plateau physiographic province. The Book Cliffs Coalfield is the southern edge of the Uintah Basin (*14*), and the Wasatch Plateau Coalfield constitutes the eastern escarpment of the Wasatch Plateau (*15*).

STRATIGRAPHIC FRAMEWORK AND DEPOSITIONAL ENVIRONMENTS

The principal stratigraphic units of the Wasatch Plateau and Book Cliffs Coalfields are, in ascending order, the Mancos Shale, the Star Point Sandstone, the Blackhawk Formation, the Castlegate Sandstone, the Price River Formation, the North Horn Formation, the Flagstaff Limestone, and the Colton Formation. Generalized stratigraphic sections for these coalfields are shown in figure 4.

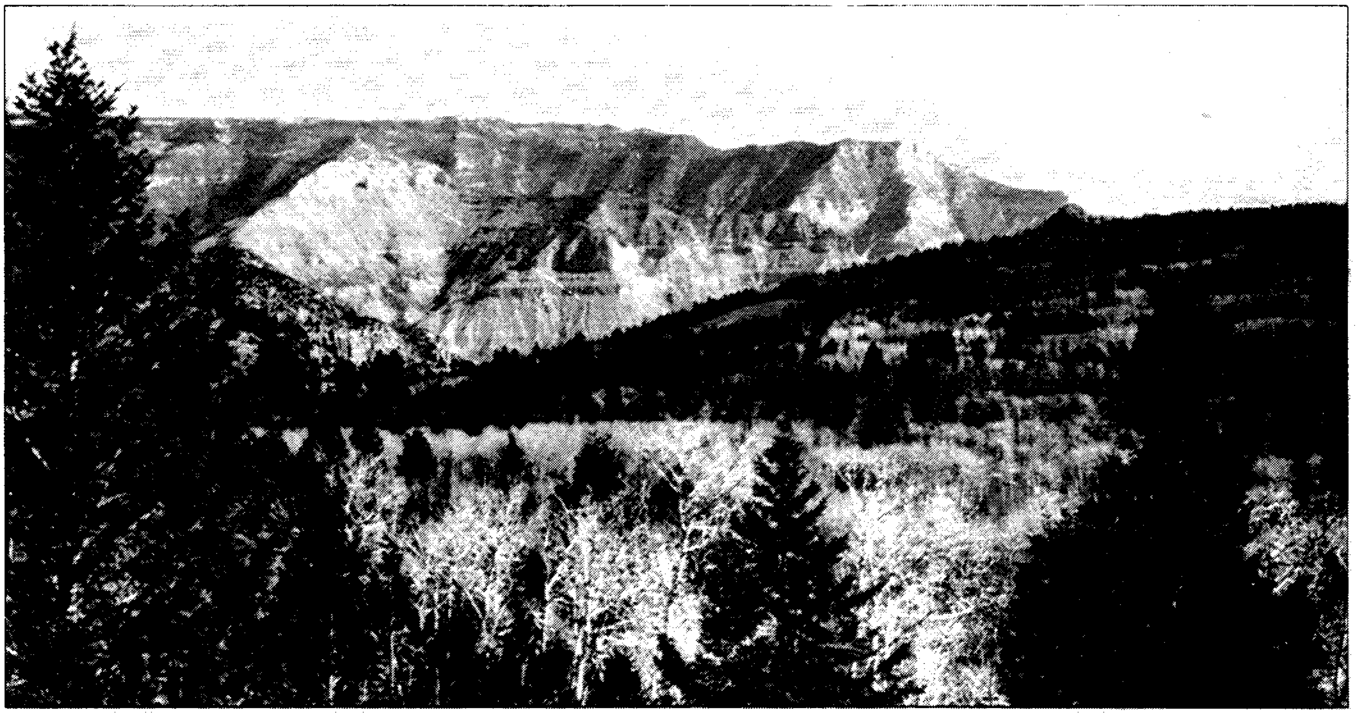
Mancos Shale

Throughout most of the Cretaceous period, a shallow sea covered much of the area that is now eastern Utah and western Colorado. The Mancos Shale was deposited in this marine environment; it consists of bluish-gray shale with a few lenses of limestone and calcareous sandstone. The Masuk Tongue is the uppermost Mancos member at the central part of the Wasatch Plateau Coalfield near Orangeville (*15*), while the uppermost Mancos unit present along the northern Wasatch Plateau and the Book Cliffs Coalfields is the Blue Gate Member (*14*, *16*). The Mancos Shale is seen in the flatland surface at the foot of the escarpments and forms the basal slopes of the escarpments.

Star Point Sandstone

During the late Cretaceous, the Mancos sea began to recede eastward, and beach sand sediments that had been eroded from the Sevier orogenic belt highlands to the west

Figure 2



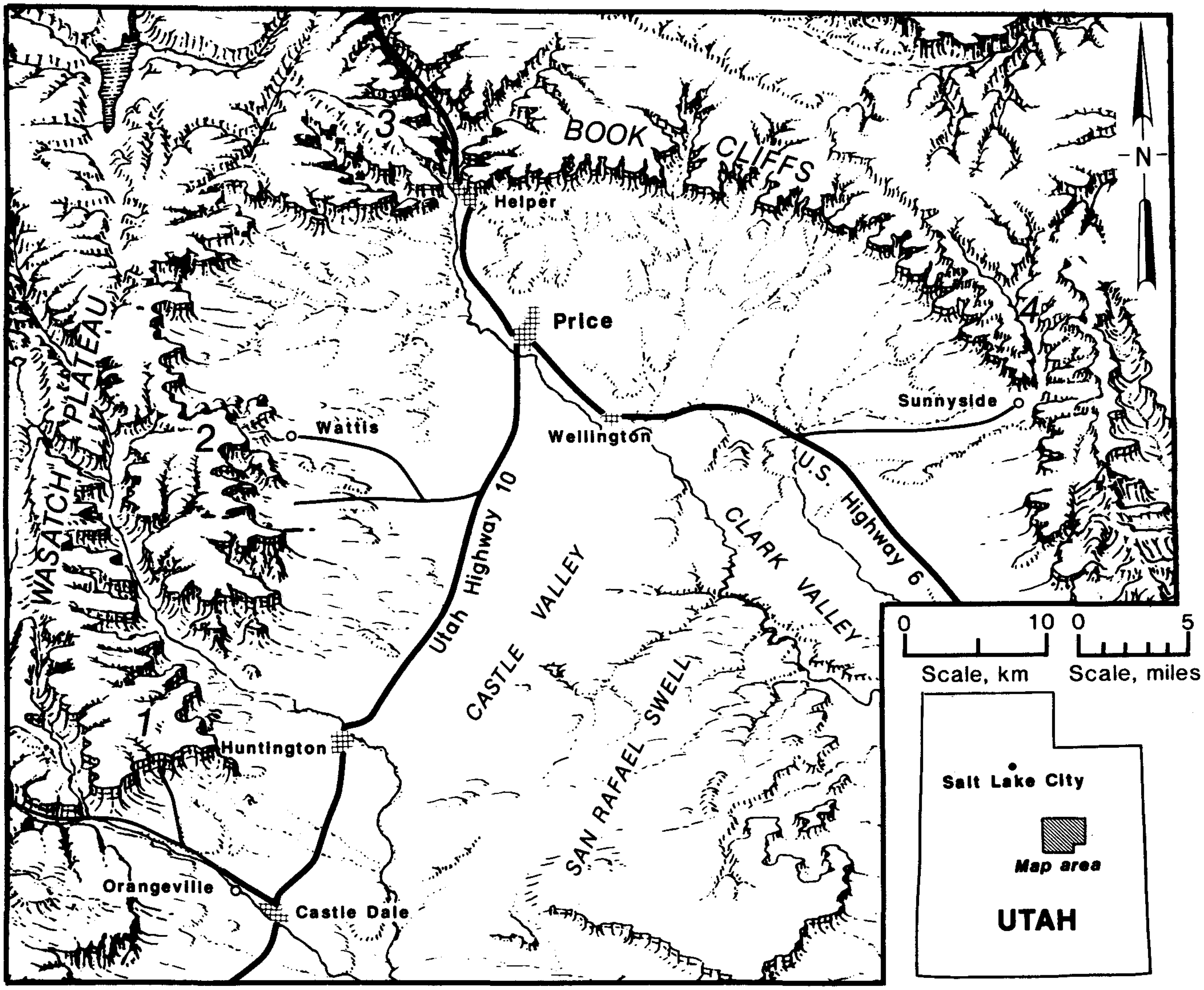
Escarpment of eastern Wasatch Plateau near Orangeville.

were deposited upon the Mancos marine sediments, forming the Star Point Sandstone (*17*). This eastward regression, rather than occurring as a uniform, uninterrupted process, took place during a number of distinct periods, separated by interludes of landward transgression probably caused by sudden "pulses" of basinal subsidence (*18*). Thus, the Star Point Sandstone consists of several littoral quartzose sandstone tongues, deposited during the periods of regression, which thin to feather edges eastward, separated by westward-pointing tongues of Mancos Shale. In the northern part of the Wasatch Plateau and western portion of the Book Cliffs, the three most prominent Star Point Sandstone beds are named, in ascending order, the Panther, Storrs, and Spring Canyon Tongues (*19-20*). These sandstone tongues wedge out in the subsurface westward, while eastward they possibly merge with each other and the Aberdeen Sandstone Member of the overlying Blackhawk Formation (*21*), then pinch out into the Mancos Shale. The Star Point Sandstone forms a prominent cliff (or several cliffs, where there are separate sandstone tongues) on the lower slopes of the escarpments of the Wasatch Plateau (figure 5) and western Book Cliffs.

Blackhawk Formation

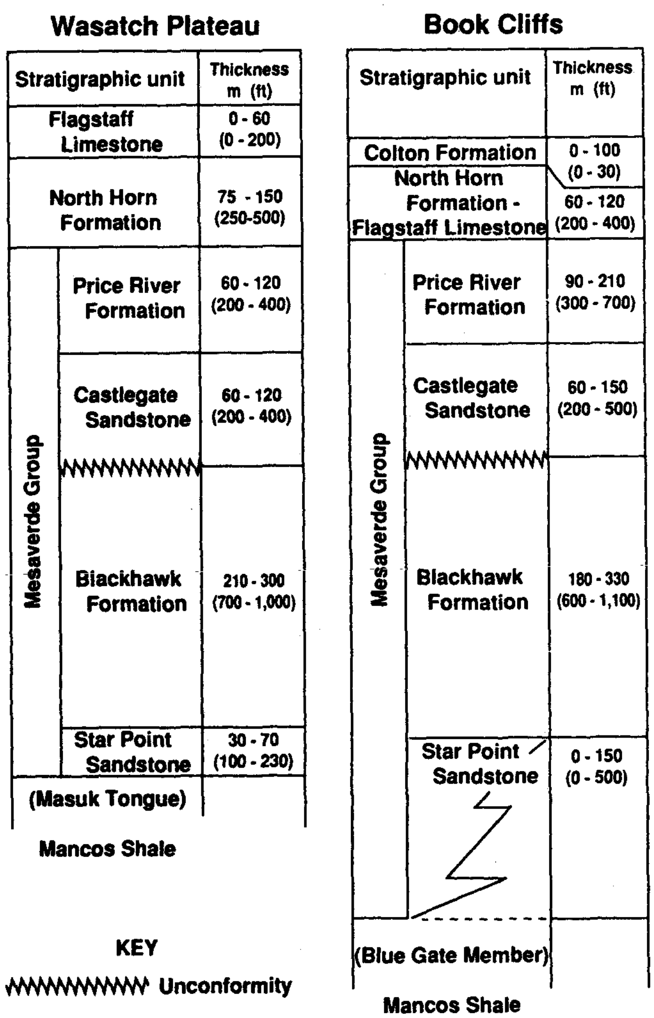
The series of sedimentary rocks that constitute the coal-bearing Blackhawk Formation in the southern and central parts of the Wasatch Plateau Coalfield (figure 6) were deposited in a beach barrier and delta plain setting that trended northwest to southeast along the Cretaceous (Mancos) sea. Regressive deposits accumulated as the shoreline and adjacent environments prograded to the east. Many contemporaneous short-headed delta-front point sources discharged sediment into the sea, which was then reworked into accretion ridges in the direction of long-shore drift. These accretion ridges are represented as tongues of the Star Point Sandstone (*22*). The Black Hawk Formation sediments accumulated in the lagoon, delta, swamp, and floodplain environments that existed both contemporaneously and successively on the landward side of the accretion ridges. The cyclical nature of the sea regressions and transgressions resulted in considerable intertonguing of the lower Blackhawk facies and the Star Point Sandstone (*21*). In the Book Cliffs area, the primary depositional environment was a wave-dominated delta system. This system is represented by fluvial coastal plain, deltaic, and barrier island facies (*22*). The coastal barrier islands of this system had a greater lateral extent and were further from the highland sediment source than those of the central Wasatch Plateau. Large deltaic lobes of sediment, such as sand, silt, mud, and peat, were deposited and abandoned. After abandonment came another episode of marine basin subsidence and sea transgression. Deposition of large sheets of sandstone, such as the

Figure 3



Book Cliffs and Wasatch Plateau Coalfields region, showing locations of the four mines. 1, Wilberg; 2, Star Point No. 2; 3, Castle Gate No. 3; 4, Sunnyside No. 1. (Modified from Ridd, M. K., Landforms of Utah in Proportional Relief. Map Supplement No. 3, Annals of the Association of American Geographers, v. 53, No. 4, December 1963.)

Figure 4



Generalized stratigraphic sections, Wasatch Plateau and Book Cliffs Coalfields.

Aberdeen, Kenilworth, and Sunnyside, represents a regressively deposited sand reworked after delta lobe abandonment and subsequent progradation of the shoreline (*22*). The contrast between the depositional environments of the southern and central Wasatch Plateau and the northern Wasatch Plateau and Book Cliffs areas is shown in figure 7.

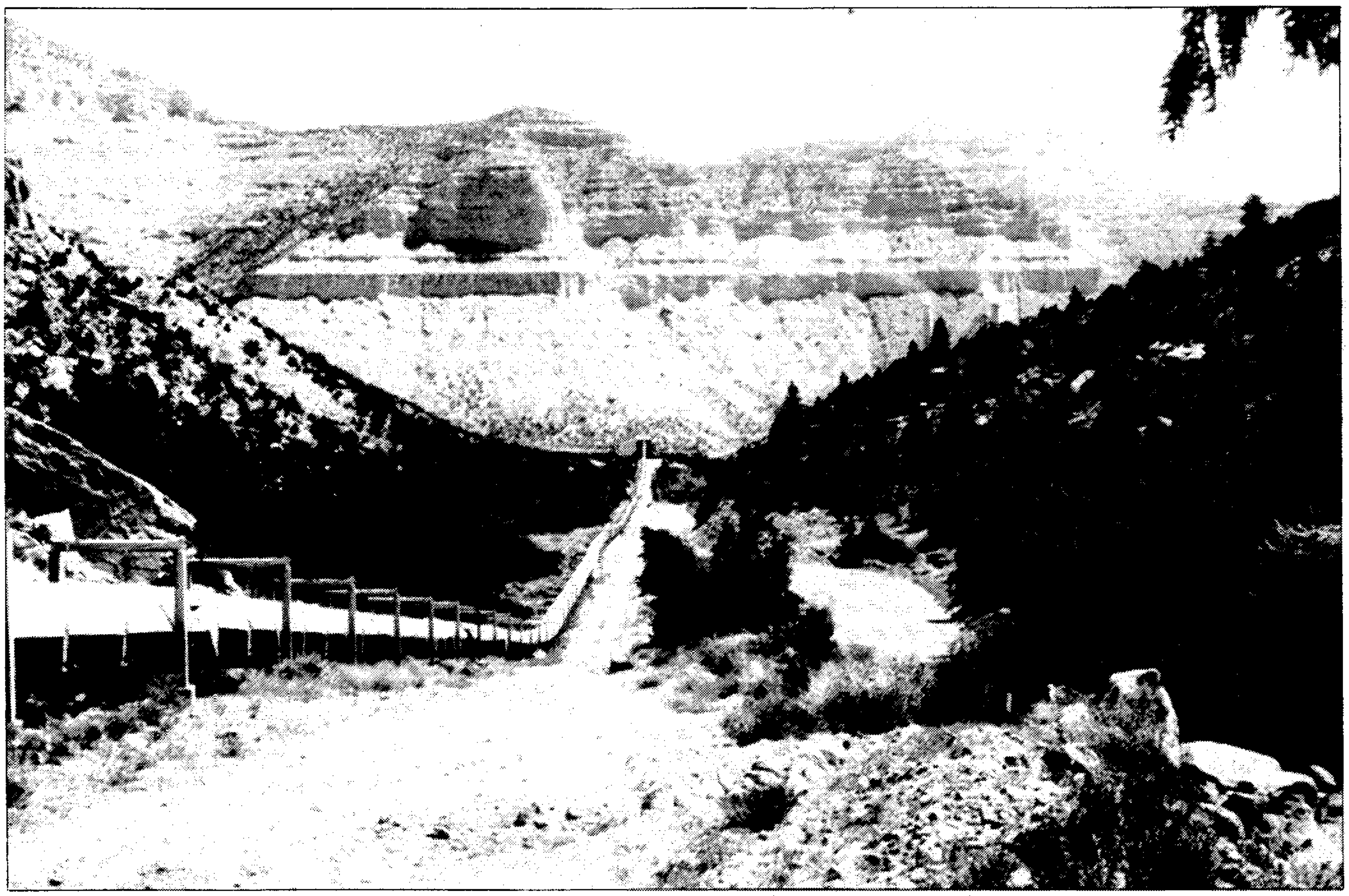
Because of the cyclic nature of the sea regressions and transgressions, the coal-bearing deposits of the Blackhawk appear to have formed under specific geologic and climatic conditions that recurred many times in the same general area during a long span of geologic time. Where completely developed, a typical Blackhawk cyclothem consists of four units: (1) a tongue of Mancos Shale resting disconformably on slightly older transitional deposits, (2) a littoral marine sandstone, which grades downward and eastward into the first unit, (3) a carbonaceous mudstone formed in lagoons and/or estuaries, and (4) a major coal seam (or seams) deposited in a paludal (swamp) environment (*18*). The most prominent units of the cyclothems are the massive beach sandstones, which provide a convenient method for subdividing the coal-bearing strata. Young (*17*) utilized these sandstones in subdividing the Blackhawk into a series of members, each consisting of a basal sandstone and the nonmarine sequence between it and the next higher prominent littoral sandstone. Members present in the area discussed in this publication are, from the base upward, the Spring Canyon (usually considered a tongue of the Star Point Sandstone), Aberdeen, Kenilworth, and Sunnyside (figure 8). The Aberdeen Sandstone disappears from the surface northeast of the town of Wellington, while the Kenilworth and Sunnyside Sandstones continue for 80.5 km (50 miles) southeastward past the report area. As a result of the intermittent eastward regression of the Cretaceous sea, the contact between the littoral marine sandstones and the underlying Mancos Shale rises about 167 m (550 ft) between the towns of Wattis and Sunnyside. The non-coal-bearing undivided upper portion of the Blackhawk is about 198 m (650 ft) thick in the Wasatch Plateau (*23-24*) and westernmost Book Cliffs (*12-13*), thinning to about 61 m (200 ft) near Sunnyside (*25*). The Blackhawk Formation is exposed primarily on the intermediate slopes of the escarpments. The littoral sandstone beds are usually bleached white at the top by organic acids that percolated downward from the swamps that formed the immediately overlying coal seams; they form prominent cliffs and ledges along the intermediate escarpment slopes and are used as marker beds for locating and tracing the associated coal seams.

All the economic coal seams of the Book Cliffs and Wasatch Plateau Coalfields occur in the lower portion of the Blackhawk Formation. Westward, in the Wasatch Plateau and western Book Cliffs, deltaic environments predominated the coal-forming coastal swamps (*22*), while eastward, in the Sunnyside area, the coal seams were deposited in large, geographically continuous swamps in which deltaic features were not as prevalent (*26*). The deltaic nature of the coal-forming swamps in the Wasatch Plateau and western Book Cliffs explains the presence of the extensive stream channel sandstones that occur in the immediate roof strata of the mines in these areas (*27-29*).

Castlegate Sandstone

As the coastline receded eastward, the depositional environments of the Wasatch Plateau and Book Cliffs became progressively more continental as the floodplain built eastward. During the latter part of the Upper Cretaceous, the sediments that form the Castlegate Sandstone were deposited upon the floodplain by a braided stream-channel

Figure 5



Escarpment of Wasatch Plateau near Huntington. Mancos Shale forms barren slope at lower half of exposure; high cliff at midheight of escarpment is Star Point Sandstone; and intertonguing Blackhawk Formation sediments make up upper half of exposure.

network arising from the highlands to the west. The Castlegate Sandstone Formation consists almost entirely of massive, white to gray, cross-bedded sandstone. It is about 91 m (300 ft) thick in the central part of the Wasatch Plateau near Orangeville (*23-24*), increases to about 152 m (500 ft) thick in the western Book Cliffs near Helper (*30-31*), then thins eastward to about 61 m (200 ft) at Sunnyside (*25*) (figure 9). The Castlegate forms a prominent white cliff above the intermediate (Blackhawk) slopes, at or near the top of the Wasatch Plateau and Book Cliffs escarpments.

Price River Formation

The Price River Formation is similar, in most respects, to the underlying Castlegate Sandstone, except that it has numerous intervals of fine-grained clastic sediments because floodplain rather than braided stream deposition dominated the system (*30*). It consists primarily of gray beds of sandstone interbedded with some conglomerate and mudstone. In the central and northern parts of the Wasatch Plateau, it is 76 to 106 m (250 to 350 ft) thick (*23-24*), increases to about 182 m (600 ft) thick in the Book Cliffs north of Helper (*30*), then thins eastward toward Sunnyside (*25*). The Price River Formation forms a repetitive slope-ledge outcrop above the Castlegate Sandstone throughout the coalfields.

North Horn Formation and Flagstaff Limestone

During the last part of the Cretaceous and early Tertiary periods, a large interior lake formed over broad sections of southern, central, and northeastern Utah, forming the depositional environment of the North Horn Formation and Flagstaff Limestone (*14*). The underlying North Horn Formation consists of interbedded claystones, mudstones, limestones, siltstones, and sandstones, representing alternating stream and lake deposition, while the overlying Flagstaff Limestone consists almost entirely of freshwater-lake limestone (*25*). In the central part of the Wasatch

Figure 6

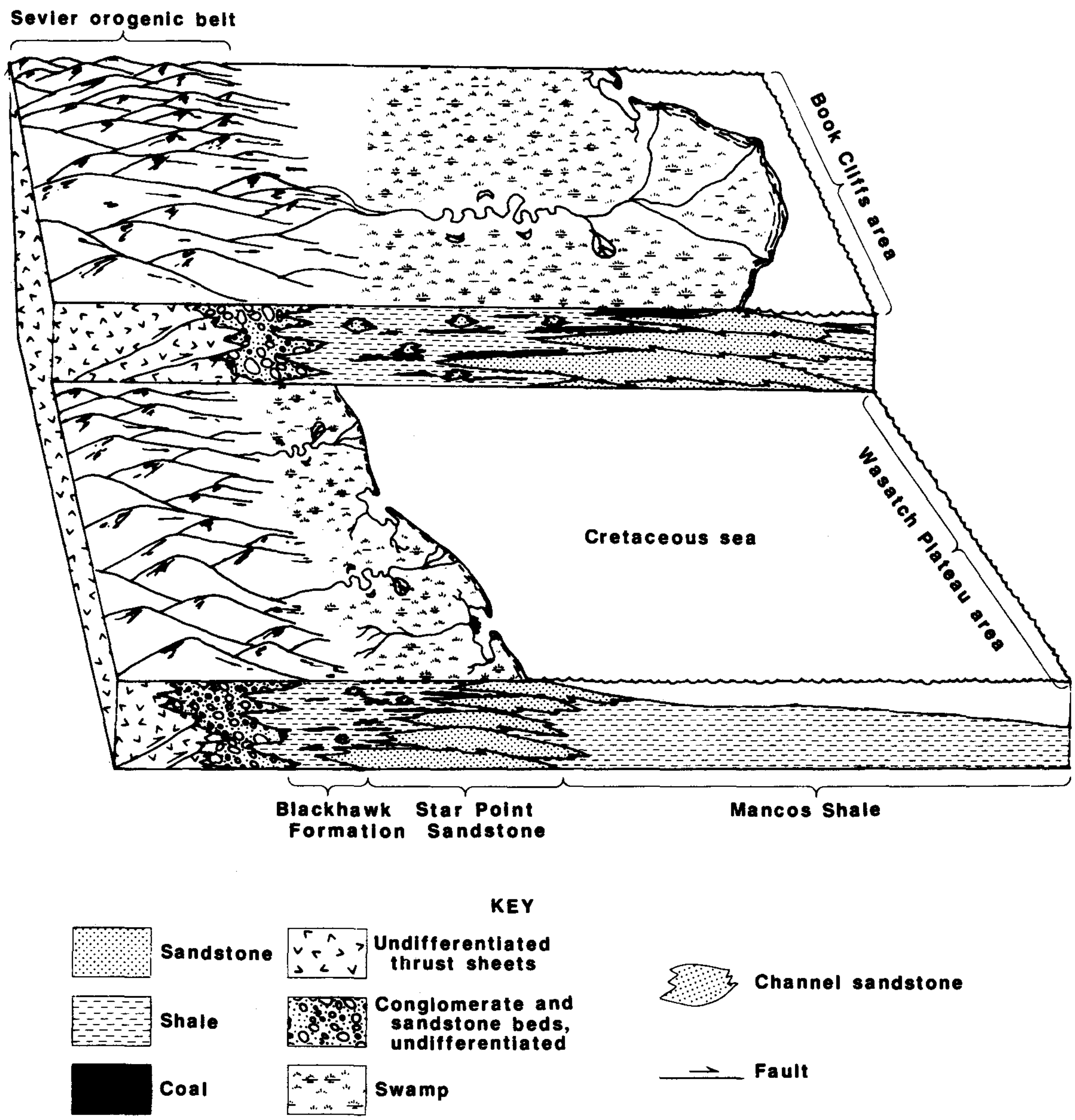
Wasatch Plateau strata near Orangeville. Intertonguing littoral sandstones of Blackhawk Formation form prominent cliffs and ledges above vegetated talus slope, and ridge is capped by the massive Castlegate Sandstone.

Plateau, where the two units are distinct and have been mapped separately, the North Horn Formation is about 152 m (500 ft) thick, and the thickness of the Flagstaff Limestone ranges from 61 to 456 m (200 to 1,500 ft) (*15*). In the area of the mines described in this publication, however, the maximum thickness of the Flagstaff Limestone is about 46 m (150 ft). In the northern part of the Wasatch Plateau and throughout the Book Cliffs, the North Horn Formation and Flagstaff Limestone intertongue extensively and cannot be separated readily; they are, therefore, mapped as an undivided unit (*14*, *16*). This unit is the youngest consolidated sedimentary strata overlying the Castle Gate No. 3 Mine (*28*), and only the lowest part of the formation is present above the deepest workings. In the Sunnyside area, the North Horn-Flagstaff unit ranges from 46 to 91 m (150 to 300 ft) thick (*25*). The unit generally forms the crests of the ridges above the Wasatch Plateau and Book Cliffs escarpments.

Colton Formation

In the areas of the mines included in this publication, the Colton Formation occurs only above the deeper workings of the Sunnyside Mines. In the Sunnyside area, the Colton was deposited in the late Paleocene and early Eocene (Tertiary) periods; it consists of alluvial, lake-margin,

Figure 7



Depositional environments of Wasatch Plateau and Book Cliffs Coalfields. (From Sanchez and Ellis (22).)

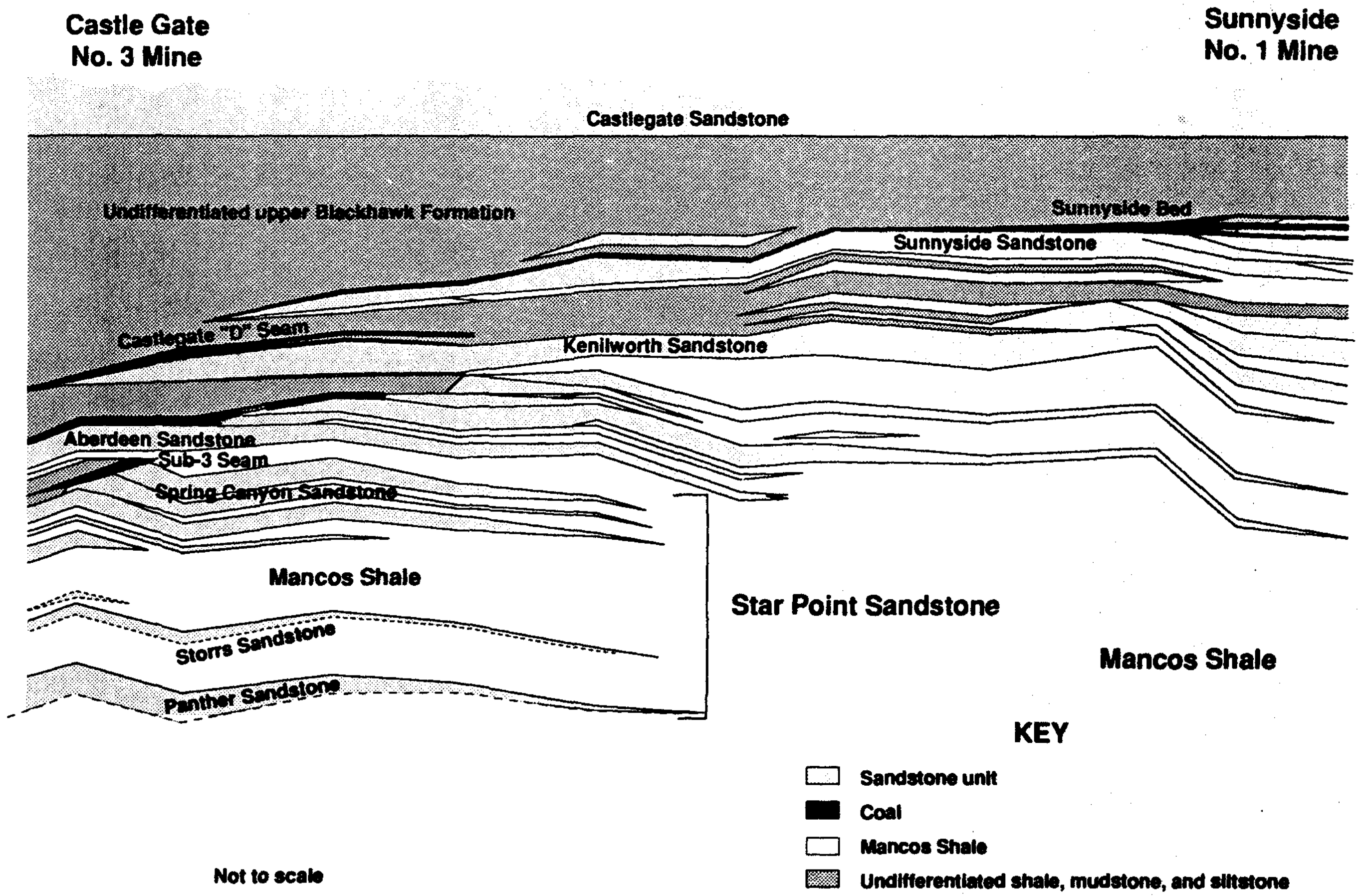
and lake deposits. An especially thick accumulation of stream-deposited materials formed the Colton fan delta, which composes much of the Colton Formation in the Sunnyside area (25). In the Sunnyside No. 1 Mine area, the Colton forms the crest of the highest ridges above the mine.

STRUCTURAL FEATURES

San Rafael Anticline

The San Rafael Swell is a broad, asymmetric, northeast-trending upwarp about 113 km (70 miles) long and some

Figure 8



Stratigraphic diagram of Book Cliffs Coalfield.

48 km (30 miles) across. The north end of the swell appears as a broad wedge flanked on the west by Castle Valley and on the east by Clark Valley (figure 3). This upwarp is part of a much larger, doubly plunging anticline—the San Rafael anticline—that also trends northeast and that extends far beyond the swell. Although the swell (the physiographic unit) and the anticline (the structural unit) are commonly viewed as the same feature because they coincide locally, they are, in fact, wholly different (*14*).

The San Rafael anticline has greatly influenced the surrounding strata. It is expressed best in the eastern Wasatch Plateau and Book Cliffs escarpments, where its slow but continued growth during post-Cretaceous time arched a widespread zone that was once near horizontal. Thus, the strata of the eastern Wasatch Plateau escarpment dip westward, Book Cliffs strata near Helper dip northward, and in the Sunnyside area the Book Cliffs beds dip to the northeast.

Faults

Wasatch Plateau

A system of high-angle normal tensional faults that range in trend from N. 10° W. to about N. 30° E., together with secondary N. 60° to 80° W. compression faulting, breaks the crest and flanks of the entire Wasatch Plateau. Locally, these faults are paired to form grabens that are as much as 64 km (40 miles) long; most of the grabens maintain a relatively constant width of about 3.2 km (2 miles). Both faults and grabens are remarkably straight; commonly, the faults persist as single breaks, or narrow fault zones consisting of numerous en echelon fractures that scissor into the downthrown blocks, and are traceable for very long distances. The faults that bound the grabens invariably dip inward. Structural relief of the grabens varies from 91 m (300 ft) to as much as 912 m (3,000 ft) and progressively increases westward. In many grabens the downthrown blocks are unbroken, but locally they are cut by a series of small internal faults that parallel the larger faults that bound the graben (*15-16*).

The faults of the eastern flank of the Plateau, such as the Pleasant Valley and Deer Creek faults, have drastically affected mine conditions and layouts throughout the Wasatch Plateau Coalfield (*27, 29*). The absence of methane gas in the mines in this area may possibly be attributable to the faulting.

The origin of the faults is uncertain. If they are of tectonic origin, they may reflect widespread crustal spreading or backsliding from deep thrust planes from extensional tectonism, which has dominated the western U.S. interior since Miocene time. The faults may, however, be related to the salt that underlies much of the Sevier-Sanpete Valley area west of the Wasatch Plateau. Many of the faults that break the crest of the Plateau, particularly those bounding large grabens, do not extend below the underlying salt-bearing Jurassic beds, suggesting that at least some of the faults and grabens may be collapse features related to withdrawal of salt (*16*).

Figure 9

Book Cliffs strata at Sunnyside. In ascending order, lowest two prominent sandstones constitute Kenilworth Sandstone, and third is Sunnyside Sandstone. Castlegate Sandstone forms cap of escarpment.

Sunnyside Area

Significant faulting within the report area also lies along the eastern Book Cliffs escarpment near Sunnyside. The faulted zone is much more limited both in linear extent and width, and movements and displacements along the individual faults are much smaller, than in the faulted area of the Wasatch Plateau. Also, Sunnyside area mines are generally gassy. The faulting in this area, however, has significantly affected ground conditions in the Sunnyside Mines.

WILBERG MINE

LOCATION

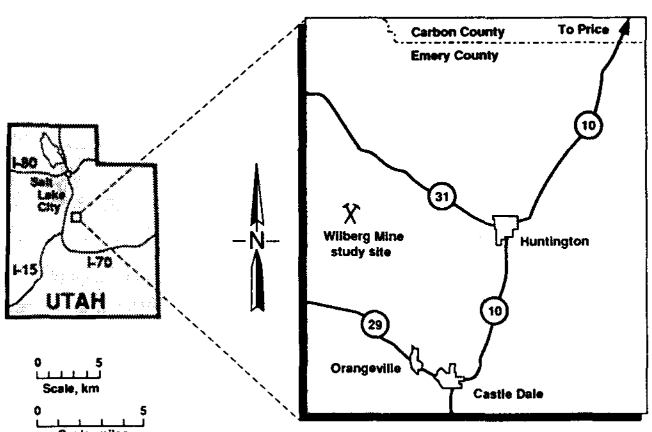
The Wilberg Mine is located at the south flank of East Mountain, a projection of the eastern escarpment of the Wasatch Plateau, approximately 10 km (6 miles) northwest of the small town of Orangeville (figure 3). The mine offices and main portals, in Grimes Wash, are accessed by a paved road that intersects Utah Highway 29 approximately 5 km (3 miles) northwest of Orangeville (figure 10).

GEOLOGIC SETTING

Stratigraphic Units

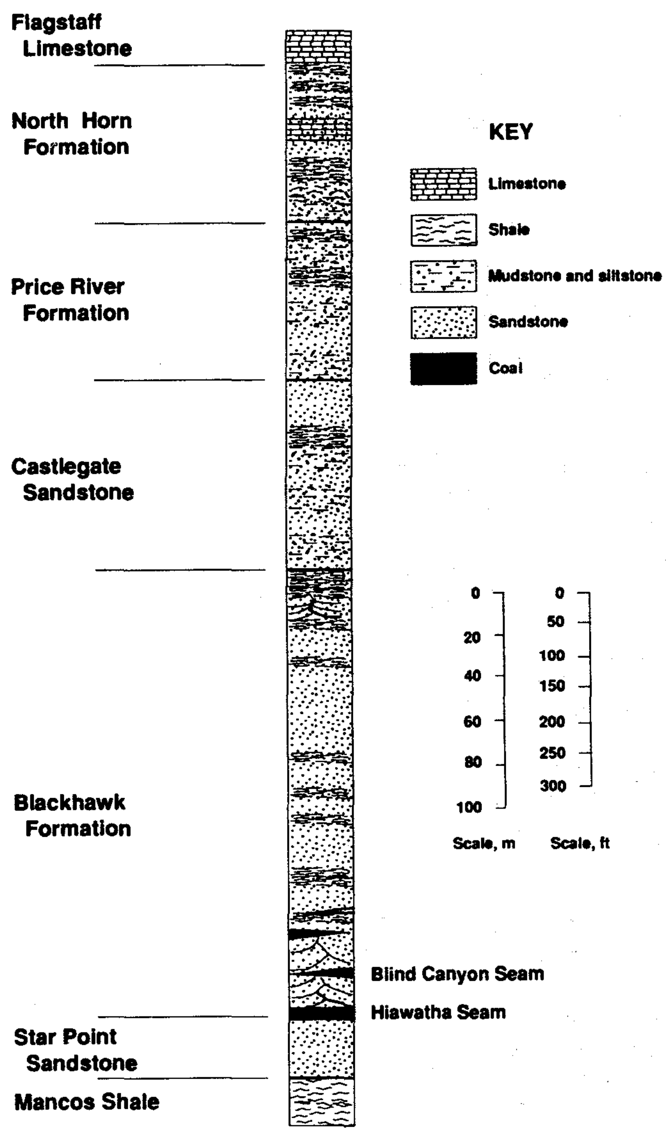
In ascending order, the principal stratigraphic units in the Wilberg Mine area are the Mancos Shale, the Star Point Sandstone, the Blackhawk Formation, the Castlegate Sandstone, the Price River Formation, the North Horn Formation, and the Flagstaff Limestone (figure 11) (*23*). The units most directly affecting Wilberg mining operations are the Star Point Sandstone, the Hiawatha Seam, the strata between the Hiawatha and Blind Canyon Seams, the Blind Canyon Seam, and the Castlegate Sandstone and overlying strata.

Figure 10



Wilberg Mine location map.

Figure 11



Stratigraphic column, Wilberg Mine.

Star Point Sandstone

The Star Point Sandstone constitutes the immediate floor rock of the Wilberg Mine (figure 12). This massive sandstone, approximately 152 m (500 ft) thick in the mine area, provides an exceptionally rigid mine floor wherever it directly contacts the coal seam, as evidenced by very little observable floor heave in the mine workings in these areas.

Hiawatha Coal Seam

The workings of the Wilberg Mine are entirely in the Hiawatha Seam, which averages about 2.4 m (8 ft) thick throughout the mine area (figure 13). The coal is moderately strong and exhibits no predominant cleat or bedding plane. Rock splits in the seam originate from a contemporaneous distributionary channel that replaces the seam north of the mine area.

Interburden Between Hiawatha and Blind Canyon Seams

According to Mercier and Lloyd (*27*), the strata overlying the Hiawatha Seam, which constitute the immediate roof rock of the Wilberg Mine, consist of discontinuous, variably thick, lenticular sandstone, mudstone, siltstone, and thinly bedded sandstones, and exhibit considerable lateral and vertical lithologic variation. The lenticular sandstones were deposited in meandering stream channels in a delta plain, whereas the mudstones, siltstones, and thinly bedded sandstones formed as overbank interchannel deposits. The channel sandstones (figure 13) are generally massive, crossbedded, and fine grained, with rock cementation and strength usually increasing upward from the base. They range from 0.9 to 15.2 m (3 to 50 ft) in thickness, with lateral extent ranging from 9 m (30 ft) to approximately 1.6 km (1 mile). Although roof falls have occurred in channel margin areas in main entries, the channel sandstones generally form strong, competent roof (figure 14), particularly desirable for long-term entries. In panel entries, however, the rigid sandstone roof, in conjunction with the massive, rigid Star Point Sandstone floor, contributes to roof-floor coal seam confinement, which may result in limited chain pillar yielding, coal bursts, and bumps in gate road pillars and possibly on longwall faces.

The areas in which the overbank mudstones, siltstones, and thin-bedded sandstones immediately overlie the coal seam are zones of differential compaction along the channel margins, which closely correlate with significant mine roof failures. The mudstones are commonly slickensided and susceptible to air slacking, and areas of immediate roof composed of interbedded siltstone and mudstone usually contain many bedding plane separations, rendering them difficult to roof bolt securely and prone to shear failure over mined areas.

Blind Canyon Coal Seam

The Blind Canyon Seam, which lies approximately 21 m (70 ft) above the Hiawatha Seam, is mined from the Deer Creek Mine. Large remnant pillars in the Deer Creek Mine may transfer excessive loads onto underlying Wilberg Mine structures, causing extremely difficult, potentially hazardous ground conditions.

Castlegate Sandstone and Other Overlying Strata

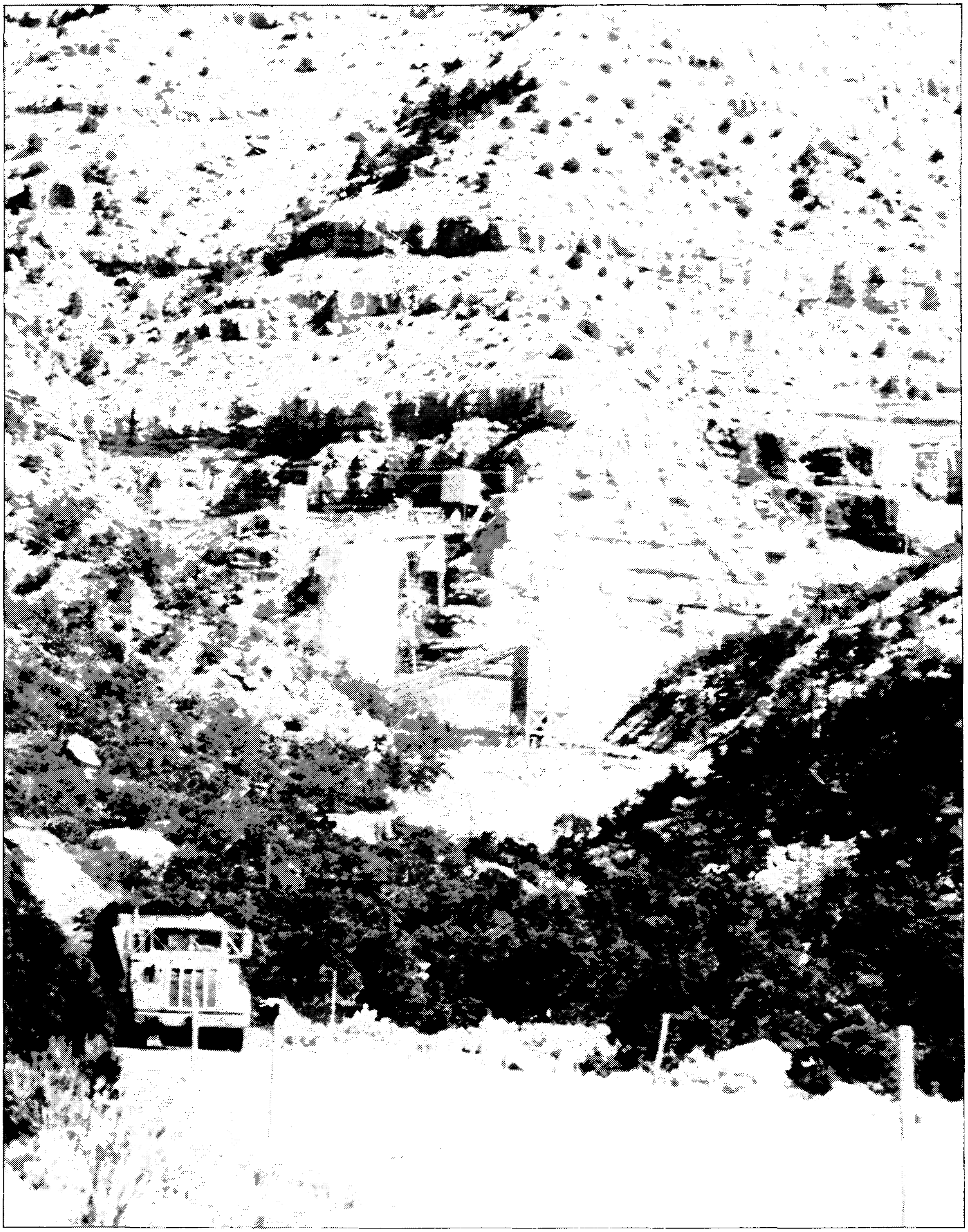
The upper two-thirds of the Blackhawk Formation, the Castlegate Sandstone, the Price River Formation, the North Horn Formation, and Flagstaff Limestone together constitute most of the overburden of the Wilberg Mine (figure 15), which averages approximately 486 m (1,600 ft) in the area of the USBM gate road study. Besides the usual difficult ground conditions expected when mining is conducted under deep cover, the massive, rigid Castlegate Sandstone, which is approximately 91 m (300 ft) thick in the mine area, may possibly delay caving over gob areas. Massive overburden strata were identified in USBM surface subsidence studies of the 1979 to 1984 Deer Creek and Wilberg longwall panels as possibly contributing to delayed surface subsidence (*32-34*); however, a time-domain-reflectometry subsidence study of subsequent panels near the southern escarpment of East Mountain has indicated that near-total subsidence occurs shortly following extraction.

Structural Features

The strata along the east flank of the Wasatch Plateau generally dip gently westward as a result of the uplift of the San Rafael anticline to the east. The Wilberg Mine, however, is located on the southeastern flank of the Straight Canyon syncline, a shallow northeast-southwest-trending fold crossing East Mountain. Consequently, the Hiawatha Seam in the mine area dips 2° to 3° to the northwest (*27*).

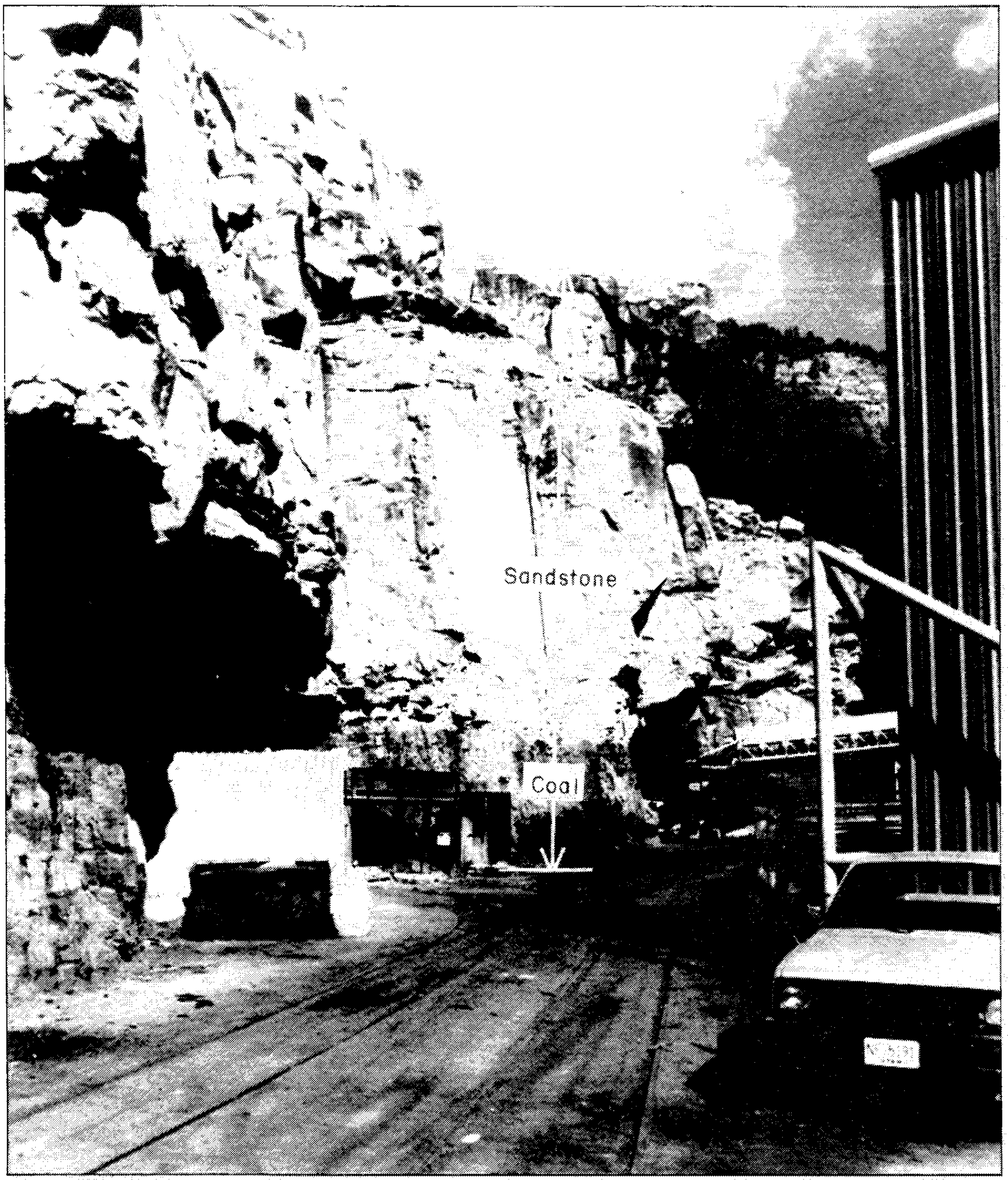
The principal structural features affecting mining operations are two north-south-trending normal faults that transect the Wilberg Mine area, as shown in figure 16. The Deer Creek Fault, which bounds the mine on the east, has zones of fractured, warped strata to about 30 m (100 ft) on both sides. The Pleasant Valley Fault, approximately 1,520 m (5,000 ft) west of the Deer Creek Fault in the mine area, decreases progressively in offset to a termination point at the southern end of the area (*27*). The USBM surface subsidence studies of several Deer Creek Mine longwall panels directly overlying the Wilberg longwall area (figure 17) indicated that the magnitude of subsidence decreased beyond the Deer Creek Fault (*32-33*), indicating minimal load "bridging" or transfer across the faults.

Figure 12



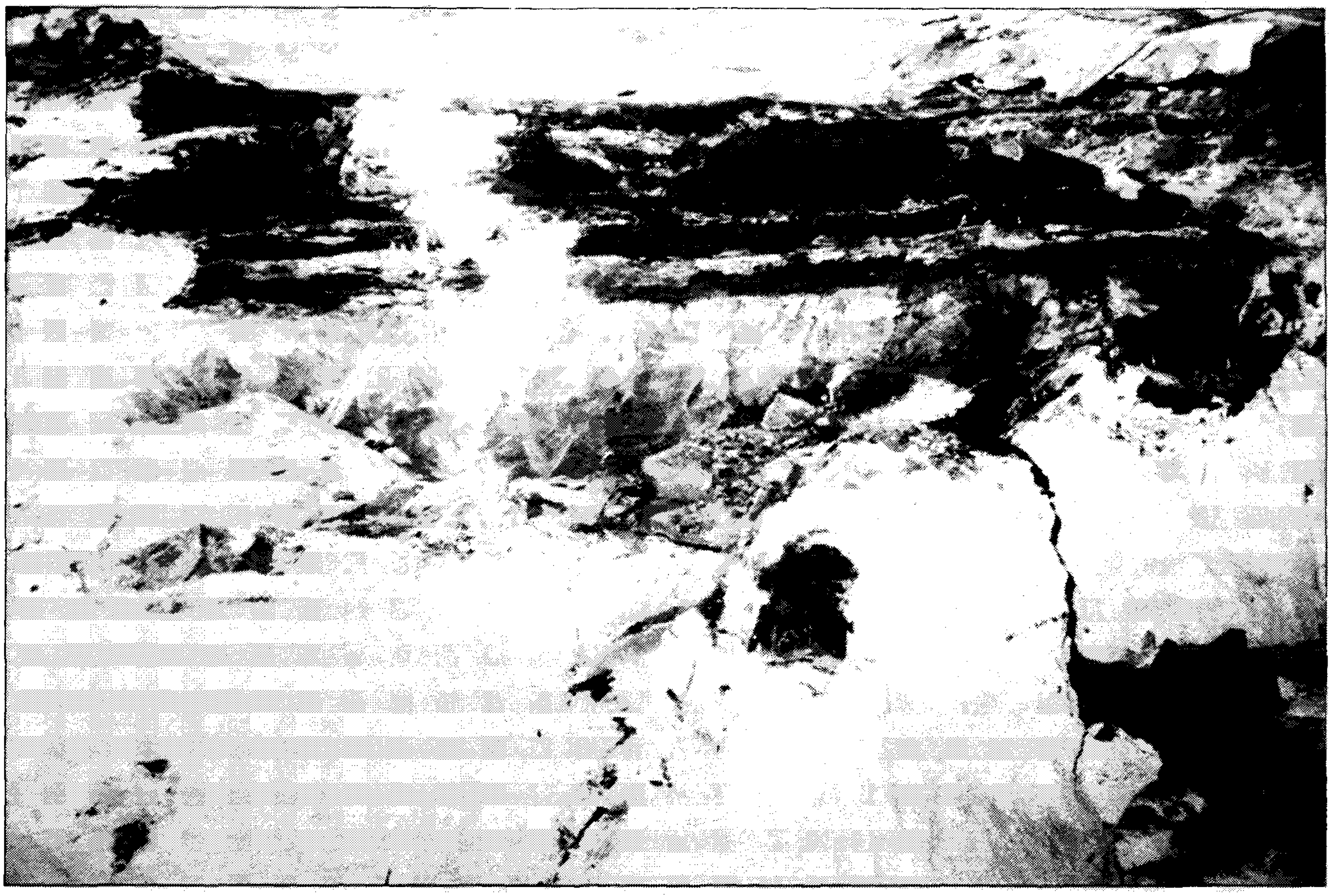
Exposure of Star Point Sandstone at Cottonwood (formerly Wilberg) Mine loading facility. Upper portion of Star Point forms white cliff above rectangular loading bin. Mine portals in Hiawatha Seam (at level of small cubical building above concrete silo) mark top of this massive sandstone.

Figure 13



Outcrop of Hiawatha Seam at original Wilberg Mine portals. Thick fluvial channel sandstone in lower portion of interburden between Hiawatha and Blind Canyon Seams forms cliff above portals.

Figure 14



Channel sandstone roof in Cottonwood (formerly Wilberg) Mine. Channel sandstone dike cutting upper portion of seam and rock split was probably formed by sand infilling of crack in coal-forming peat and underlying mud.

WILBERG MINE OPERATING HISTORY

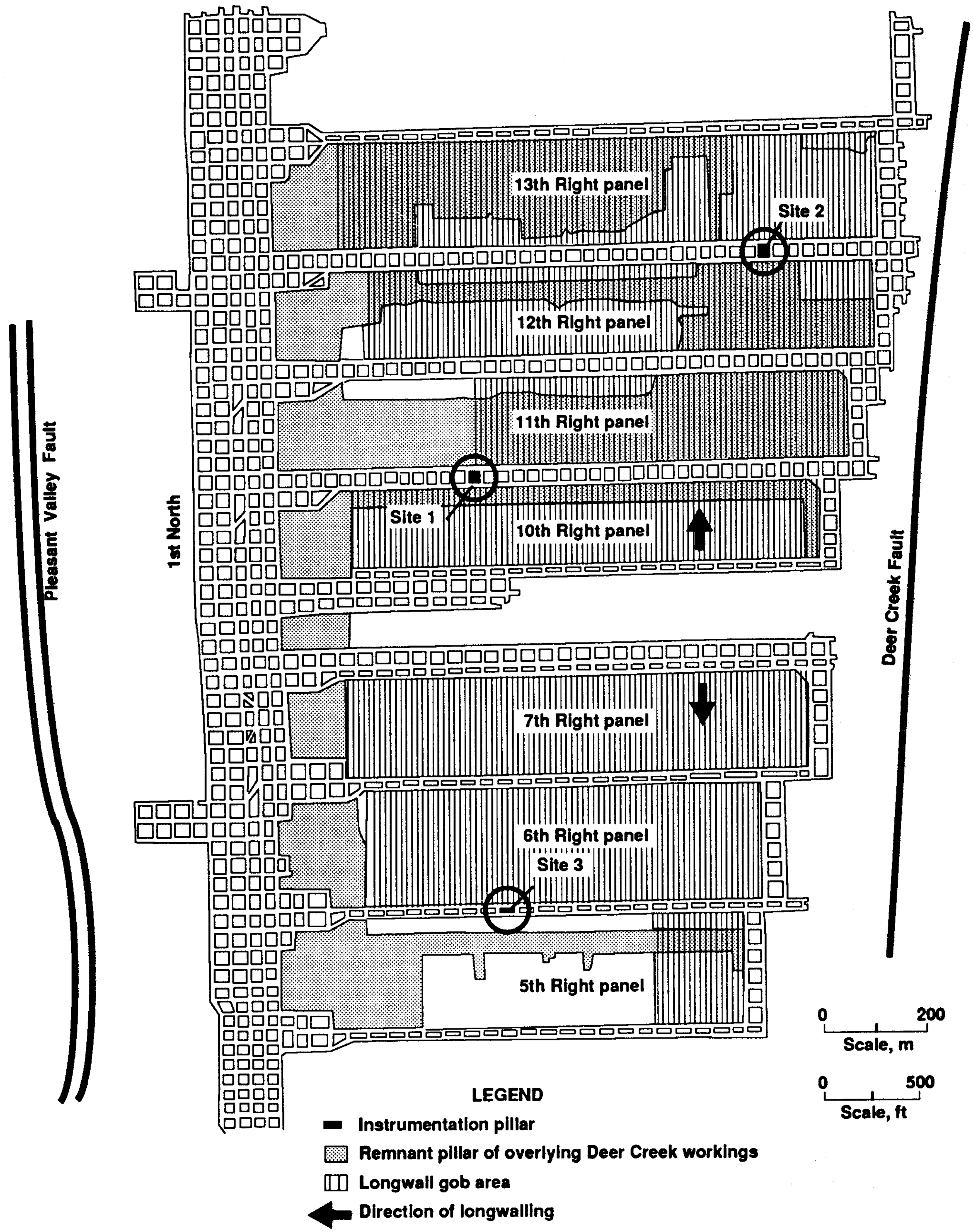
Mining in the Wilberg Mine area began in 1898 and continued intermittently until 1936 in several small "wagon mine" operations, the Fox, Straight Canyon, Castle Valley, Crow, and Reed mines (*23*). In 1936, the Wilberg family acquired these small properties and consolidated them into the Wilberg Mine, which the family operated at small production fairly continuously until 1967. Peabody Coal Co. acquired and reopened the mine in 1968 (*35*). In 1976, the Utah Power and Light Co. (UP&L) acquired most of the East Mountain coal properties, including the Deer Creek and Wilberg Mines (*27*). The mines were operated for UP&L by American Coal until 1978, when the contract mining rights were acquired by Emery Mining Corp., a subsidiary of Savage Brothers Transportation Co.

Longwall operations were begun in the Deer Creek Mine in May 1979 and in the Wilberg Mine in August 1980. Following the successful development and extraction of three Deer Creek panels (figure 17), all but the first of which utilized two-entry gate roads with 9.1-m (30-ft) wide chain pillars (*36*), longwall operations in the Wilberg Mine study area commenced in April 1982. The 1st North submain entries, accessing the study longwall area, and the first sets of gate road entries, 9th through 12th Right, were developed from 1979 to 1982 in an area bounded on the east by the Deer Creek Fault, immediately east of the panel starting rooms, and on the west by the Pleasant Valley Fault, approximately 122 m (400 ft) west of the 1st North submains (figure 16). Except for gate roads adjacent to barrier pillars, where a third entry was utilized as a bleeder entry, all gate roads were developed with a two-entry configuration. The Wilberg panels were designed and oriented so that the Wilberg gate roads would generally be overlain by mined-out Deer Creek panels, and the chain pillars of the Deer Creek gate roads would overlie the approximate centerlines of the Wilberg panels, as shown in figure 17. The first set of Wilberg study area panels was consecutively extracted northward, beginning with 10th Right (figures 16 and 17). This panel utilized 10th Right, which had 22.8-m (75-ft) wide pillars, for the headgate, and a tailgate with 9.1-m (30-ft) wide pillars, and it was

Figure 15

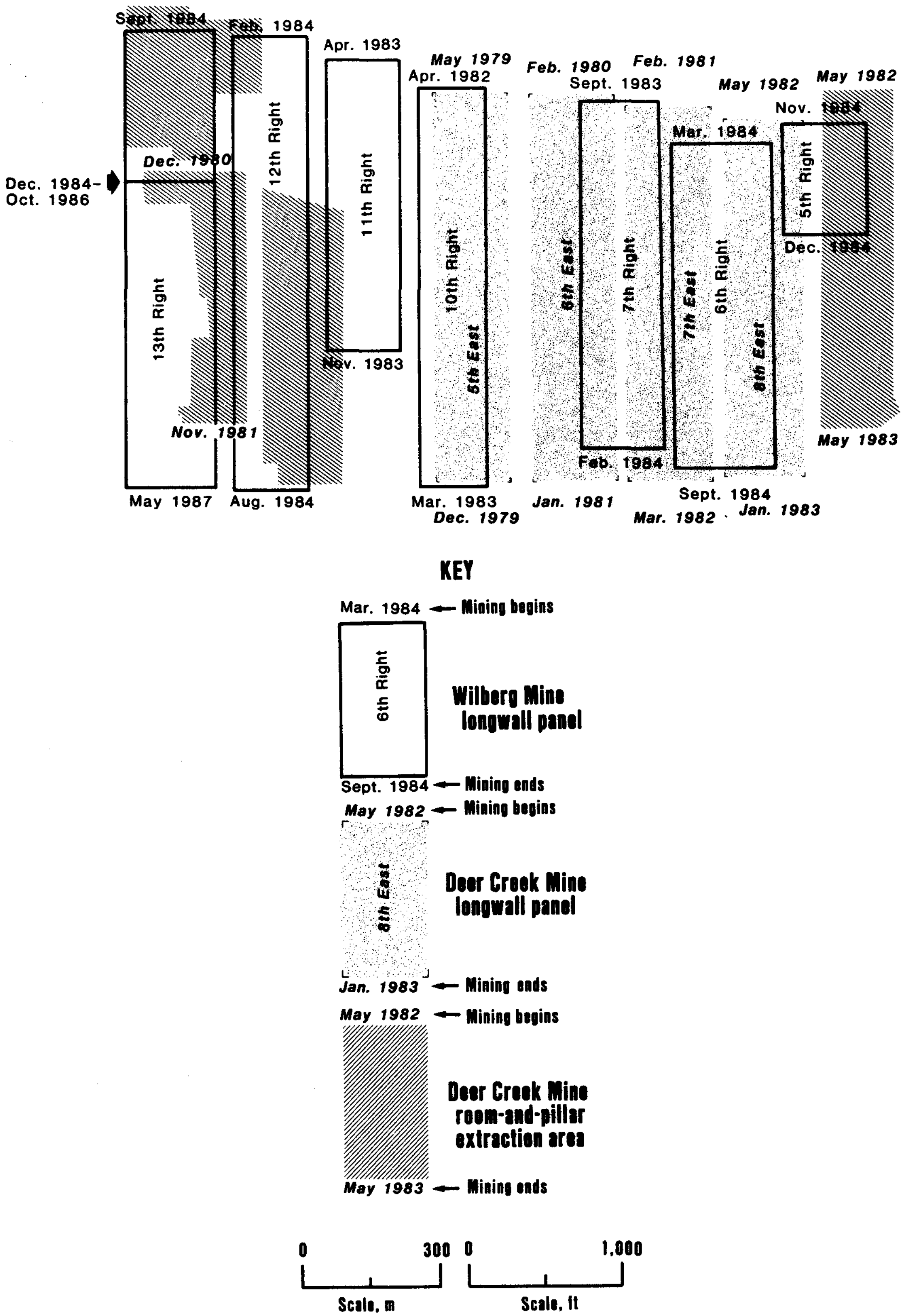
Overburden strata of Wilberg Mine. Hiawatha Seam is located at level of large conifers in foreground. Blackhawk Formation makes up slopes extending upward to base of high cliff formed by Castlegate Sandstone. Wooded ridge top above cliff is composed of lower Price River Formation sediments.

Figure 16



Study area longwall panels at Wilberg Mine.

Figure 17



Relative positions and extraction sequence of Wilberg Mine study area panels and overlying Deer Creek Mine workings.

successfully extracted to the planned end barrier with few significant ground-related problems. Mining of the fourth Deer Creek Mine panel, overlying the future 6th Right and 5th Right panels (figure 17), was completed during the 10th Right panel retreat. The 11th Right and 12th Right panels were developed with 24.3-m (80-ft) wide pillars in both the headgates and tailgates. When a fluvial channel scour was encountered during 11th Right extraction, mining was halted at about 74% of the planned panel length. The 12th Right panel, however, was completely extracted to the planned end barrier. Violent tailgate pillar bumps occurred during the retreat of both panels. The 13th Right headgate pillars were developed 9.1 m (30 ft) wide to alleviate the violent bump conditions experienced in the preceding two panels. This panel was mined without significant bump occurrences until the December 1984 mine fire forced panel closure at approximately 182 m (600 ft) retreat distance.

Beginning in September 1983, a second set of panels was consecutively extracted southward from the 106-m (350-ft) barrier separating the two panel sets (figures 16 and 17). The 7th to 5th Right set, therefore, was mined concurrently with 11th to 13th Right. In order to avoid or alleviate anticipated bump-prone conditions arising from a broad sandstone channel in the immediate roof throughout much of the longwall area, the 7th to 5th Right two-entry gate roads were developed with 9.1-m (30-ft) wide yield pillars. The 7th and 6th Right panels were extracted nearly to the planned end barriers despite gradually worsening gate road roof conditions. During 6th Right panel mining, roof and floor conditions in the 6th Right gate road became so severe that, despite spot rebolting and solid wood cribbing along the full extent of the entry, about 76 to 92 m (250 to 300 ft) of the tailgate entry had already been rendered impassible by a 15-m (50-ft) long roof fall and about 61 to 76 m (200 to 250 ft) of roof sag and floor heave when the 5th Right panel was mined (*37*). Extraction of this panel was terminated by the fire and subsequent mine closure at approximately 216 m (710 ft) retreat distance in December 1984.

USBM FIELD STUDY

Beginning in late 1983 and continuing through 1984, the USBM conducted gate-road pillar design investigations in three study sites located respectively in the 10th, 12th, and 6th Right entries (figure 16). One pillar in each of these gate roads was instrumented with USBM-fabricated borehole pressure cells (BPC's) oriented to monitor vertical and horizontal pressures. Because the primary data were obtained from the vertical cells, only these BPC's are shown in figures 18 to 20. The instrumented pillars in sites 1 and 2 were 22.8 m (75 ft) and 24.4 m (80 ft) wide, respectively, and the developed width of the site 3 pillar was 9.1 m (30 ft). Vertical pressure profiles for each pillar at different stages of first- and second-panel extraction are shown in figures 21 to 23. In these figures (and throughout this report), a negative (-) face position indicates the longwall face was inby the instrument location, and a positive (+) face position indicates that the face had progressed outby the instruments.

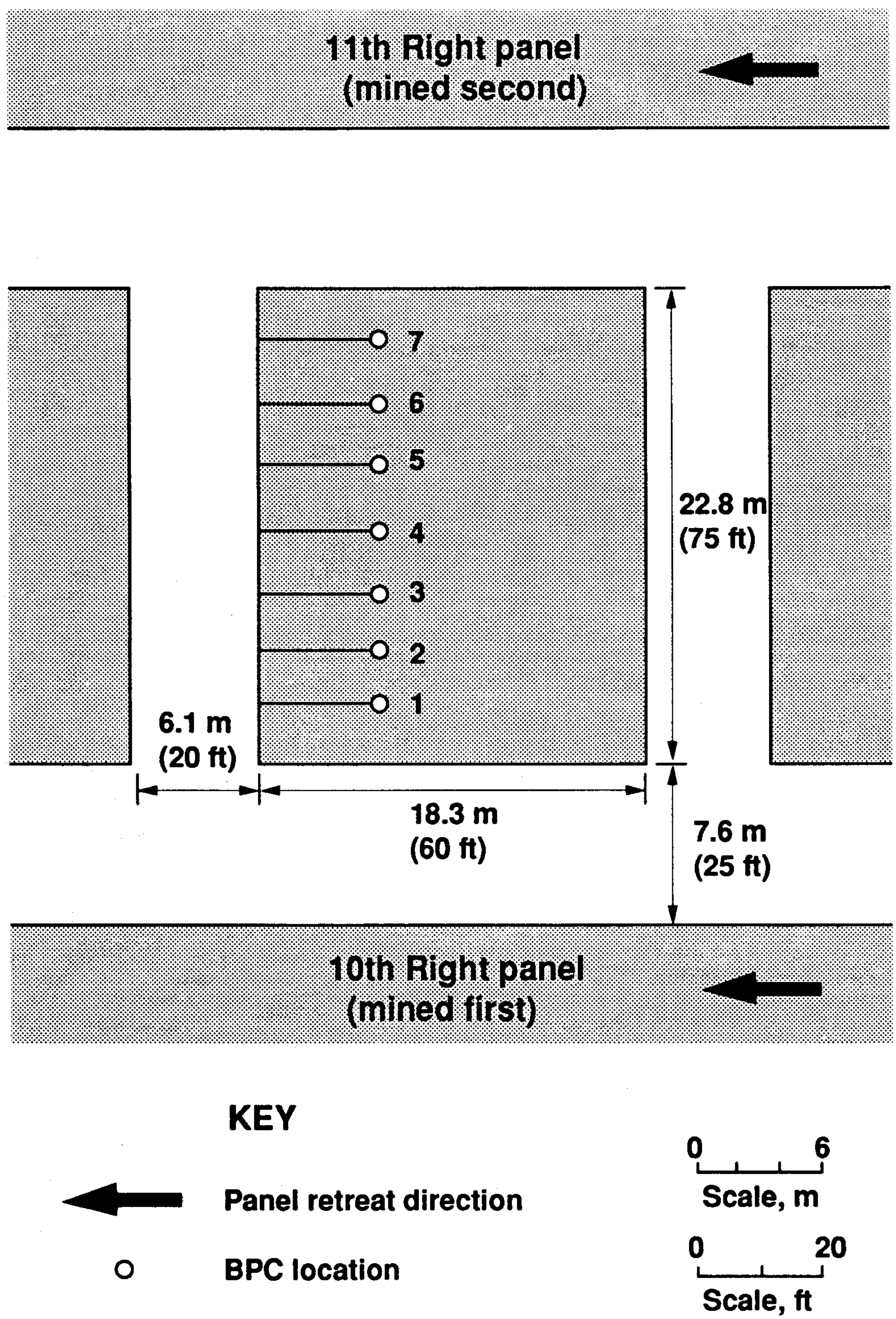
Site 1

The 22.8- by 18.3-m (75- by 60-ft) pillar was instrumented with seven BPC's (figure 18) when the 10th Right (first) panel face was approximately 305 m (1,000 ft) inby the site. Pressures were monitored through the rest of 10th Right panel retreat. Because of BPC failures and instrumentation damage during mining, only 10th Right panel extraction could be monitored; no data were obtained for 11th Right (second) panel extraction. When the 10th Right panel face was 119 m (390 ft) inby the site, pressures on the 11th Right panel side of the pillar exceeded those on the 10th Right panel side (figure 21), probably because of loading attributable to a large Deer Creek Mine barrier pillar and remnant panel pillars overlying most of the width of the 11th Right panel and only a small portion of the 10th Right panel (figures 16 and 17). When the face drew even with the site (0 face position), pressures on the 10th Right panel side increased because of front and side abutment loading from panel extraction, but remained less than pressures on the 11th Right panel side. By the time the face was 61 m (200 ft) outby the site, the pillar was subjected to nearly full side abutment loading, and pressure on the 10th Right panel side was about 25% greater than on the 11th Right panel side. As 10th Right panel extraction continued farther outby, recorded pillar pressures gradually decreased until data collection was terminated.

Site 2

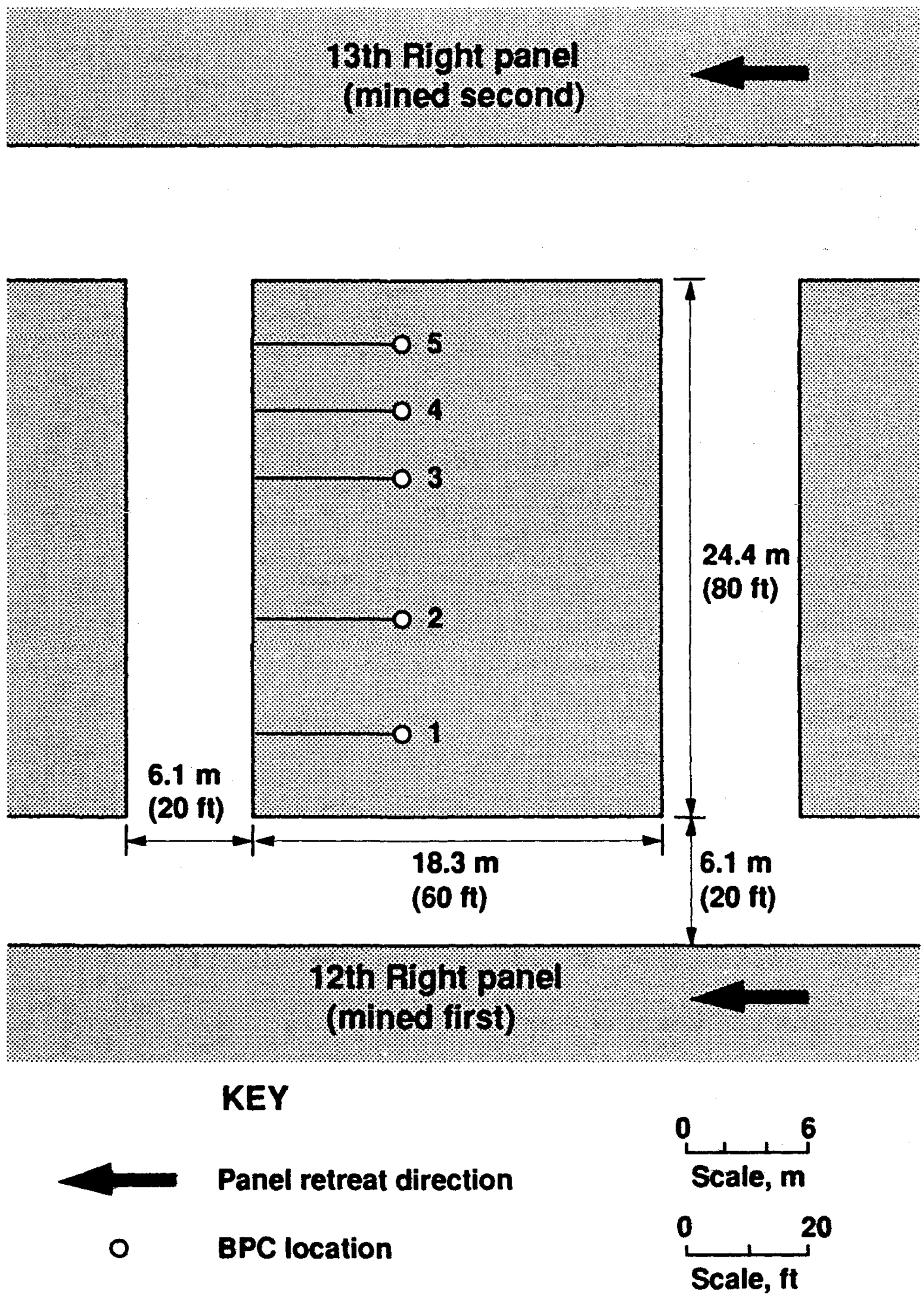
Five BPC's were installed in the 24.4- by 18.3-m (80- by 60-ft) pillar (figure 19) when the face was approximately 152 m (500 ft) inby the site. As shown in figures 16 and 17, the edge of a remnant Deer Creek Mine pillar overlies the 12th Right panel near the instrument site. Measured pressures, however, did not indicate severe loading attributable to this overlying pillar, probably owing to destressing from the mined-out area overlying the 13th Right panel and yielding at the edge of the remnant pillar. When monitoring was initiated, pressures across the instrumented pillar were nearly uniform, as shown in figure 22. As 12th Right (first) panel extraction progressed outby the site, pressures increased fairly evenly across the pillar, with side abutment loading reflected by a slight pressure shift to the 12th Right panel side of the pillar. When the 13th

Figure 18



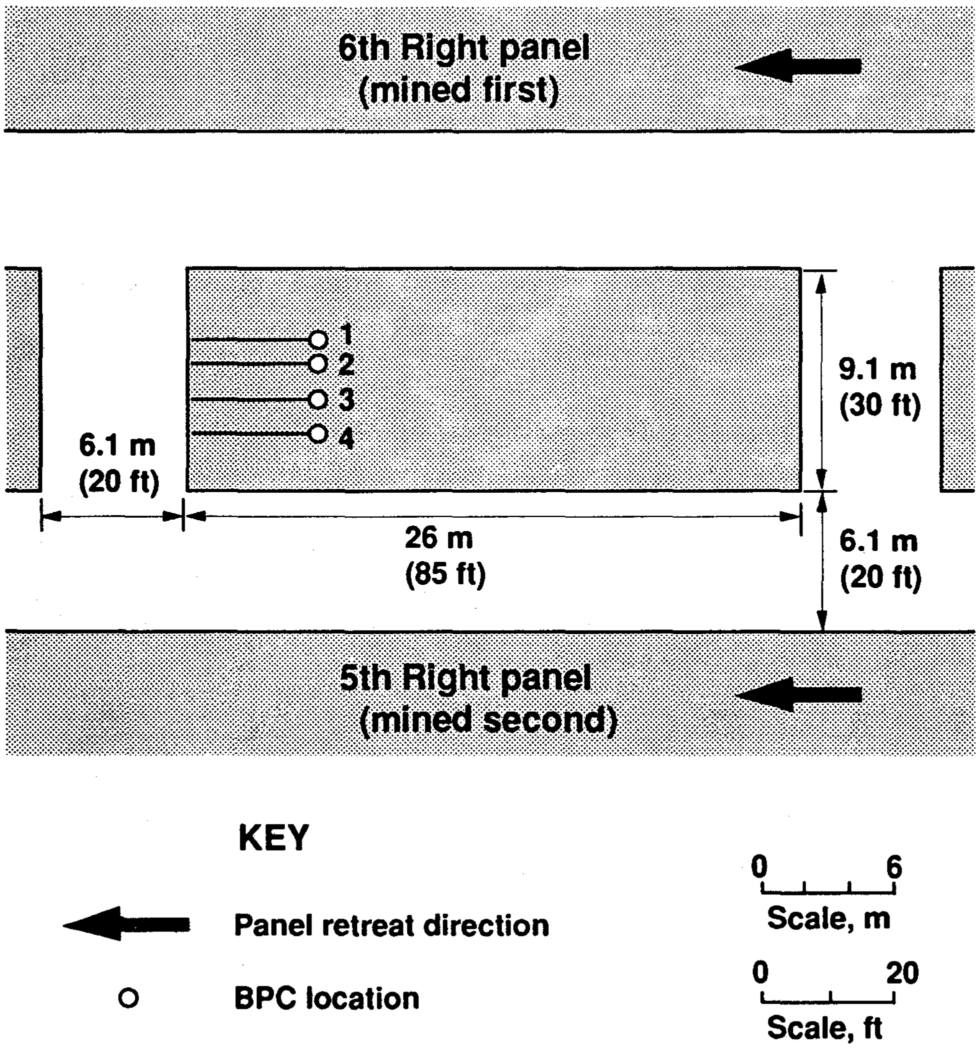
Site 1 instrumentation in 10th Right gate road, Wilberg Mine.

Figure 19



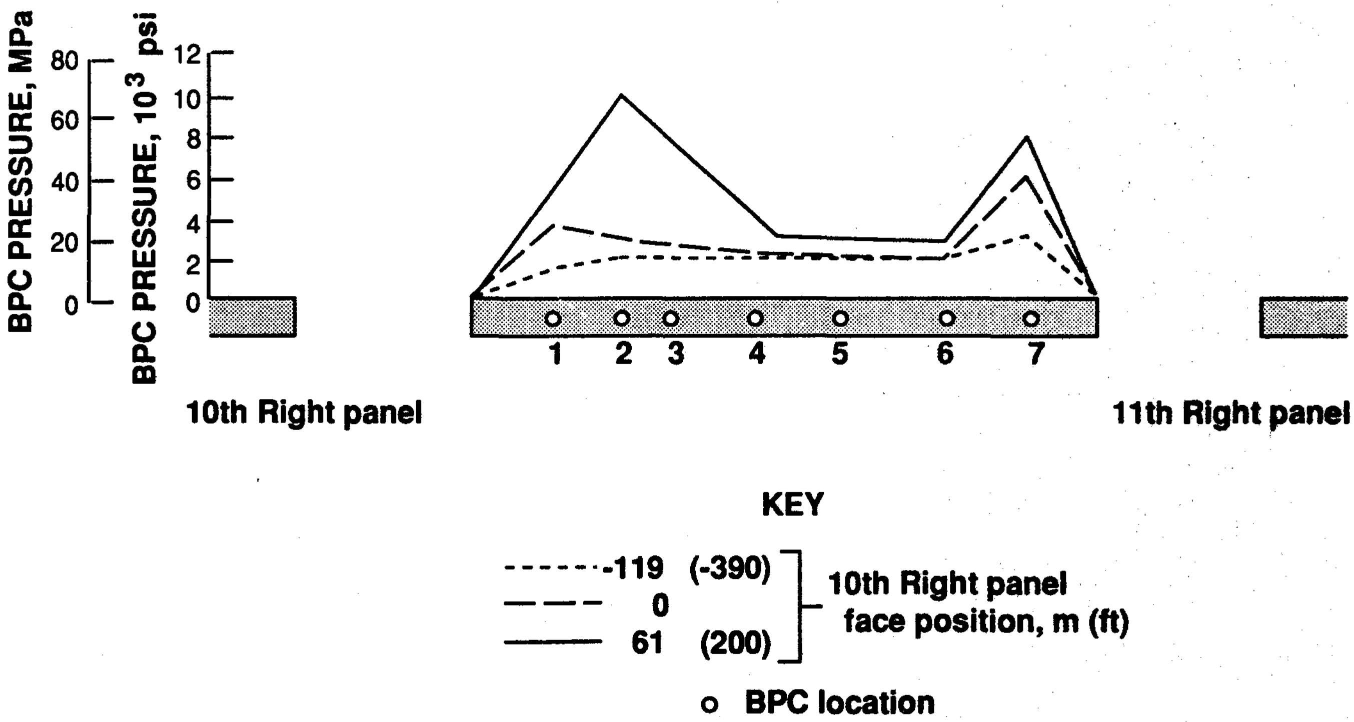
Site 2 instrumentation in 12th Right gate road, Wilberg Mine.

Figure 20



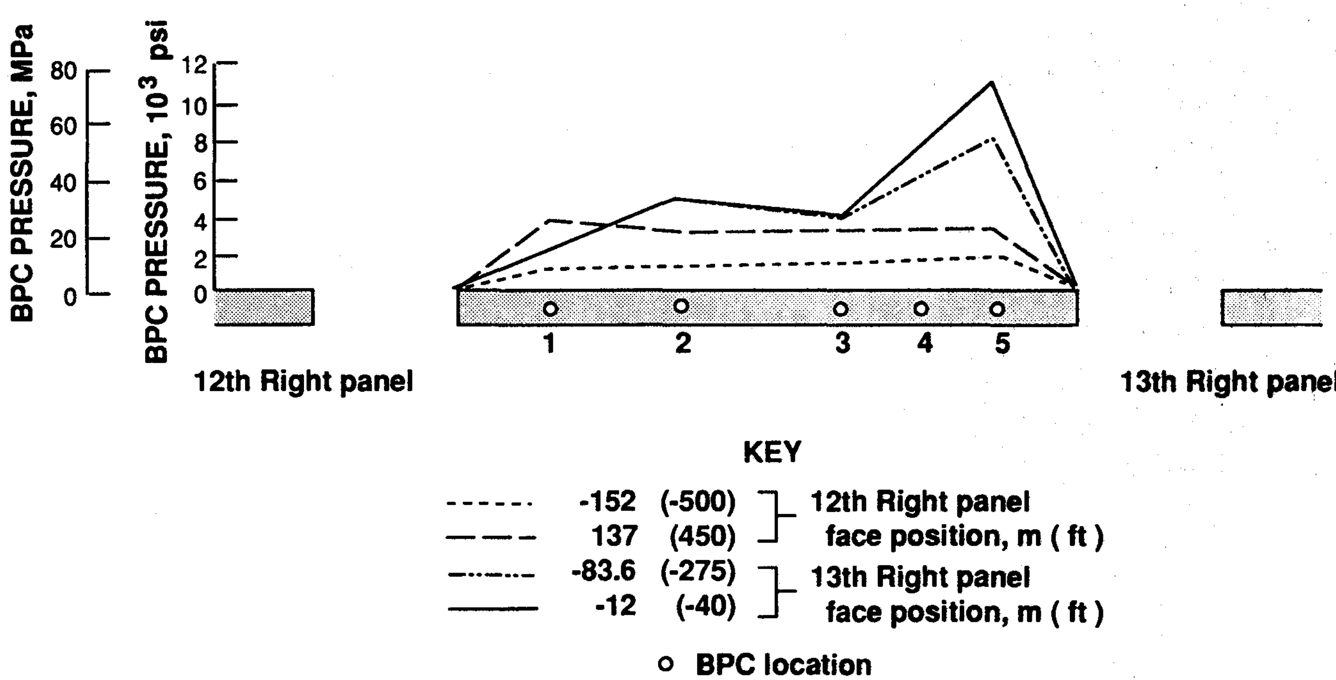
Site 3 instrumentation in 6th Right gate road, Wilberg Mine.

Figure 21



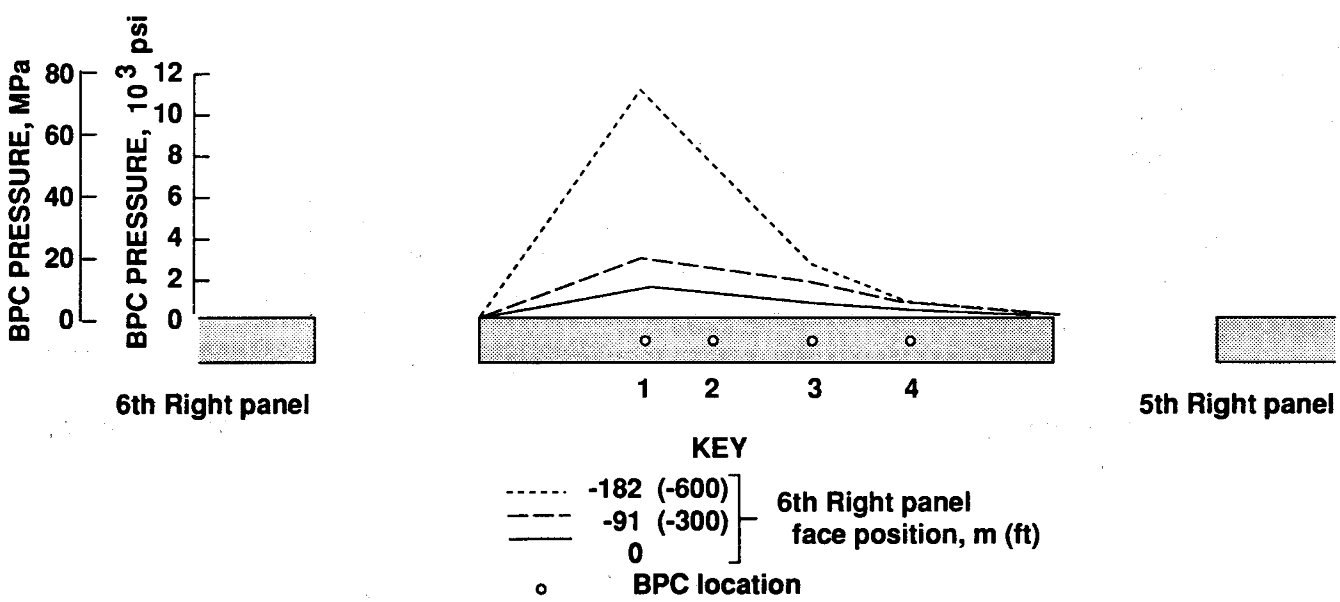
Vertical pressure profiles at site 1, Wilberg Mine.

Figure 22



Vertical pressure profiles at site 2, Wilberg Mine.

Figure 23



Vertical pressure profiles at site 3, Wilberg Mine.

Right (second) panel face was 83.6 m (275 ft) inby the site, pillar pressures had concentrated primarily onto the 13th Right panel side of the pillar because of second-panel front abutment loading. By the time the face had approached to within 12 m (40 ft) inby the site, full front abutment loading had dramatically increased pressures on the 13th Right panel side. Monitoring was discontinued in November 1984 when passage of the 13th Right face rendered the site inaccessible.

Site 3

Although the 6th Right gate road was developed with 9.1-m (30-ft) wide chain pillars and 6.1-m (20-ft) wide entries, pillar loading and consequent rib yielding and sloughage had reduced the site pillar width to 8.5 m (28 ft) (and increased the width of both entries to 6.4 m (21 ft)) at the time of instrument installation, when the 6th Right (first) panel face was at least 305 m (1,000 ft) inby the site. Considerable difficulties were encountered during borehole drilling and BPC emplacement because of high stresses and fracturing in the pillar. Four BPC's were installed across the pillar, as shown in figure 20. When the 6th Right panel face was 182 m (600 ft) inby the site, pressure on the 6th Right panel side of the pillar exceeded 69 MPa (10,000 psi) (figure 23). As the face progressed toward the site, pillar pressures steadily decreased, indicating destressing of the pillar in response to the front abutment load from the approaching face, either through pillar yielding or pillar "punching" into the floor, causing severe floor heave in the adjacent entries. At this time, approximately 50% of the pillar exhibited extensive fracturing, primarily at the entry-side ribs. As the 6th Right panel face approached and passed by the site (0 face position), pressures at the two-thirds of the pillar nearest the 5th Right (second) panel side dropped nearly to 0, and pressure at the remaining pillar core, approximately one-third of the pillar width from the 6th Right panel side, diminished to less than the initial installation pressure of 13.8 MPa (2,000 psi). Monitoring was discontinued in August 1984 when the 6th Right panel face was approximately 122 m (400 ft) outby the site. At this time, a combination of floor heave, roof falls, and rib sloughage had made the site nearly inaccessible, and the pressure recorders were removed from the site. No data were obtained during 5th Right (second) panel mining.

Wilberg Mine Conclusions

Loading of the site 1 pillar was much greater than loading of the site 2 pillar, although both pillars were approximately the same size, because the site 1 pillar was located beneath an overlying remnant pillar, whereas the site 2 pillar was located below a mined-out area of the overlying seam. The 10th Right gate road was essentially stable, at least throughout first-panel mining, and the 12th Right gate road maintained stability throughout both first- and second-panel mining (*38*).

Loading of the site 3 pillar was very high even before first-panel mining was initiated, and load-carrying capacity of the pillar was almost entirely lost during passage of the first face. During this period, loading in excess of the little remaining load-bearing capacity of the pillar was transferred to the 5th Right (second) panel. After mining of the first panel, the tailgate entry of the second panel was almost entirely supported by the massive secondary support, solid wood packs together with four- and nine-point cribs (*38*).

The site 1 and site 2 pillars were nearly at the center of a broad channel sandstone. When compressed between the sandstone immediate roof and the rigid Star Point Sandstone floor, these pillars could experience significant roof-floor confinement stresses. In the case of the site 1 pillar, subjected to concentrated loading from directly overlying large remnant pillars, this confinement most likely accounts for the high loads along the pillar ribs (figure 21) and the limited yielding demonstrated by the pillar. Moreover, confinement quite possibly contributed significantly to the bumps in the 10th and 11th Right gate roads during second-panel mining. In contrast, much of the 7th, 6th, and 5th Right panels, including the site 3 pillar, was immediately overlain by a roof-fall-prone overbank channel margin composed of interbedded thin sandstones and siltstone. Thus, the roof of the 6th Right gate entries could not withstand the deflection induced by compression of the yielding gate pillars. Moreover, the floor of the 6th Right gate road apparently consisted of thin, interbedded mudstone, siltstone, and clay, rather than a massive rigid sandstone, and proved unable to withstand loads transferred through the pillar and panel ribs, resulting in severe floor heave. The combination of these factors created a set of adverse conditions unique to this gate road, probably accounting for its poor performance in contrast to the demonstrated success of two-entry yield-pillar gate roads in previous and subsequent longwall panels in the mine.

LONGWALL MINING FOLLOWING STUDY

Following the Wilberg Mine fire, UP&L assumed operation of the mine from Emery Mining Corp. Longwall mining was initiated in the developed portion of the Wilberg that could be reopened following postfire recovery. UP&L instituted firm policies of having two panels fully developed before extracting the next available panel, i.e., having a panel in waiting, if practical. The 13th Right panel was restarted in late September 1986 and completed in late March 1987. Two additional panels, 4th Right and 3rd Right, were extracted in this section of the mine from September 1987 to January 1988 (*35*). According to mine personnel, no major ground control problems were encountered during the mining of these panels.

UP&L began development of the Cottonwood Mine in July 1985, approximately 7 months after the fire, in the large portion of the Wilberg reserves allocated to the Cottonwood Mine (*35*). Because the mine was not allowed to use two-entry gate roads at the time of development, a three-entry design using two yield pillars, both 9.1 m (30 ft) wide, was utilized in order to reduce bump potential. Tailgate cantilevering of the channel sandstone roof from the tailgate panel rib over the width of the gate road was believed responsible for excessive tailgate entry loading and bumps along almost half the face nearest the tailgate. The yielding of the tailgate pillars is not believed to have contributed to the face bumping. More recently, the current operator, Energy West Mining Co., a subsidiary of PacifiCorp (formerly UP&L), reintroduced two-entry yield-pillar gate roads, alleviating many of these problems. Mining with this gate road design currently continues, with only minor apparent gate stability problems experienced along the overbank channel sandstone margins (*37*). In the "post-Wilberg" Cottonwood workings, at least six three-entry and five two-entry yield-pillar gate road systems have been employed as tailgate roads in second-panel longwall mining.

STAR POINT NO. 2 MINE

LOCATION

The Star Point No. 2 Mine is located in the northern portion of Gentry Mountain, a projection of the eastern escarpment of the Wasatch Plateau, adjacent to the small town of Wattis, the terminus of Utah Highway 50 (figure 3).

GEOLOGIC SETTING

Stratigraphic Units

In ascending order, the principal stratigraphic units in the Star Point No. 2 Mine area are the Mancos Shale, the Star Point Sandstone, the Blackhawk Formation, the Castlegate Sandstone (figure 24), the Price River Formation, and the North Horn Formation (figure 25) (*24*). The units that probably most directly affect Star Point No. 2 mining operations are the Hiawatha, Third Bed, and Wattis coal seams, the interburden intervals separating them, the portion of the Blackhawk Formation immediately overlying the Wattis Seam, and the units that make up the overburden above the coal seams, particularly the Castlegate Sandstone.

Star Point No. 2 Coal Seams and Interburden

The coal seams that have historically been mined in the Star Point No. 2 Mine area are, in ascending order, the Hiawatha Seam, the Third Bed (referred to in some

Figure 24

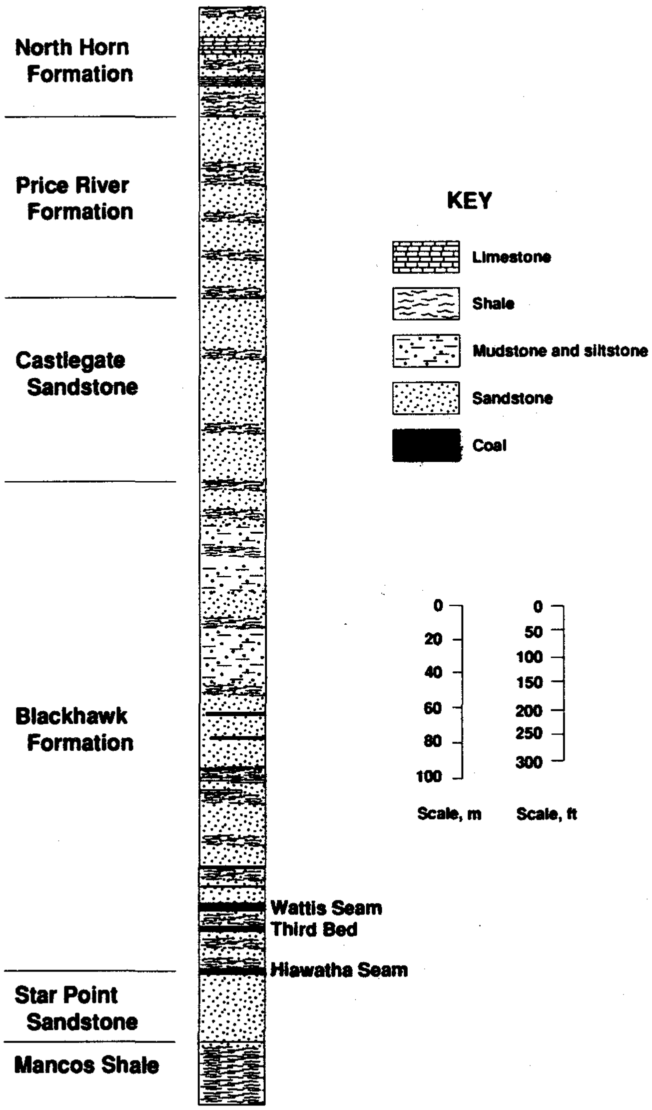
Strata at Star Point No. 2 Mine. From canyon bottom (at lower left foreground) upward, Mancos Shale makes up wooded slope extending up to foot of low cliff and ledge formed by Star Point Sandstone (to left of mine road in foreground). Mine portals in Wattis Seam (in Blackhawk Formation) are at level of bench cut across face of high ridge in background. Blackhawk Formation extends upward to foot of Castlegate Sandstone cliff at crest of ridge.

literature as the Middle Seam), and the Wattis Seam. The Hiawatha Seam immediately overlies the Star Point Sandstone and is separated from the Third Bed by 15.2 to 21.3 m (50 to 70 ft) of channel sandstone, overbank mudstone and siltstone, and minor coal seams. The Third Bed, which is 2.4 to 3.6 m (8 to 12 ft) thick in the Star Point No. 2 longwall area, is separated from the Wattis Seam by 11 to 17 m (35 to 55 ft) of channel sandstones in areas of thick interburden or by overbank mudstones and siltstones in areas of thinner interburden (*29*). The Wattis Seam, 2.4 to 3.6 m (8 to 12 ft) thick in the mine area, has a well-defined structure with two sets of persistent cleats. Since the Hiawatha Seam is not of economically minable thickness in the Star Point No. 2 longwall area, mining operations are restricted to the Wattis Seam and Third Bed.

Strata Immediately Overlying Wattis Seam

The Blackhawk Formation strata lying within about 3 m (10 ft) of the upper surface of the Wattis Seam form the immediate roof of the Wattis workings of the Star Point No. 2 Mine. The strata are composed primarily of weak mudstone and siltstone immediately overlying the seam, with rock strength gradually increasing into the roof (*29*). Numerous channel sandstones transect the roof throughout the Wattis Seam workings. When a channel sandstone contacts the seam or lies relatively close to the seam top, the resulting confinement delays near-rib yielding, as evidenced by areas of minimal rib sloughage directly beneath the channel sandstones. When a channel sandstone extends across the entire width of a gate road, loading bridges across gate entries onto pillar cores or solid panel

Figure 25



Stratigraphic column, Star Point No. 2 Mine.

blocks (*29*). Areas in which overbank mudstone and/or siltstone channel margins form the immediate roof or in which a channel sandstone bottom lies within 1.5 to 2.1 m (5 to 7 ft) of the mine roof have the most difficult-to-support roof and have been closely correlated with roof falls or longwall tailgate support problems (figure 26). The mudstone roof in these areas is highly jointed and is secured with bolts anchored in the overlying channel sandstone in areas where this condition has been identified by examination of bolt hole cuttings (*39*).

Overburden Strata

The undifferentiated upper two-thirds of the Blackhawk Formation, the Castlegate Sandstone and Price River Formation, the North Horn Formation, and the Flagstaff Limestone, together ranging from 334 to 486 m (1,100 to 1,600 ft) thick, averaging about 365 m (1,200 ft), make up the main overburden above the Star Point No. 2 Wattis Seam workings. The Castlegate Sandstone is about 61 to 91 m (200 to 300 ft) thick in the mine area.

Structural Features

The strata in the Star Point No. 2 Mine area dip 2° to 3° southeastward; they are transected by the faulting common to the eastern Wasatch Plateau. The northward-trending Bear Canyon graben, which is approximately 760 m (2,500 ft) wide and includes numerous faults with significant displacement, bounds the western side of the longwall area. Also, a fault-dike system trending N. 80° W., infilled with intrusive material, forming dikes up to 2.4 m (8 ft) thick, transects the central part of the longwall area (*29*). Subsidence movement was recorded on this fault system when it was apparently reactivated by mining operations. Although these faults undoubtedly significantly affect ground conditions in immediately adjacent areas, their effect on ground stability and load transfer in the study area panels is difficult to determine.

STAR POINT NO. 2 MINE OPERATING HISTORY

The Wattis area mines were first opened in 1917 by the Lion Coal Co., which continued room-and-pillar operations until 1964, producing approximately 7.03 Mt (7.75 million U.S. short tons) in this period. The properties were acquired by Plateau Limited in 1967, then by Plateau Mining Co., a subsidiary of United Nuclear Corp., in 1971 (*24*). Getty Oil Co. acquired Plateau Mining during the late 1970's. The current owner, Cyprus-Plateau Mining Co. (CPMC), a subsidiary of Cyprus Minerals Corp., has operated the mine since 1986.

The operating history, extraction sequence, and gate road design performance of the Star Point No. 2 Wattis Seam longwall panels shown in figure 27 are described by Maleki (*29*):

> Longwall retreat was first initiated in 1984 from the 8th Left headgate, a three-entry system using two 50-ft by 80-ft pillars. During the [panel 1 in figure 27] retreat, roof, rib, and floor stability problems developed in the headgate entries, influencing access and ventilation. Excessive signs of loading and deformation were noted at the headgate, resulting in a number of roof falls immediately behind the face. Floor heave reached several feet behind the face, tilting cribs in the tailgate entry and reducing their effectiveness for tailgate

support. Pillars were loaded to their strength, contributing to roof, rib, and floor stability problems. During the retreat of the second (7th Left) and third (6th Left) panels [panel 2 and panel 3 in figure 27], ground conditions deteriorated as load transfer to panel boundaries increased. This increased pillar stresses and resulted in excessive roof, floor, and rib movements. CPMC increased the density of secondary support, adding extra cribs, combination bolts, truss bolts, etc. This helped ground conditions in the second panel, but ground squeeze continued in 6th Left, impacting crib support effectiveness. It became very difficult to maintain an open walkway through the tailgate entry [figure 28].

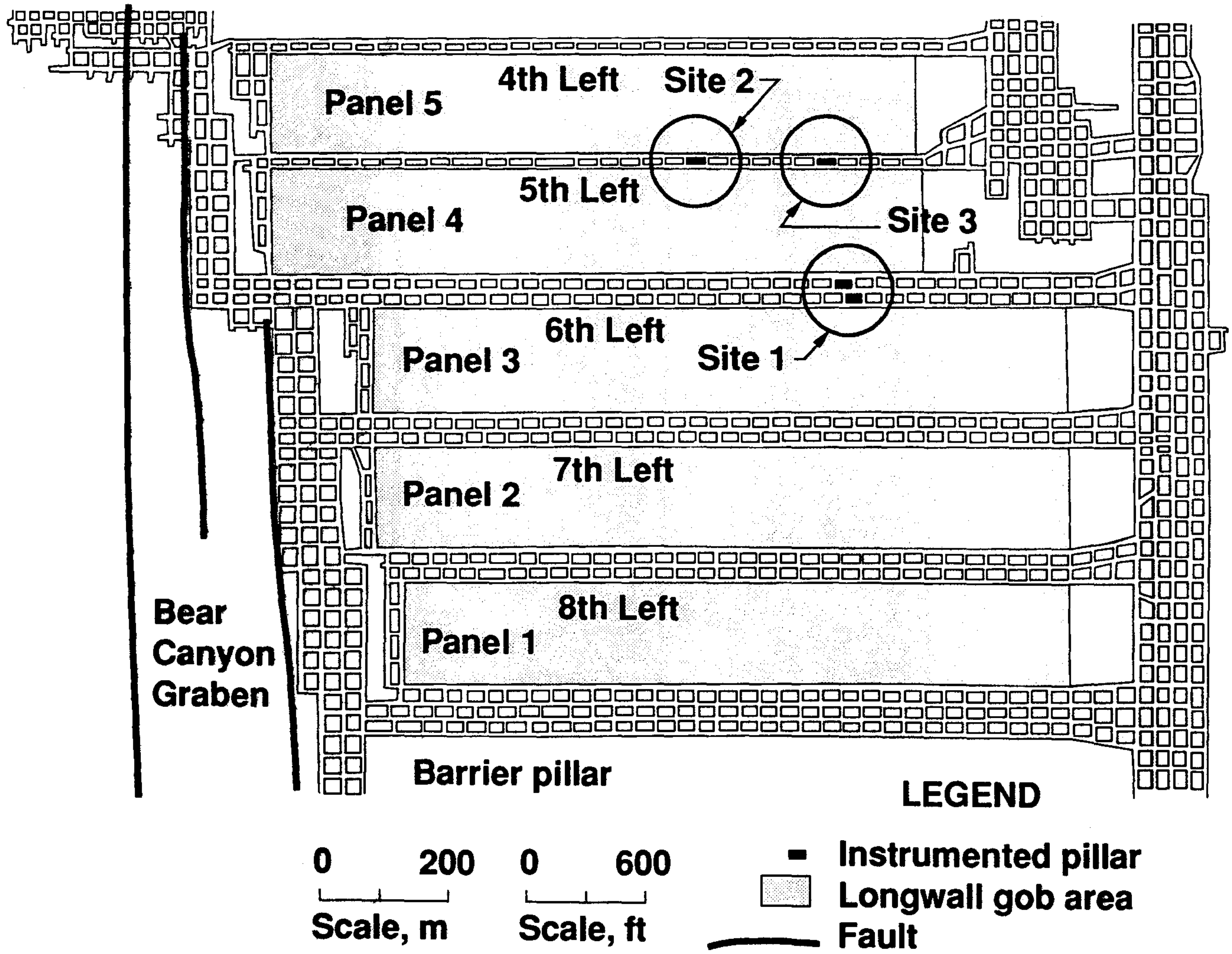
Gateroad conditions improved during the retreat of subsequent panels due to a switch to a two-entry yield pillar system and, possibly, improvements in cave conditions.

On a local scale, the geologic impact on roof stability became very evident during the retreat. Roof movements accelerated again where channels were within 0 to 7 ft from the roof. Supplementary support, such as 8-ft-long combination bolts and truss bolts, helped control roof movements and prevented roof falls. For extreme loading conditions, however, cribs and longer bolts were required to maintain an open and accessible walkway through the tailgate entry for the three-entry system. With yield pillars, 8-ft-long resin bolts and yieldable steel sets worked best [figure 29]. This support system is most compatible with pillar yielding and the associated strata deformation.

Figure 26

Channel margin roof in Star Point No. 2 Mine. About 2.5 m (8 ft) of interbedded siltstone-mudstone immediate roof has fallen, exposing bottom surface of overlying channel sandstone (dark area surrounding rectangular roof bolt plate).

Figure 27



Study area longwall panels at Star Point No. 2 Mine.

USBM FIELD STUDY *(40)*

Instrumentation

Beginning in late 1985 and continuing through 1987, the USBM investigated chain pillar and gate entry stability in three sites located in the 6th Left and 5th Left gate roads (figure 27). The instrumentation utilized for all three sites included BPC's oriented to measure vertical pressures, as well as stations to measure entry closure near the instrumented pillars, and roof and floor coring to provide supplemental site information. At site 1, located in the three-entry 6th Left gate road, two 15.2-m (50-ft) wide chain pillars and the tailgate rib of the fourth (5th Left) longwall panel block were instrumented as shown in figure 30. At sites 2 and 3, located in the two-entry 5th Left gate road, instrumentation was installed in one 9.1-m (30-ft) wide chain pillar and the tailgate rib of the fifth (4th Left) longwall panel at each site, as shown in figure 31, the instrumentation plan used for both sites.

Instrument Monitoring and Data Analysis

Six face positions were chosen to compare stability characteristics for each study site—five for first-panel mining and one for second-panel mining. Failure of (and inability to repressurize) the site 1 BPC's during extraction of panel 4 (figure 27) rendered data interpretation at face positions closer than 152 m (500 ft) inby the site location very difficult. These events thus necessitated the single face position, -152 m (-500 ft), for second-panel mining. The hydraulic tubing connecting the BPC's to the pressure recorders at sites 2 and 3 was severed when the panel 5 face was approximately 152 m (500 ft) inby site 2.

Figure 28

Sixth Left tailgate escapeway during panel 4 mining. USBM engineer wedged between cribs indicates narrow width of escape walkway.

Figure 29

Yieldable steel sets in 5th Left gate road. In subsequent gate roads, this expensive secondary support system was abandoned in favor of conventional wooden cribbing.

Because of adverse ground conditions at the site 2 crosscut, the tubing could not be reconnected, and site 2 BPC monitoring was discontinued. At site 3, however, the tubing was reconnected, and BPC monitoring was reinitiated and continued until the panel 5 face had approached to approximately 152 m (500 ft) inby the site. At this time, roof falls in the site 3 crosscut broke the tubing connections, terminating BPC data collection for the site. Vertical pressure profiles of the instrumented pillars and panel ribs for sites 1, 2, and 3 are shown respectively in figures 32, 33, and 34.

Site 1

As the panel 3 face approached the site, progressive rib failure occurred in pillar 1 (figure 32), with load transferred to pillar 2 and panel 4; pillar 2 experienced decreased rib loading, indicating narrowing of its confined core. When panel 3 was 152 m (500 ft) inby the site, panel 4 experienced only loads attributable to the nearby openings. By the time panel 3 had approached to 55 m (180 ft) inby the site, front abutment loading and loads transferred from pillars 1 and 2 created a yield-abutment zone in the first 6 m (20 ft) of panel 4. As the panel 3 face drew even with the site instrumentation (0 face position), decreasing load-bearing capacity of pillars 1 and 2 was reflected by significant increases in the abutment zone width and overall panel loading. Additional panel 4 side abutment loading, observed when the panel 3 face was 61 m (200 ft) outby site 1, showed continued deterioration of the load-bearing capacity of the gate pillars after the face had passed by the site.

In second-panel mining, when the panel 4 face was 152 m (500 ft) inby the site, the peak abutment load had shifted to about 9 m (30 ft) from the panel rib, with significant loading extending at least 6 m (20 ft) further into the panel. Although the gate pillars had mostly yielded, they probably retained sufficient residual strength to keep the intermediate or main roof intact, thus allowing most of the loading arising from panel 3 extraction to cantilever onto panel 4. The consequent extended high tailgate loading probably contributed to tailgate entry roof instability and related tailgate-side face problems experienced during panel 4 mining.

Site 2

The vertical BPC data shown in figure 33 indicate that the 9-m (30-ft) wide pillar began yielding during or soon after development. The pillar core, which appears confined to the central 3 m (10 ft) of the pillar, retained significant load-bearing capacity until the face of panel 4 drew even with the site instrumentation. As the panel continued outby, total pillar load was steadily reduced by progressive pillar failure and consequent load shifting onto panel 5.

The load sequence for panel 5 correlates with that of the gate pillar, although not as obviously as at site 1. When panel 4 was 37 m (120 ft) inby the site, the panel 5 peak abutment pressure was nearly at the panel rib. As the panel 4 face approached near and continued outby from the site, the panel 5 abutment load shifted about 3 m (10 ft) toward the panel center, with corresponding load redistributions further into the panel.

Site 3

The chain pillar at site 3 was not subjected to the high-magnitude loading observed at site 2, which may have been attributable to a channel sandstone in the roof near site 2. The greater relative stiffness of the sandstone may have concentrated abutment loading more effectively onto the chain pillar prior to panel 4 face retreat beyond the site.

As shown in figure 34, the gate pillar at site 3 had a well-defined pillar core, which did not significantly change in response to mining the adjacent panels 4 and 5. Although the BPC in pillar hole C failed shortly after the panel 4 face passed by the site, preventing further detailed analysis of pillar performance, the load trends of the remaining pillar BPC's indicated no significant change in the pillar-loading profile.

Since the loading geometry and magnitude of the pillar showed little change throughout site 3 monitoring, the increases in peak and total loads experienced by panel 5 were directly attributable to abutment loading from the extraction of panel 4. As the panel 4 face approached the site, peak and total loads steadily increased, and peak loading shifted deeper into panel 5. Once the face had passed by the site instrumentation, however, little significant load increase occurred, and the panel 5 load profile remained relatively constant.

Entry Closure (Roof-to-Floor Convergence) Monitoring

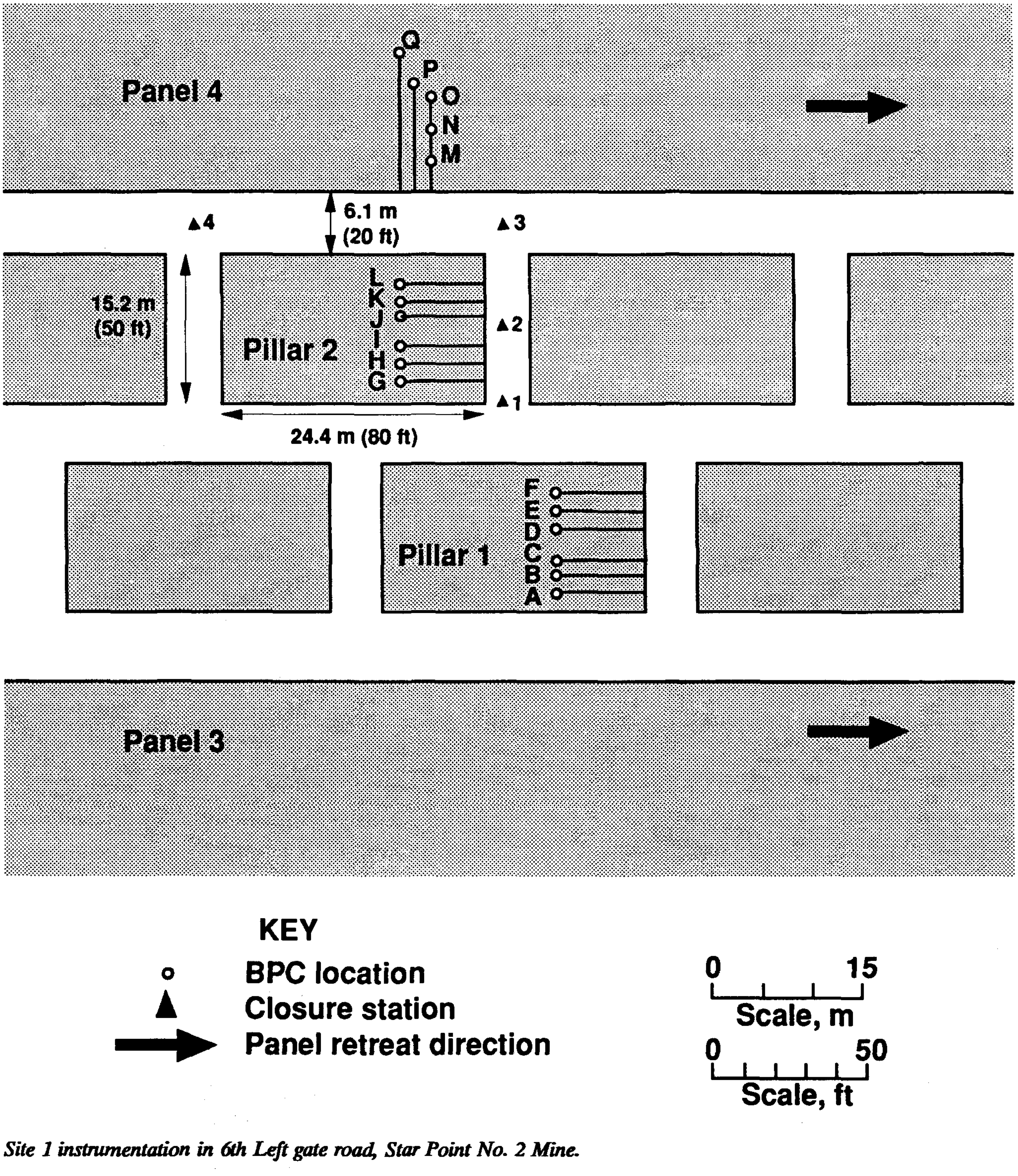
Coincident with the pillar stress monitoring, entry closure during first panel mining was measured to supplement BPC data for the three sites. At site 1, closure measurements were not initiated until the panel 3 face had approached and was even with the instrumentation. Because there were no closure data for face positions inby the instrument site (figure 35), the total closure curves for the existing site 1 data were projected back to a face position 152 m (500 ft) inby the site, which the BPC data indicated to be an average position for the onset of front abutment loading. Based on these projections of observed initial closure rates, the inferred maximum closure was approximately 20 cm (8 in) in the center entry, with 9 cm (3.5 in) of convergence in the tailgate crosscut.

Because closure measurements in the 5th Left gate road could be initiated well in advance of the approach of the panel 4 face, more complete closure records were obtained for sites 2 and 3, (figures 36-37). At both sites, the onset of front abutment loading began when the panel 4 face was approximately 152 to 183 m (500 to 600 ft) inby the site, with significant closure rate increases when the face was about 61 m (200 ft) inby and again when the face had approached within 30 m (100 ft) of the site. These data are corroborated by the pressure profiles for sites 2 and 3. When pillar pressures decreased and/or panel abutment loading shifted deeper into the rib, indicating pillar yielding and/or development of the yield zone along the panel rib, closure rates increased. Together, limited site 1 roof sag measurements, showing approximately 2.5 cm (1 in) maximum sag, and visual observations of roof and floor conditions during panel retreat indicated that floor heave accounted for most of the measured closure. No roof sag data were gathered at sites 2 and 3.

Results

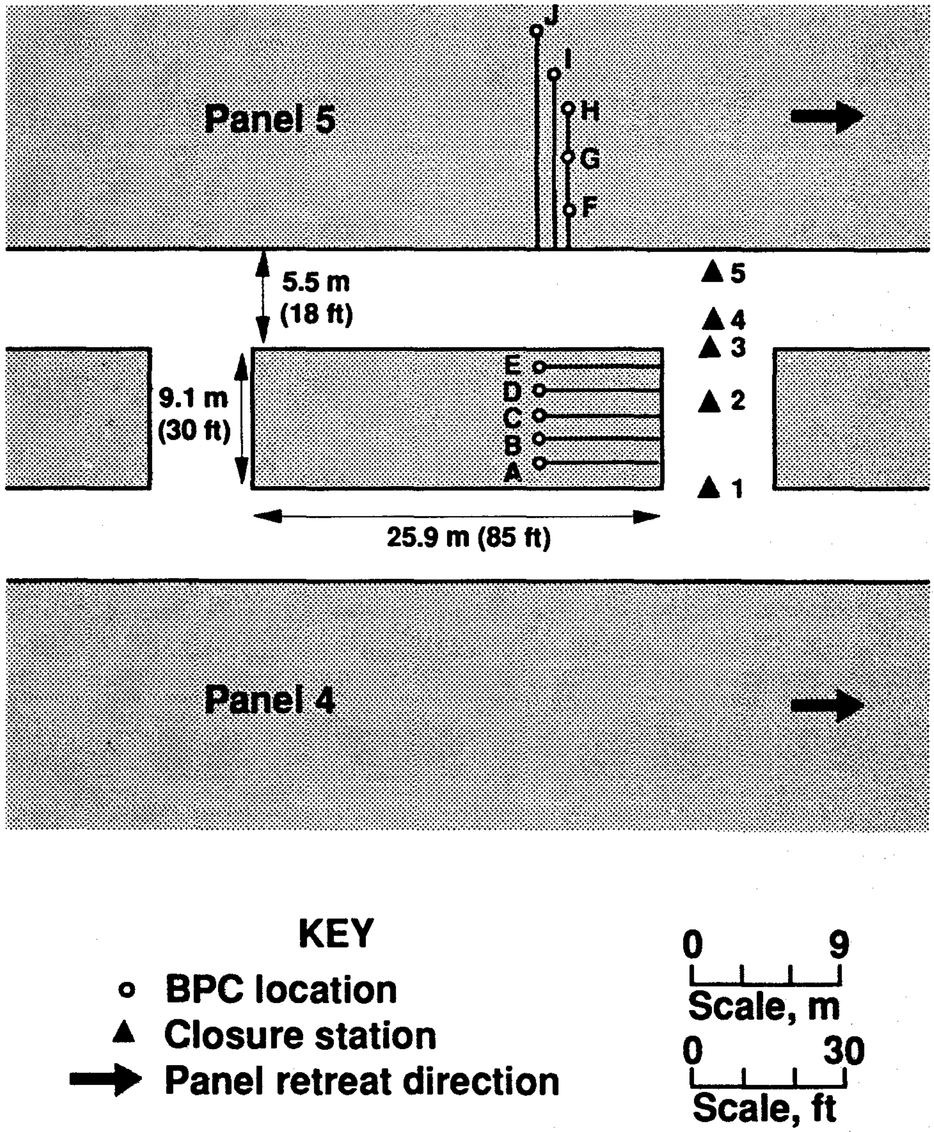
Significant roof instability and severe floor heave occurred in the three-entry 6th Left gate road throughout site 1 monitoring, particularly during panel 4 retreat. Secondary support timbers, cribs, and steel sets commonly crushed, buckled, or rolled out, as shown in figures 38 and 39, losing support capability. Moreover, serious roof problems occurred at the tailgate (6th Left) side of panel 4 during much of the panel retreat. In contrast, ground conditions observed in the two-entry 5th Left gate road (figure 40) remained relatively stable in both first and second panel mining during sites 2 and 3 monitoring. Although some roof falls occurred during panel 5 retreat, especially along channel margins, overall ground instability,

Figure 30



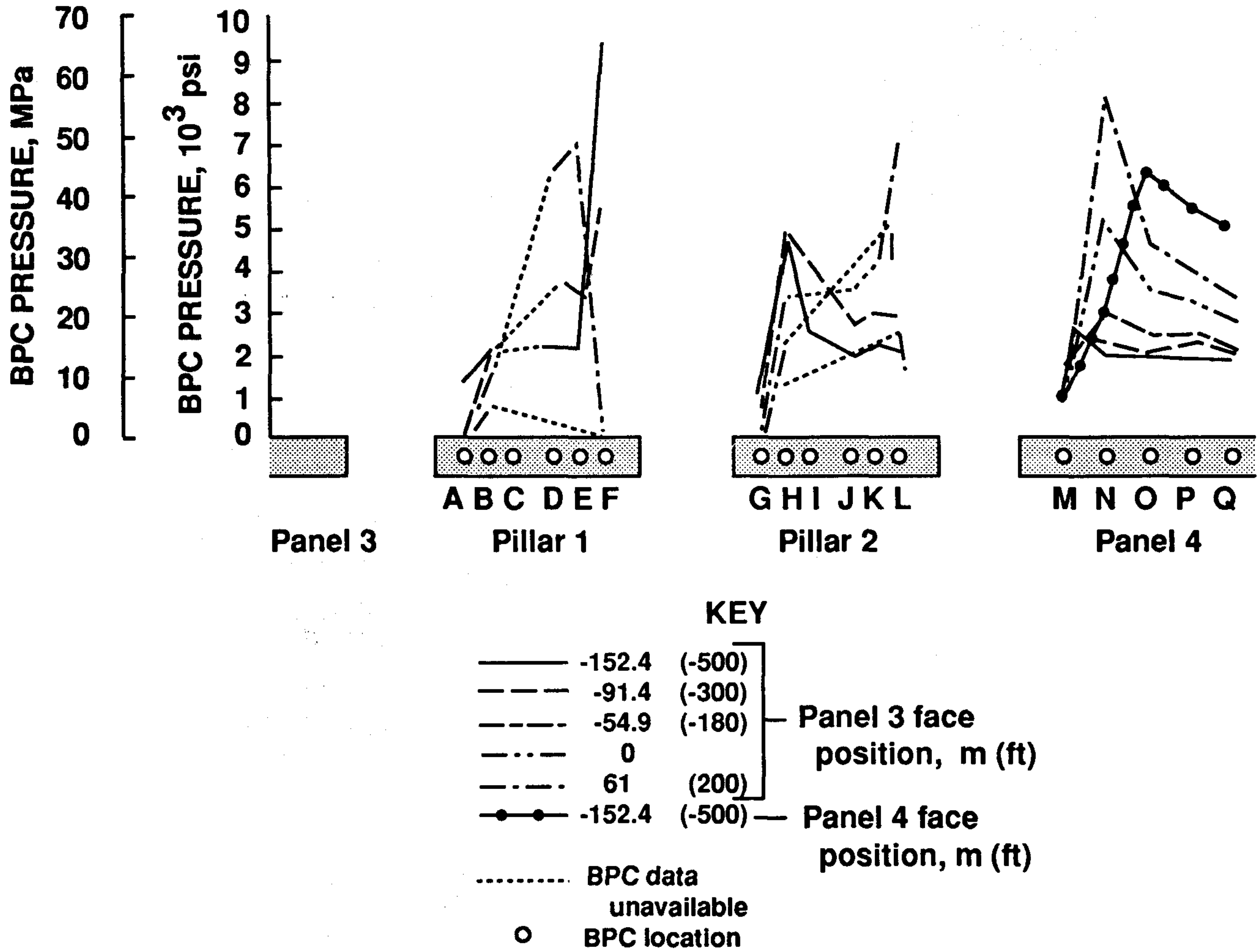
Site 1 instrumentation in 6th Left gate road, Star Point No. 2 Mine.

Figure 31



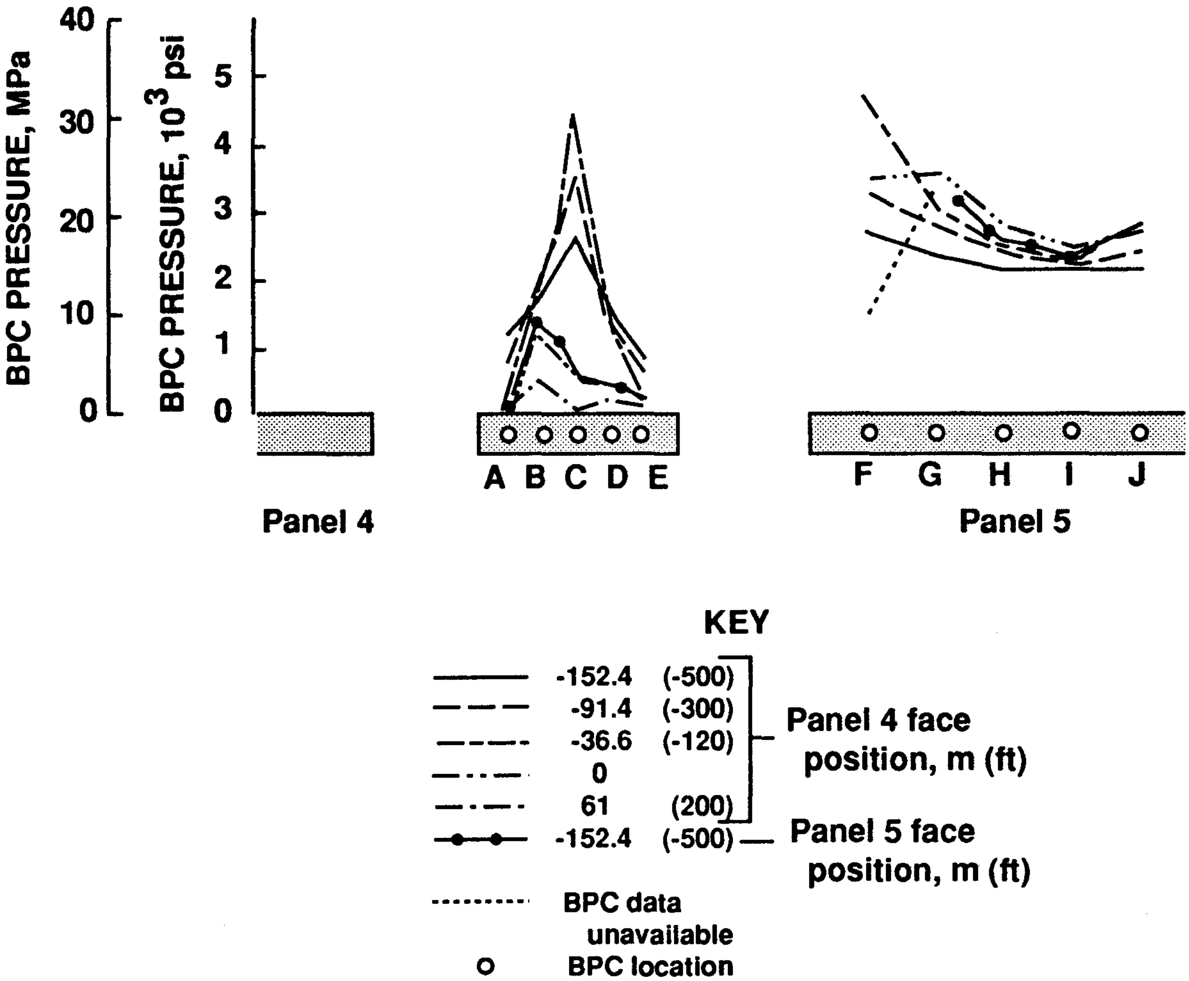
Sites 2 and 3 instrumentation in 5th Left gate road, Star Point No. 2 Mine.

Figure 32



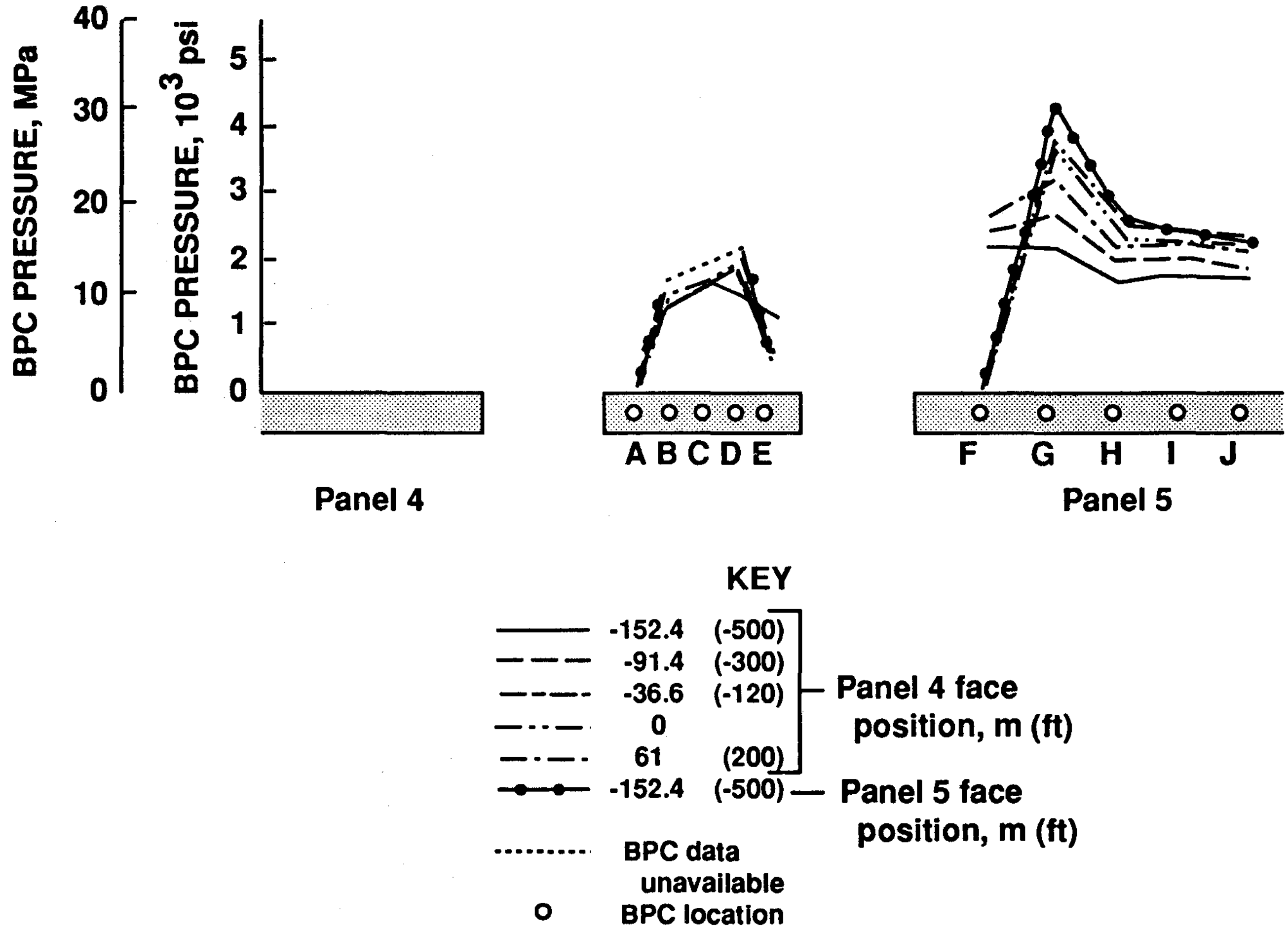
Vertical pressure profiles at site 1, Star Point No. 2 Mine.

Figure 33



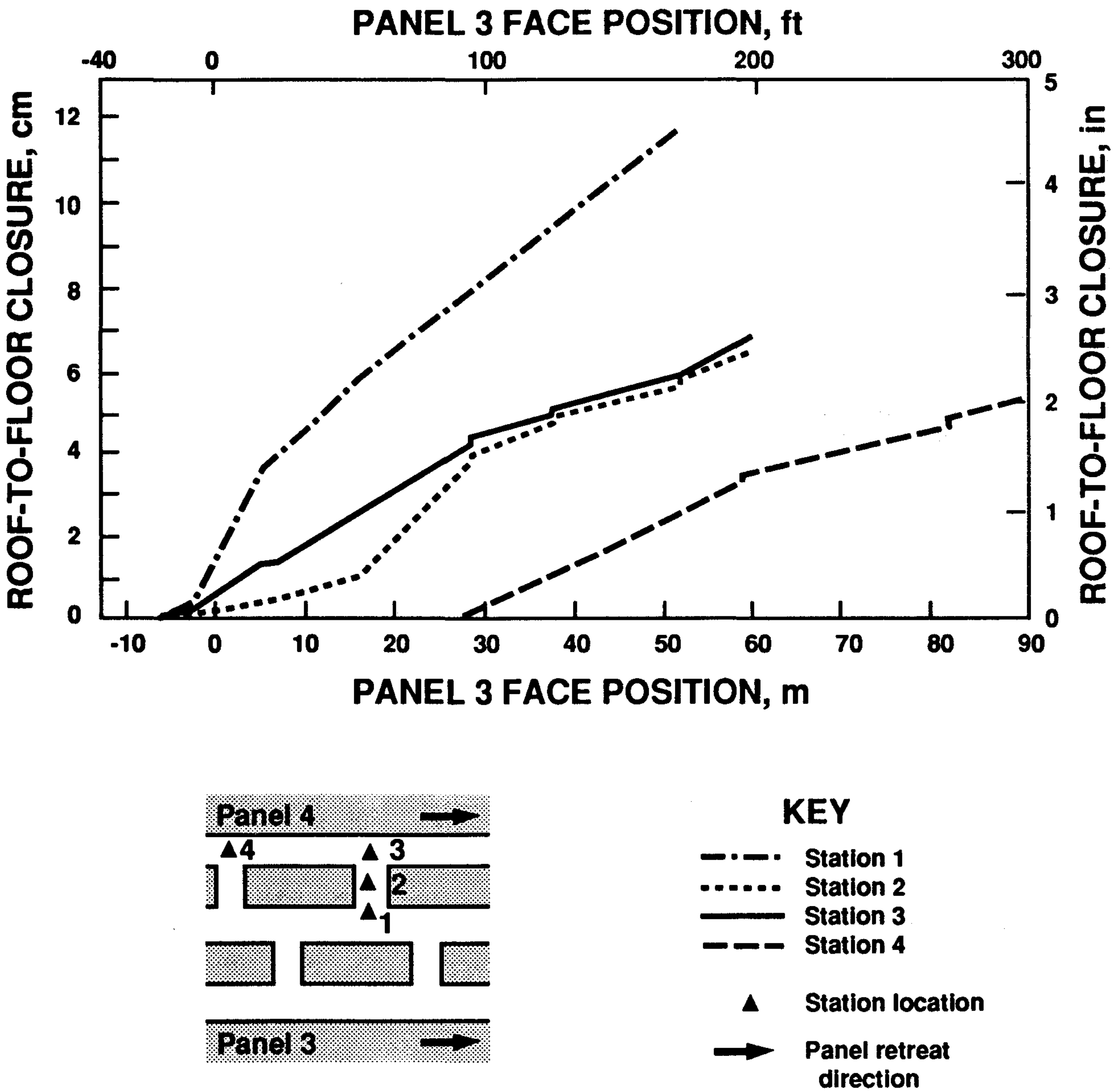
Vertical pressure profiles at site 2, Star Point No. 2 Mine.

Figure 34



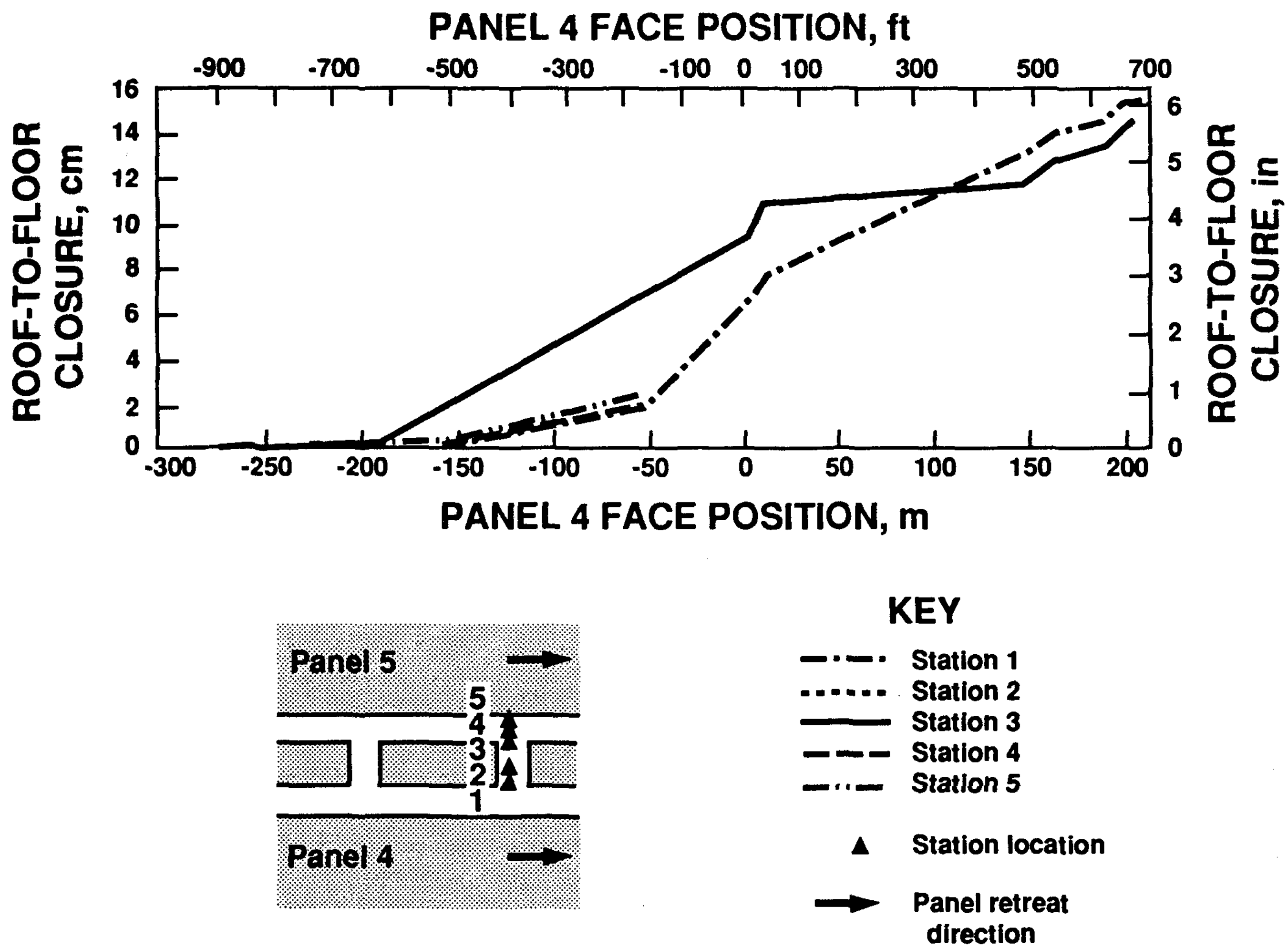
Vertical pressure profiles at site 3, Star Point No. 2 Mine.

Figure 35



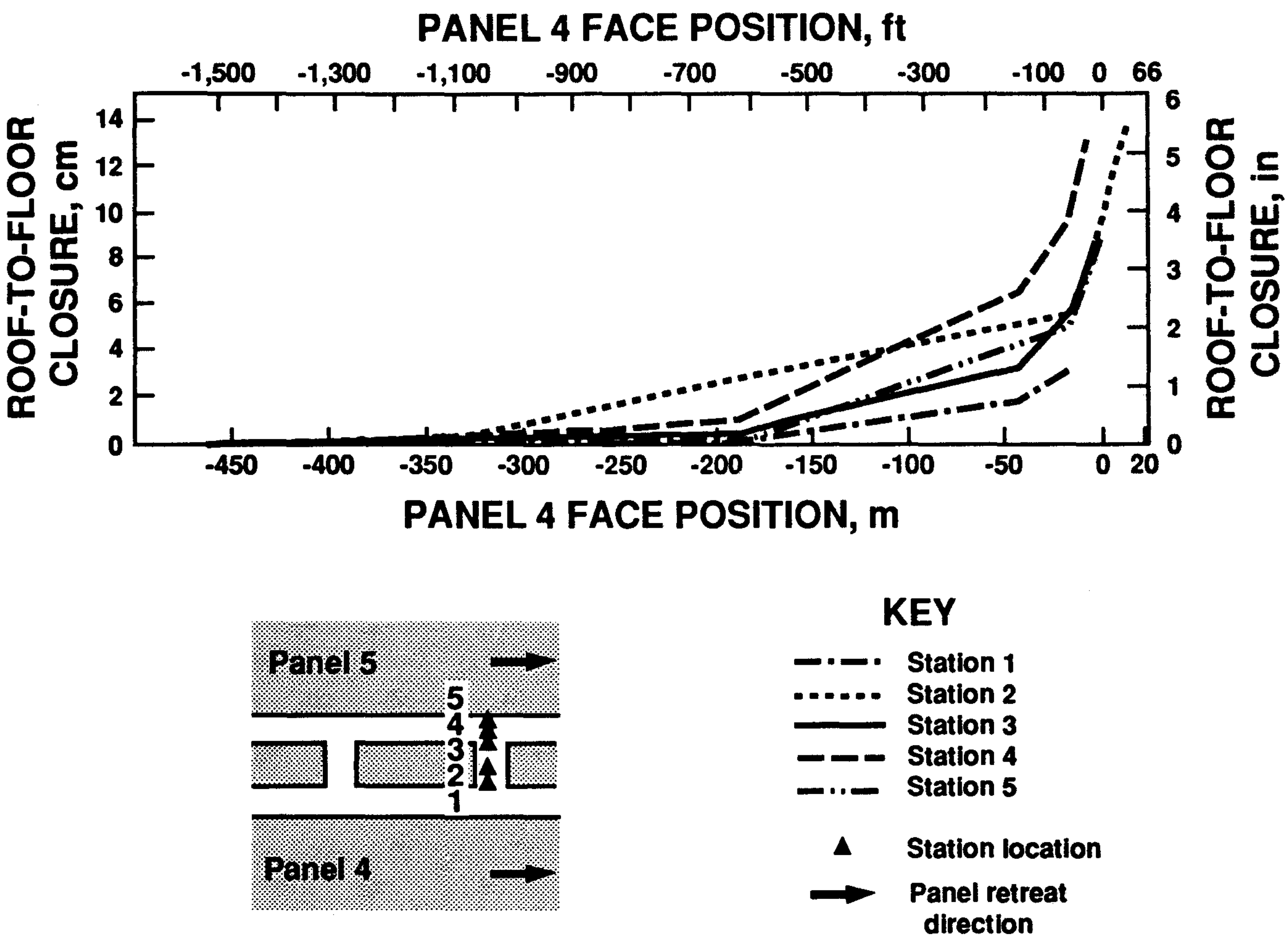
Entry closure relative to panel 3 face position at site 1, Star Point No. 2 Mine.

Figure 36



Entry closure relative to panel 4 face position at site 2, Star Point No. 2 Mine.

Figure 37



Entry closure relative to panel 4 face position at site 3, Star Point No. 2 Mine.

particularly floor heave, was much less than that observed in the 6th Left gate road.

Although each site presented a unique loading history, there were significant differences in measured behavior between the two gate road systems. Peak and total pillar and panel loads across site 1 in the 6th Left gate road were often double those experienced at sites 2 and 3 in the 5th Left gate road at comparable face positions.

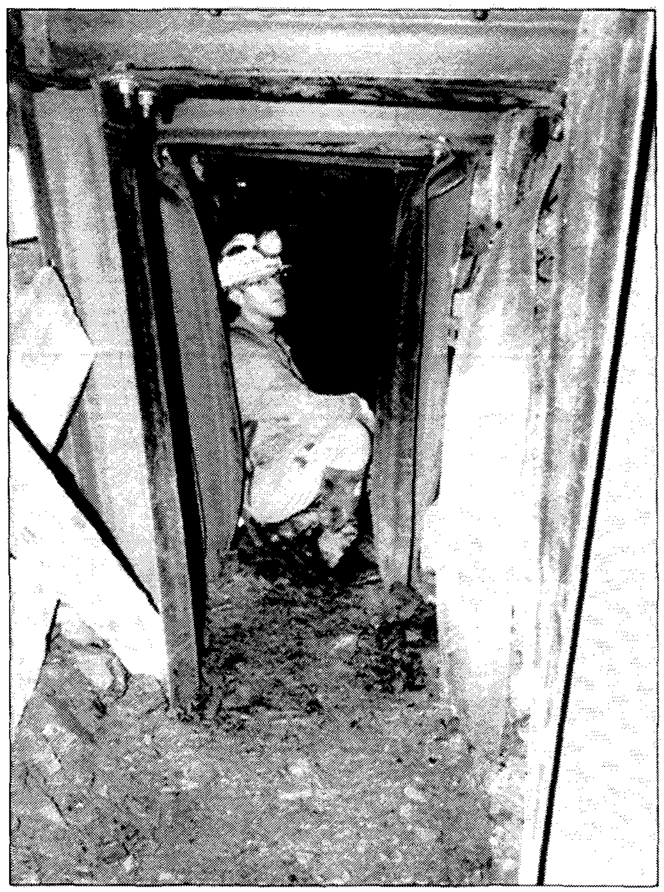
In the three-entry system, the two chain pillars apparently carried most of the abutment loading from first-panel mining, with most of the total load shifting to the next panel as the first-panel face passed by the site. In contrast, peak second-panel loads were higher than or equalled observed pillar loads throughout the mining sequence in the two-entry gate road. The three-entry gate pillars, therefore, were apparently "stiff" or "abutment" pillars, whereas those in the two-entry system met the criteria for yield pillars.

Finally, major increases in second-panel loading probably extended well beyond 15 m (50 ft) into the panel in the three-entry system, but were confined to the first 12 m (40 ft) from the panel rib in the two-entry system. The width of the panel abutment and yield zone was apparently 50% greater for the three-entry gate road than for the two-entry system, which is likely attributable to the relative difference in overall gate road width between the two systems.

Figure 38

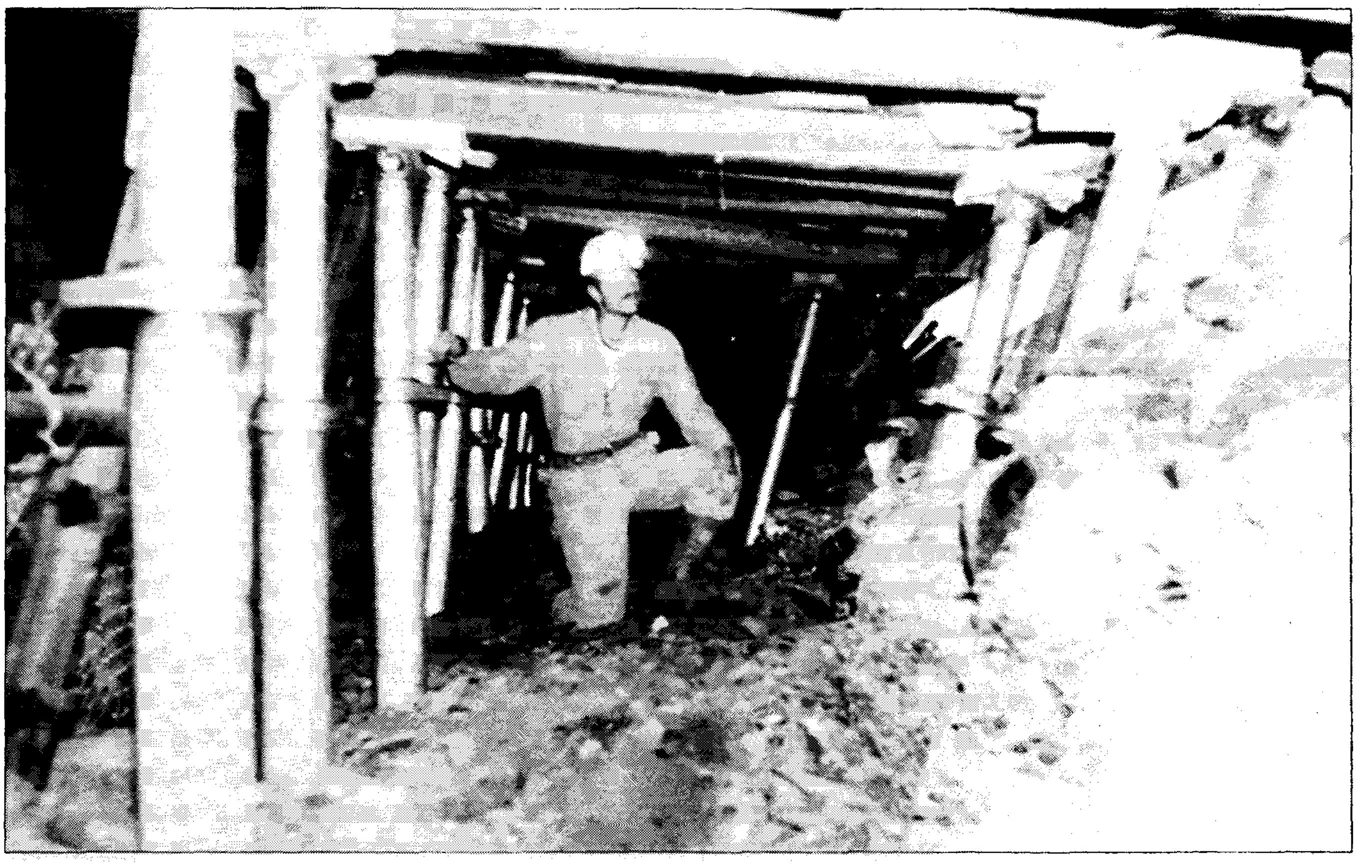
Sixth Left tailgate entry during panel 4 mining. In addition to extensive cribbing, which has crushed and rolled out at chain pillar side of the entry, steel I-beams and wood lagging have been used to support deteriorating roof.

Figure 39



"Door-frame" steel sets in 6th Left tailgate entry. Despite structural rigidity of I-beams used to construct sets, excessive loads imposed by panel 4 extraction have buckled and twisted uprights. Height of tailgate walkway, about 1.5 m (5 ft), is indicated by USBM engineer; original mining height at development was approximately 2.1 m (7 ft).

Figure 40



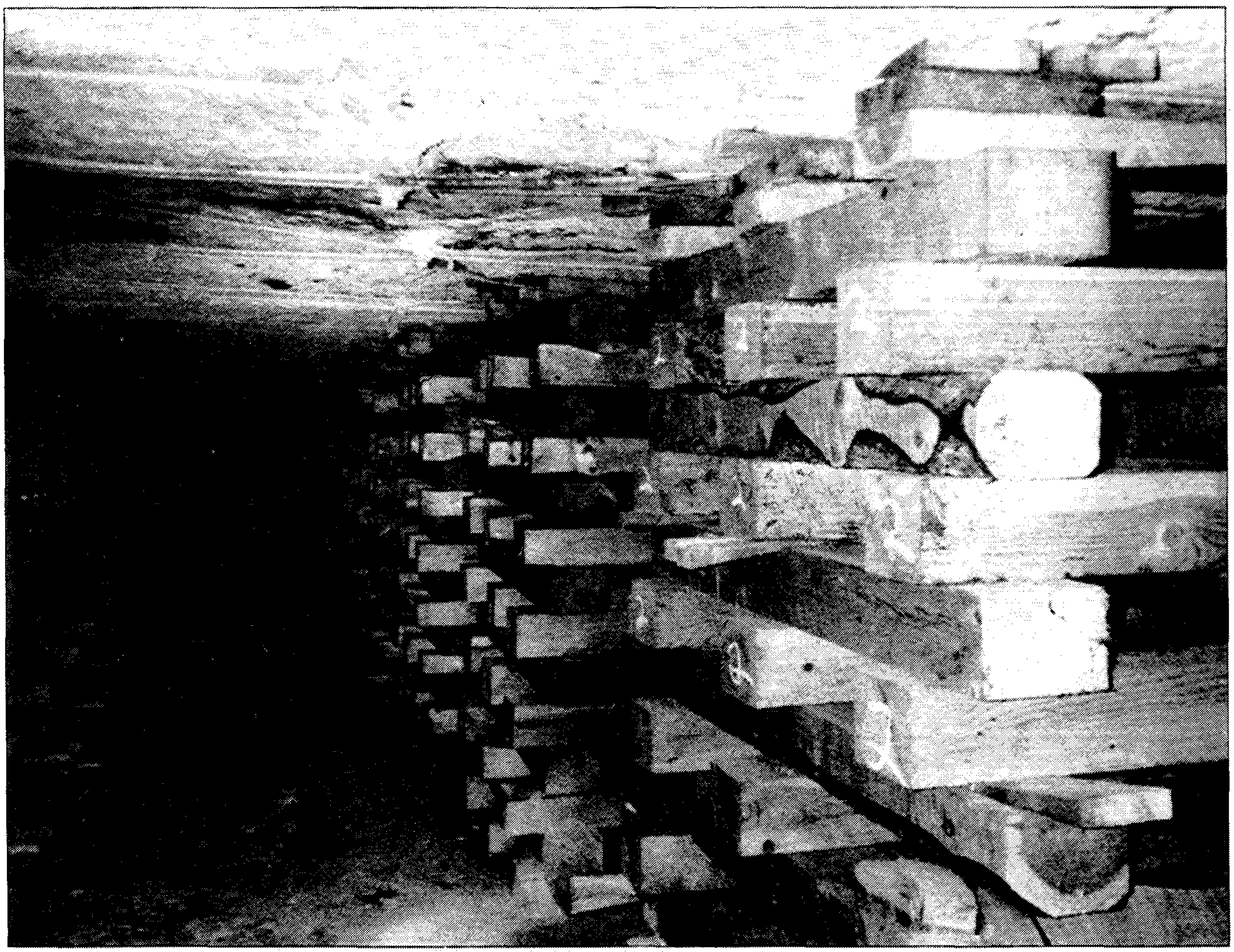
Fifth Left tailgate entry during panel 5 mining. Hydraulic supports have been used to reinforce yieldable steel sets against front abutment loading from nearby approaching face; wire mesh was utilized to prevent infilling of escape walkway by loose material from rib sloughage and minor roof falls.

STAR POINT NO. 2 MINE CONCLUSIONS

By converting from a three-entry gate road system with undersized abutment pillars to a two-entry yield-pillar system, CPMC realized several benefits (*41*):

- A marked improvement in gate entry stability, with limited minor floor heave and reduced rib sloughage.
- A reduction in roof falls, both in gate road development and panel retreat.
- A reduction in ground support requirements, particularly in tailgate mining, where the massive support required to maintain the tailgate entries in the three-entry system was replaced with standard resin-grouted bolts and yieldable flat-topped steel sets, which were, in turn, replaced with conventional wooden cribbing (figure 41).
- A reduction in concentrated load transfer onto underlying Third Bed workings, resulting in improved ground conditions for mining this seam (*37*).

Figure 41



Nine-point cribbing used for secondary support in subsequent gate roads of Star Point No. 2 Mine.

CASTLE GATE NO. 3 MINE

LOCATION

The Castle Gate No. 3 Mine is located approximately 3.2 km (2 miles) northwest of the town of Helper. The main portal and mine offices, in Hardscrabble Canyon, are accessed by a paved road that intersects U.S. Highway 6 approximately 1 km (0.6 mile) north of Helper (figure 42).

GEOLOGIC SETTING

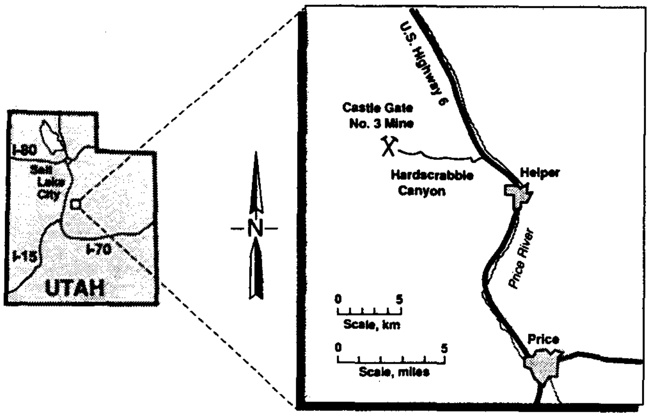
Stratigraphic Units

In ascending order, the principal stratigraphic units in the Castle Gate No. 3 Mine area are the Mancos Shale, units of the Star Point Sandstone, the Blackhawk Formation, the Castlegate Sandstone, the Price River Formation, and the North Horn Formation (figure 43) (*30*). The stratigraphic units that most affect mining operations in the No. 3 Mine are the Spring Canyon Sandstone, the Sub-3 Seam, the strata immediately overlying the Sub-3 Seam, the Aberdeen Sandstone, the Castlegate "D" Seam, the Castlegate Sandstone, and the Price River Formation (figure 44).

Spring Canyon Sandstone

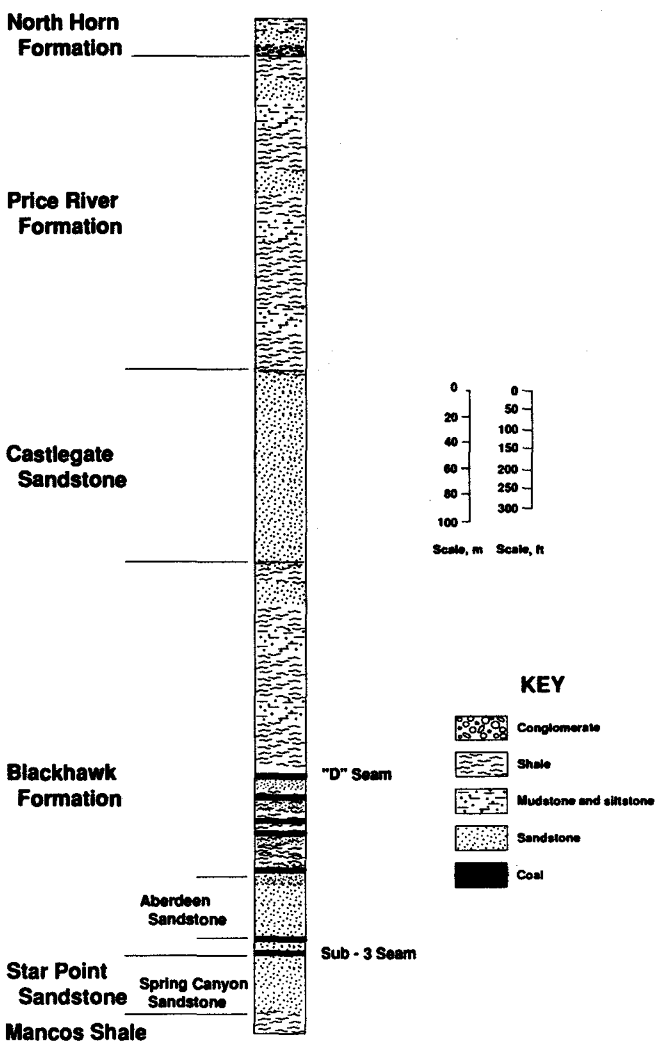
The Spring Canyon Sandstone, the uppermost sandstone unit of the Star Point Sandstone, consists of gray siltstone at the base grading upward into fine-to-medium-grained crossbedded sandstone. It is about 49 m (160 ft)

Figure 42



Castle Gate No. 3 Mine location map.

Figure 43



Stratigraphic column, Castle Gate No. 3 Mine.

thick in the Castle Gate mines area and constitutes the floor rock of the No. 3 Mine. The thickness, rigidity, and strength of this unit render it extremely competent (figure 45), as evidenced by the absence of significant floor heave or pillar punching throughout the mine. However, these same attributes undoubtedly contribute to significant pillar confinement, which in turn leads to bumping of the coal seam during mining.

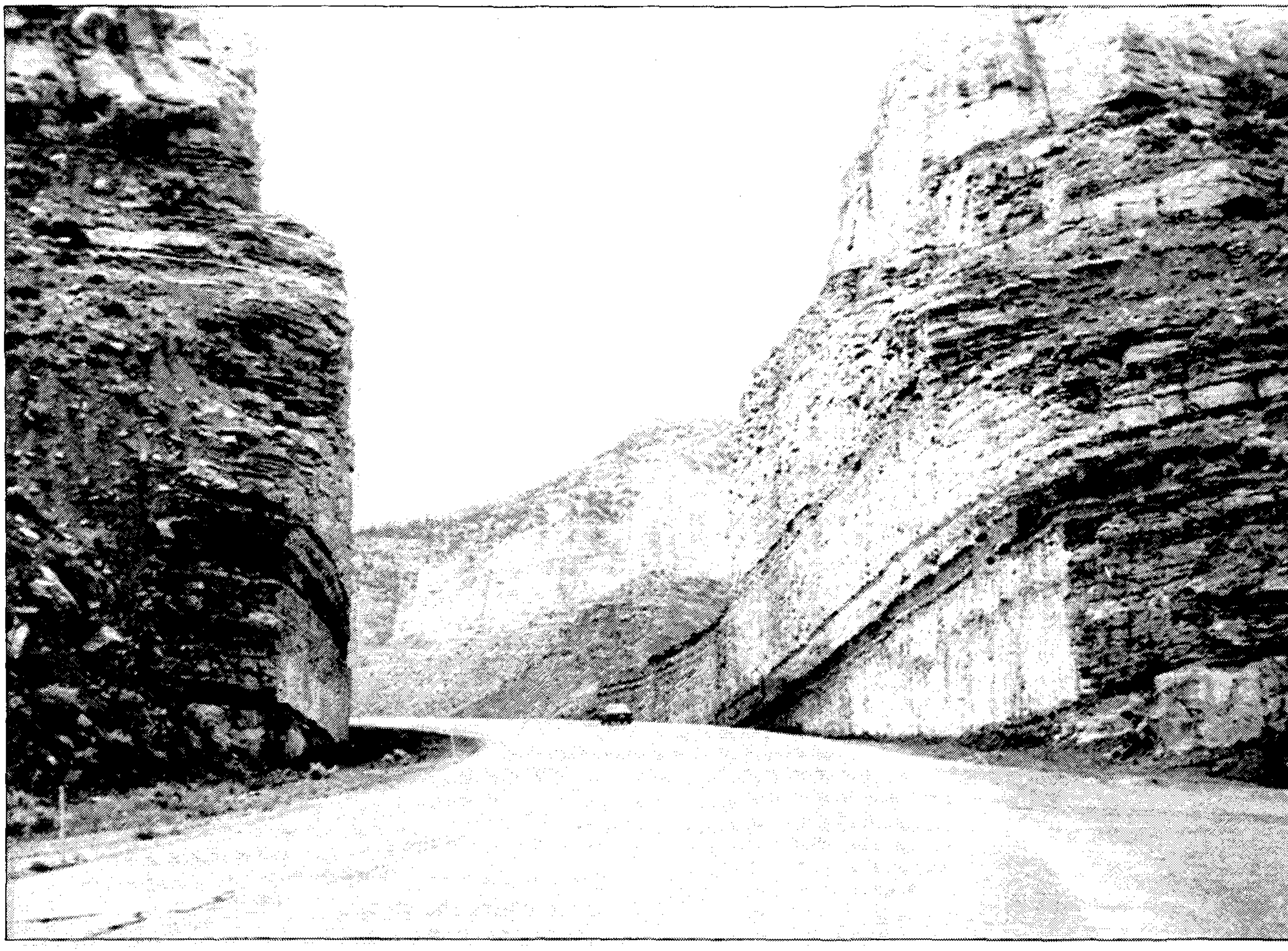
Sub-3 Coal Seam

The Sub-3 Seam, in which the Castle Gate No. 3 Mine is located, directly contacts the underlying Spring Canyon Sandstone. The seam is about 1.8 to 2.1 m (6 to 7 ft) thick in the No. 3 Mine area. Numerous bumps occurring during nearly a century of mining in the seam have amply demonstrated that the Sub-3 Seam coal can fail in a violent, explosive manner when subjected to high-magnitude loads. Linear, vertical clastic intrusions into the coal, termed "rock spars" or "clastic dikes," and rolls (localized undulations of the seam) impede mining and decrease productivity, but do not constitute ground control hazards. A hard siltstone parting at approximately midseam height is present throughout the mine.

Strata Immediately Overlying Sub-3 Seam

The strata overlying the Sub-3 Seam constitute the immediate roof rock of the No. 3 Mine. In 1932, Tomlinson (*3*) characterized the roof rock of the Sub-3 Seam as having the greatest variability of any coal mine roof in Utah and presenting the greatest roof-fall hazards in the Book Cliffs Coalfield. As noted previously, these strata exhibit a wide degree of both vertical and horizontal lithologic variation (figure 46). Most of the Sub-3 Seam is overlain by a layer of thinly laminated overbank siltstone ranging up to 3 m (10 ft) thick, the lower 15 cm (6 in) of which is a hard, brittle carbonaceous shale or siltstone termed "snaprock" by mine personnel. The siltstone is missing in some areas, where stream channels washed out the silt down to the coal-forming peat. These depressions were filled with sand, forming channel sandstones that often extend for large distances in the mine roof. The exposed bases range from 9 to 61 m (30 to 200 ft) wide, and thicknesses exposed by roof falls range from 1 m (3 ft) to at least 5 m (15 ft). When the channel sandstones contact the coal, forming the entire supported roof, roof conditions are usually good. However, many roof falls have occurred in channel sandstone zones, usually along highly slickensided mudstone laminations. In some of these roof falls, large blocks of sandstone, with roof bolts included, have fallen, leaving high, dome-shaped caved areas. The greatest roof-fall hazards associated with channel sandstones occur when 0.2 to 1.0 m (0.67 to 3 ft) of siltstone separate

Figure 44



Castle Gate area strata at U.S. Highway 6 in Price River Canyon. In road-cut exposure about 2 km (1.25 miles) east of Castle Gate No. 3 Mine, Spring Canyon Sandstone is seen at vertical sides of cut in immediate foreground. Weathered unit at top of both sides of cut is Aberdeen Sandstone; Castlegate Sandstone forms high cliff in background. Price River Formation makes up wooded slopes above Castlegate cliff; small point on ridge crest is formed of lower North Horn Formation sediments.

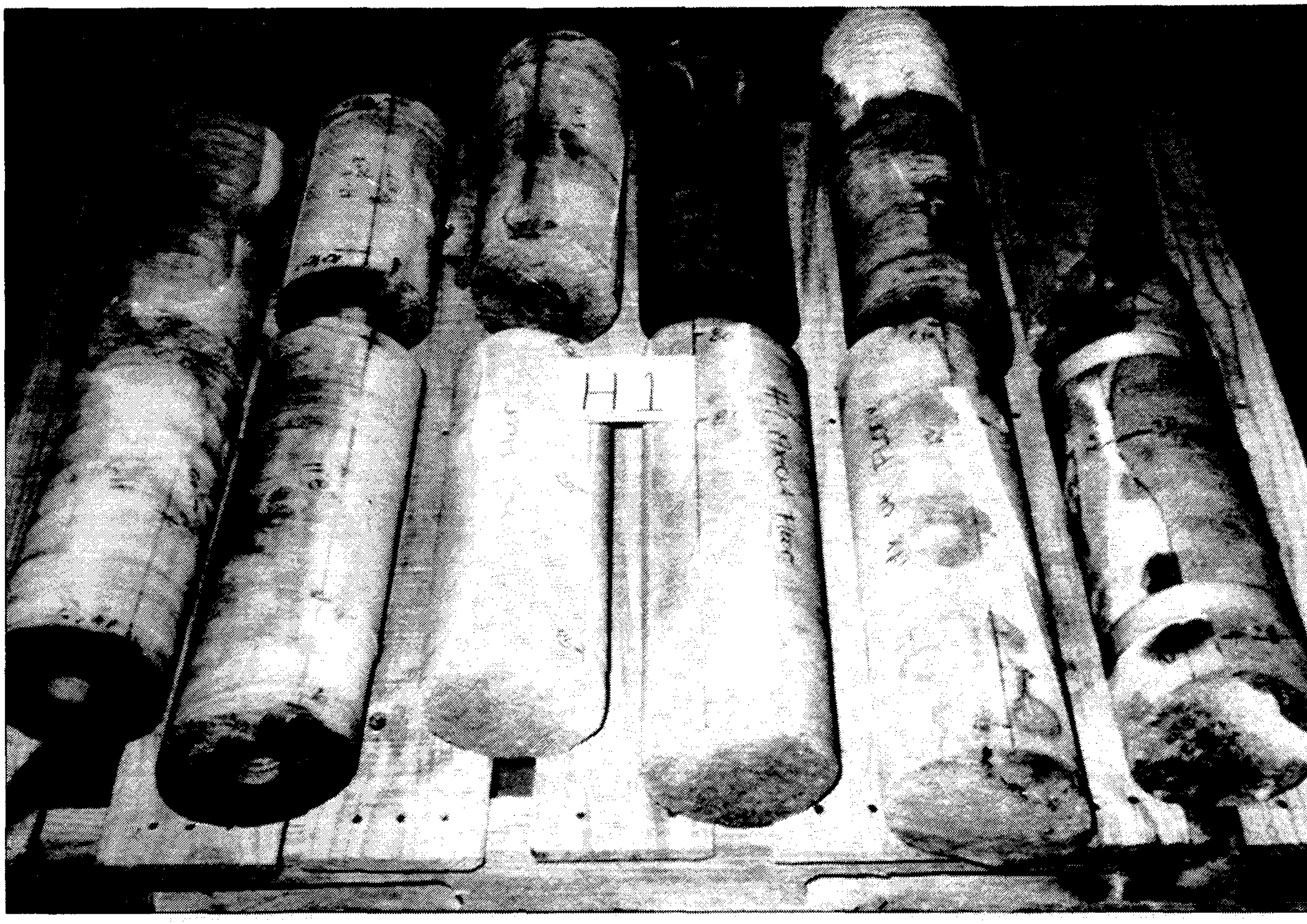
the sandstone from the coal. In these areas, fractures form in the siltstone and rapidly increase in number and severity following entry development. Large slabs of the siltstone then separate from the overlying sandstone between the roof bolts, causing roof falls or leaving dangerous large slabs suspended by the bolts (*28*). A possible additional effect of the channel sandstones may be transfer or concentration of mining-induced stresses, particularly during longwall extraction operations.

The most unstable and unpredictable roof in the No. 3 Mine consists of overbank deposits of thinly bedded sandstone, particularly in areas where cover depth exceeds 608 m (2,000 ft). The sandstone beds separate along bedding planes, creating a fractured zone of 5- to 20-cm (2- to 8-in) thick slabs, which extends up to 2 m (7 ft) into the roof (figure 47). If the uppermost bed separations are above the roof bolt anchors, large areas of the entire roof, including the bolts, will fall. When the bolt anchors are above the bed separations, a series of dangerous fractured slabs is suspended by the bolts (*28*).

Aberdeen Sandstone

The Aberdeen Sandstone is approximately 24 to 46 m (80 to 150 ft) thick in the mine area, and it is separated from the Sub-3 Seam by 12 to 30 m (40 to 100 ft) of sandstone, siltstone, shale, and coal. This thick, massively bedded, fine-to-medium-grained sandstone probably resists fracturing and delays caving above the longwall panels and could act as a rigid structure to redistribute mining-induced loads from operations in the No. 3 Mine onto unmined areas.

Figure 45



Spring Canyon Sandstone cored from immediate Castle Gate No. 3 Mine floor. Structural competency of this massive sandstone, which directly contacts bottom of Sub-3 Seam, is indicated by absence of significant fracturing or interbedding.

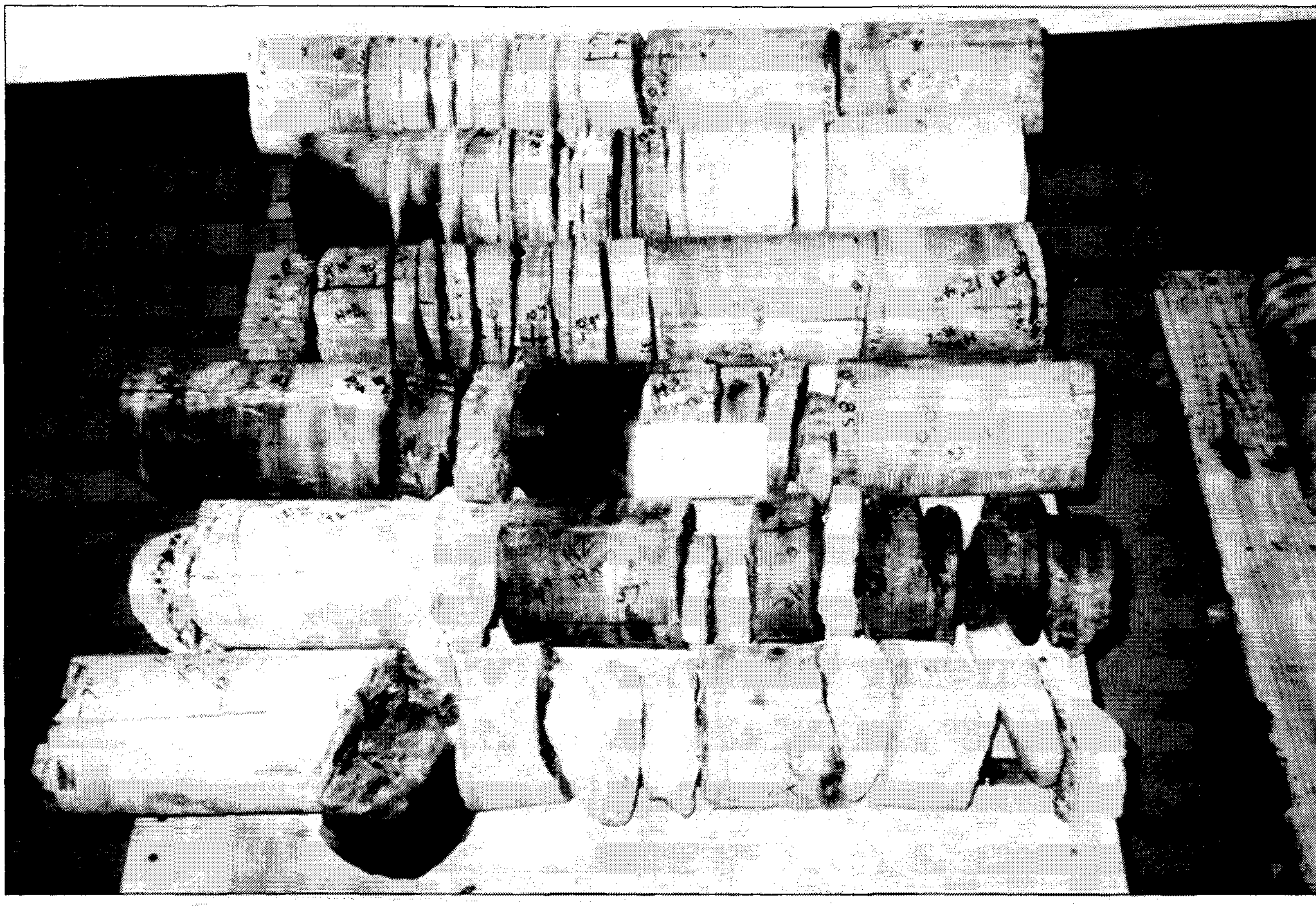
Castlegate "D" Seam

The Castle Gate No. 5 Mine and the abandoned Royal Mine are located in the "D" Seam, which lies about 73 m (240 ft) above the Aberdeen Sandstone, or approximately 131 m (430 ft) above the Sub-3 Seam. Although much of the "D" Seam workings overlies the Castle Gate No. 3 Mine, the old Royal workings and operations in the No. 5 Mine seem to have no significant effects on ground conditions in the No. 3 Mine. This may be due to load redistribution in the Aberdeen Sandstone isolating the Sub-3 Seam workings from possible "D" Seam remnant pillar stress concentrations.

Castlegate Sandstone

The Castlegate Sandstone lies unconformably upon the Blackhawk Formation about 304 m (1,000 ft) above the Sub-3 Seam. In the No. 3 Mine area, this fine-to-medium-grained sandstone is approximately 152 m (500 ft) thick, forming prominent cliffs on the upper slopes. Because of its great thickness and massive structure, the Castlegate Sandstone almost certainly has significant effects on any underlying mining operations. The primary effects are significantly increased applied loads on underlying structures and resistance to fracturing, leading to delayed caving of the main roof of the longwall panels. A USBM surface subsidence study (*42*) conducted from 1979 to 1981 on the 5th East and 6th East longwall panels of the No. 3 Mine (figure 48) showed that surface subsidence above the 5th East panel, mostly overlain by the Castlegate Sandstone, did not begin until the panel had retreated between 359 and 441 m (1,180 and 1,450 ft) (7 to 10 months after retreat began) and did not end until about 18 months after the cessation of longwall mining in the panel. Delayed caving may have led to increased forward and

Figure 46



Core taken from immediate roof of Castle Gate No. 3 Mine. Lithologic variation of this mine roof is exhibited by numerous interbedded layers of sandstone, siltstone, mudstone, and coal in first 6.1 m (20 ft) of strata overlying Sub-3 Seam. Numerous, closely spaced bedding plane fractures make this roof very difficult to support. Frequent core disking probably indicates presence of horizontal confining stresses in mine roof.

side abutment loading in active longwall panels, thus contributing to the occurrence of bumps and outbursts of coal along the gate entries and at the working faces.

Price River Formation

The Price River Formation, consisting of lenticular bedded sandstone, siltstone, and shale, conformably overlies the Castlegate Sandstone. In the mine area, it is approximately 243 m (800 ft) thick, and it makes up the uppermost slopes and most of the crests of the highest ridges. This formation accounts for up to 243 m (800 ft) of additional overburden loading at various locations across the No. 3 Mine.

Structural Features

The strata in the No. 3 Mine area dip 4° to 6° to the northeast, the steeper dips occurring in the northern portion of the area. Minor folding has occurred in the southern portion of the area. No major faults have been observed in the No. 3 Mine or the immediate surrounding area. The primary structural feature affecting ground conditions in the No. 3 Mine is the 85- to 113-m/km (450- to 600-ft/mi) dip of the coal seam from 2,025 m (6,660 ft) elevation at the mine portal to 1,626 m (5,350 ft) at the northernmost workings (both outside the area of figure 48). Combined with the general northward surface elevation rise, from 2,025 m (6,660 ft) at the mine portal

Figure 47

Castle Gate No. 3 Mine roof strata exposed at shaft station in north side of mine. Large slabs of thinly bedded sandstone, siltstone, and mudstone separate along bedding planes, leading to roof falls.

to 2,614 m (8,600 ft) over the westernmost entries, this dip results in cover depths ranging from 0 at the outcrop to nearly 821 m (2,700 ft) in the western workings of the mine (outside the area of figure 48).

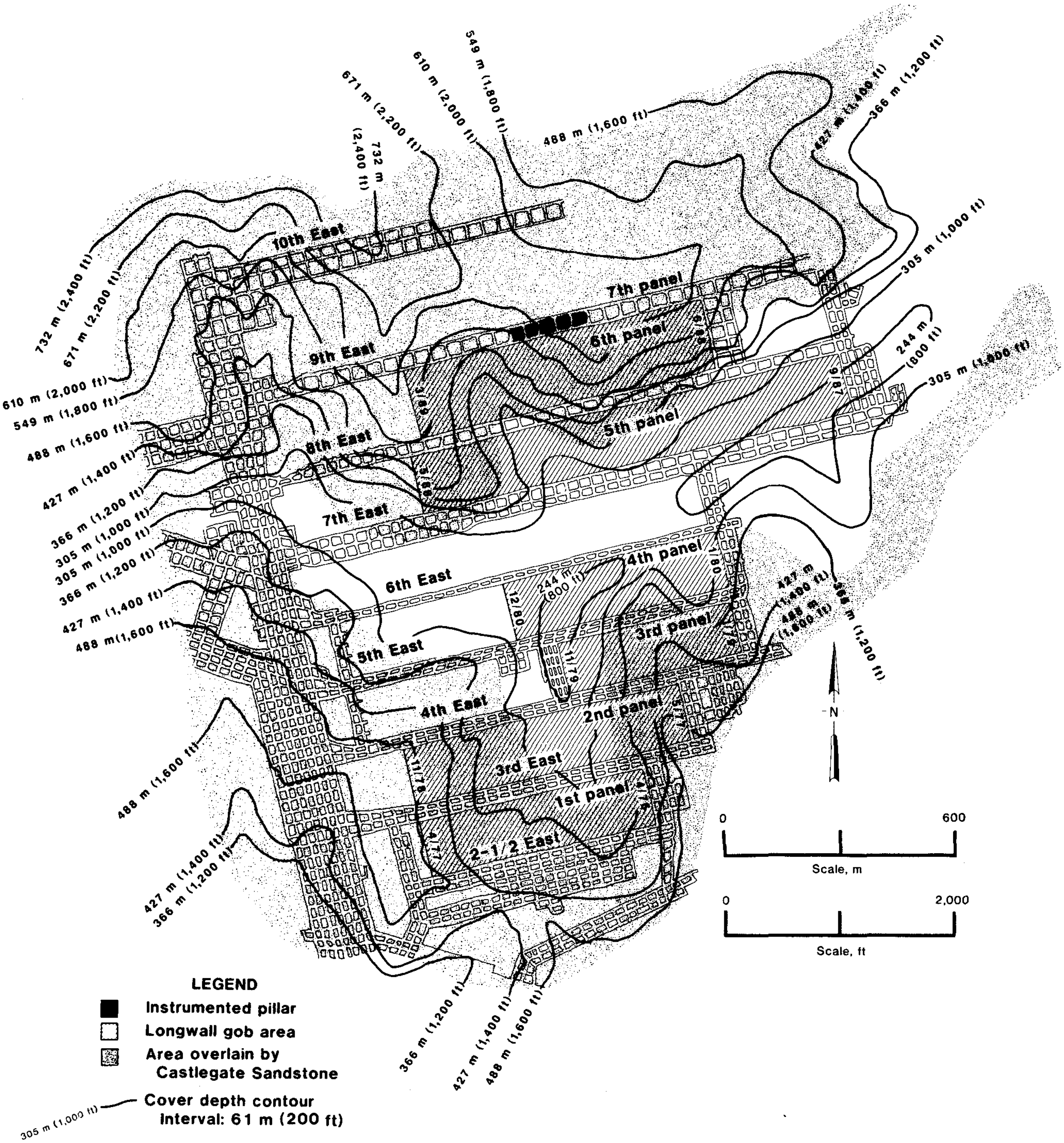
CASTLE GATE NO. 3 MINE OPERATING HISTORY[3]

Mining Prior to Longwall

From 1896, when mining in the Sub-3 Seam began, until 1974, intermittent room-and-pillar mining by several owner-operators gradually advanced the workings in the seam northward. As mining progressed downdip beneath the steep elevation rise of the mountainous surface, cover depths above the active workings continually increased, and the catastrophic events commonly associated with deep mining in mountain topography—bumps, coal outbursts, and bump-related gas outbursts and roof falls—occurred with increasing frequency. As these increasingly hazardous conditions caused more accidents, sometimes with injuries or fatalities, pillar extraction operations were terminated, and drivage of entries and rooms remained as the only type of mining permitted. By 1974, even development-only mining was considered both to be overly hazardous and to have reached its economic limits. At this time, the numerous small mining properties and operations in the Castlegate area were consolidated under the ownership of a large corporation that had the financial resources needed to equip and operate the only form of mining considered feasible in the worsening

[3]McKenna, T. A. Property Ground Control History. Internal Memorandum to D. Bryant, Castle Gate Coal Co., April 14, 1987, 15 pp.; available from L. R. Barron, U.S. Bureau of Mines, Denver, CO.

Figure 48



Longwall panels at Castle Gate No. 3 Mine. In the panel nomenclature system used at the mine, each panel was named in accordance with its headgate road; thus, the 1st panel was termed the "3rd East panel."

ground conditions—retreat longwall. This method was chosen primarily because of its success under similar conditions in Kaiser Steel Corp.'s Sunnyside Mines nearby. Development of the first longwall panel in the No. 3 Mine, 3rd East, commenced in the first quarter of 1975.

Longwall Mining

Six longwall panels were developed and extracted in the No. 3 Mine. Figure 48 shows the layout and extraction sequence of the panels, indicates the cover depth over the panels, and shows the portion of the area overlain by the Castlegate Sandstone. All longwall gate roads discussed in this section have 6.1-m (20-ft) entry and crosscut widths.

First (3rd East) Panel

The 2-1/2 East three-entry tailgate was developed in 1975 with 15.2- by 25.8-m (50- by 85-ft) pillars. These relatively narrow pillar dimensions were initially used at the Sunnyside Mines to create gate road pillars that would remain stable during entry drivage, yet yield nonviolently during longwall retreat, i.e., yield pillars. After mine personnel learned, through communications with Kaiser Steel personnel, that most of the 15.2-m (50-ft) wide pillars in longwall gate roads at the Sunnyside Mine were bumping during longwall retreat, the 3rd East three-entry headgate was developed with 12.2- by 25.8-m (40- by 85-ft) pillars in hopes that they would yield more readily, particularly during the retreat of the adjacent longwall panel (4th East). The 3rd East panel was 152 m (500 ft) wide by 632 m (2,080 ft) long and was equipped with a double-drum shearer and 500-ton-yield four-legged chocks. Panel retreat began in April 1976 and was halted in April 1977 at 605 m (1,990 ft) of retreat, leaving a 27.3-m (90-ft) barrier pillar. The 2-1/2 East tailgate pillars began failing soon after the start of panel retreat, leading to hazardous tailgate escapeway conditions. Pillar bumps and bump-related roof falls were common. Although these conditions were dangerous, they did not prevent a full retreat to the planned barrier pillar. Cover depths above this panel ranged from 334 m (1,100 ft) at the starting room to 486 m (1,600 ft) at the final barrier pillar, and the entire panel was overlain by the Castlegate Sandstone.

Second (4th East) Panel

The tailgate for this panel was the 3rd East entries developed in 1975 with 12.2- by 25.8-m (40- by 85-ft) pillars. The 4th East three-entry headgate was developed from January to December 1976 with 9.1- by 25.8-m (30- by 85-ft) pillars, in the hope that they would yield more readily during retreat of the next panel (5th East) and present less severe bump problems than did the 15.2- by 25.8-m (50- by 85-ft) tailgate pillars of the 3rd East panel. The 4th East panel was developed 152 m (500 ft) wide by approximately 806 m (2,650 ft) long, and it utilized the 3rd East panel face equipment. Panel retreat began in May 1977. Bumping in the tailgate pillars and at the working face began when the face drew even with the inby gob line (starting room) of the preceding 3rd East panel (figure 48) and became more frequent and severe as retreat progressed. Tailgate pillars 90 to 150 m (300 to 500 ft) outby the face failed violently without warning, filling the entries with loose coal and causing major roof falls. Face bumps became severe enough to throw the face conveyor off the floor and to sever one of the ranging arms from the shearer. Motorcycle helmets with impact-resistant plastic facial shields were issued to longwall personnel as protection from pieces of coal exploding from the face. Consequently, productivity in the panel was very low (481 t (530 U.S. short tons) per shift) and panel retreat was extremely slow. Mining was halted prematurely at 736 m (2,420 ft) (about 90% of the planned retreat distance) after a major bump and outburst of coal at the tailgate entry. Cover depth above the panel was about 456 m (1,500 ft) at the starting room, decreasing to approximately 274 m (900 ft) near the center of the panel length, then increasing to about 486 m (1,600 ft) near the panel halt line. Approximately the first 152 m (500 ft) and final 365 m (1,200 ft) of the panel length were overlain by the Castlegate Sandstone (figure 48).

Third (5th East) Panel

Longwall Retreat

The 4th East entries developed in 1976 with 9.1- by 25.8-m (30- by 85-ft) pillars served as the tailgate of the 5th East panel. The 5th East three-entry headgate was developed from March 1977 until June 1978 with 9.1- by 25.8-m (30- by 85-ft) pillars and angled crosscuts. It was hoped that the sharp corners of the parallelogram-shaped pillars would facilitate yielding during extraction of the next (6th East) panel. Panel width was reduced to 137 m (450 ft) in hopes that rapid retreat would alleviate overburden loading conditions at the tailgate. Planned panel retreat length was approximately 997 m (3,280 ft). Panel retreat began in January 1979 utilizing a new double-drum shearer and face conveyor, but reusing the chocks from the previous two panels. This panel experienced extremely violent bumping and outbursting of coal in the tailgate entries and at the working face, quite similar to the conditions encountered during retreat of the 4th East panel. A major bump in May 1979 destroyed the 4th East tailgate entries inby the face, and again tailgate pillar bumps 90 to 150 m (300 to 500 ft) outby the face were common. In November 1979, when the panel had retreated 465 m

(1,530 ft) (approximately 47% of the planned retreat length), face bumps became severe enough to cause total equipment failure, bringing mining to a halt. Cover depth over the panel was about 395 m (1,300 ft) at the starting room, rising to 425 m (1,400 ft) at approximately 122 to 228 m (400 to 750 ft) retreat distance in the tailgate, then decreasing to approximately 243 m (800 ft) at the halt line. Approximately the first 304 m (1,000 ft) of the mined portion of this panel was overlain by the Castlegate Sandstone.

Room-and-Pillar Mining

Following longwall halt, an attempt was made to extract some of the remainder of the panel by the continuous miner room-and-pillar method. Four rows of 6.1-m (20-ft) wide pillars were developed across the panel width (figure 48), but never extracted. The rooms between the pillars stayed open approximately 1 year, and the pillars appeared to yield gradually, with no bumps occurring. These pillars may represent the closest approximation of a true yield-pillar design for the mine's set of conditions.

Fourth (6th East) Panel

The tailgate of this panel was the 5th East entries, which had 9.1- by 25.8-m (30- by 85-ft) parallelogram-shaped pillars. The 6th East headgate consisted of two entries driven from October 1978 to July 1979, also having 9.1- by 25.8-m (30- by 85-ft) pillars. The two-entry gate road design was adopted because of its apparent success in alleviating tailgate ground control problems encountered in adjacent longwall panels in other western U.S. coal mines operating under deep cover, particularly the Sunnyside Mines. The panel width was 152 m (500 ft), and the planned retreat length was approximately 1,003 m (3,300 ft). Longwall retreat began in January 1980, utilizing new heavy-duty face equipment, including double-telescoping four-legged chocks. Tailgate and face bump problems similar to those in the previous panels were encountered. The panel retreat was halted prematurely at 578 m (1,900 ft) (about 58% of the planned retreat length) when a fire in the 5th East gob area forced the company to seal the entire longwall mine area (2-1/2 to 6th East) and abandon all machinery in the area, including the new 6th East longwall face equipment. Cover depth above the panel ranged from 304 m (1,000 ft) at the starting room to approximately 258 m (850 ft) at the retreat halt line. Only the first 243 m (800 ft) of the tailgate side of the panel was overlain by the Castlegate Sandstone.

Consultants' Reports

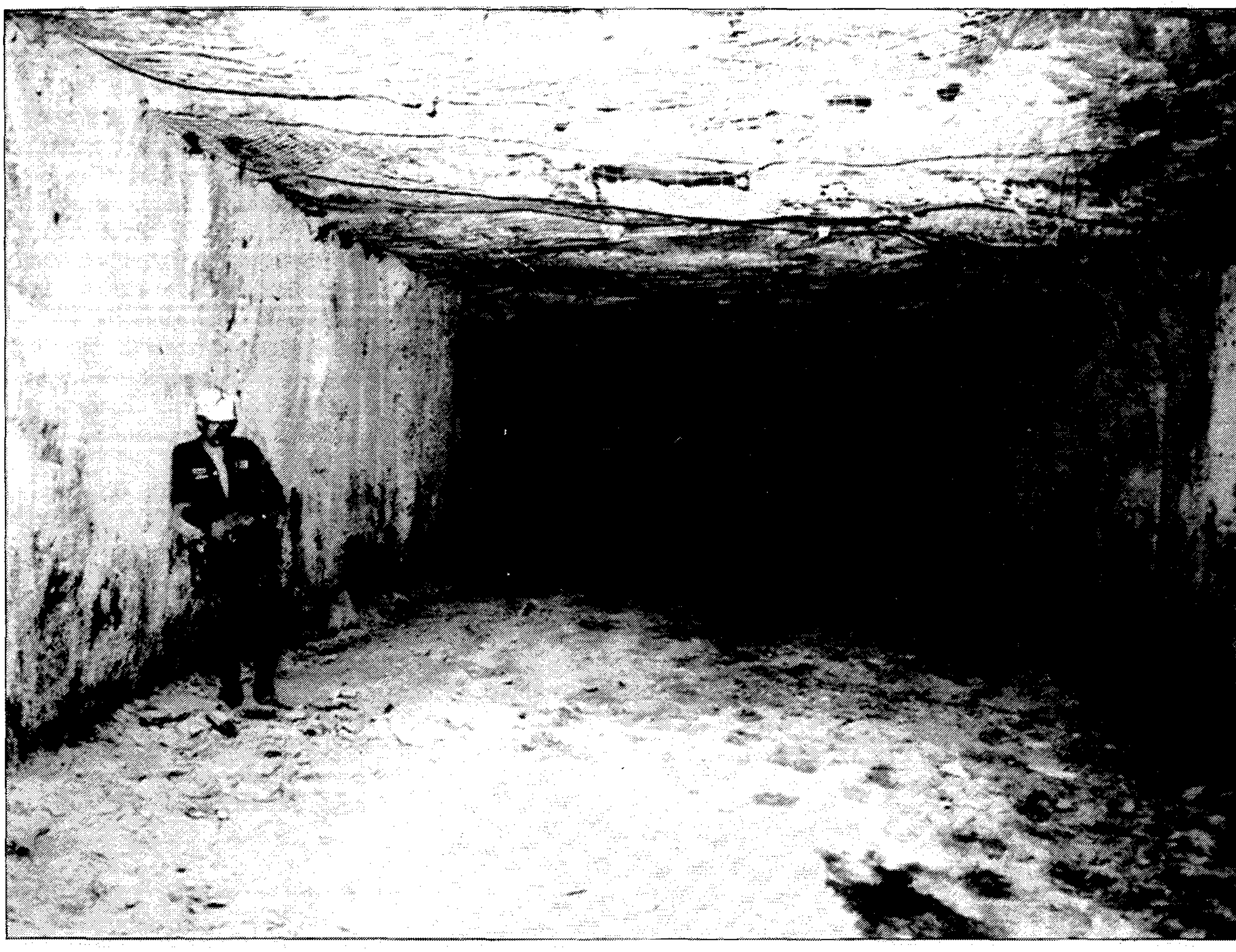
During first-panel mining, Charles T. Holland and Jack Parker, noted U.S. ground control authorities, submitted separate reports detailing design practices and entry layouts for the entire mine. Although Holland and Parker disagreed about the configuration of main and submain entries and chain pillar sizes, they both recommended longwall gate road pillars that would crush or yield during panel extraction.

In June 1979, during third-panel retreat, PD-NCB Consultants, Ltd., of Great Britain submitted a report of their study of longwall design and operations during extraction of the second panel and startup of the third panel. Based on analyses using design methods developed by A. H. Wilson, noted British ground control authority, PD-NCB recommended that future gate roads utilize large, block pillars capable of withstanding full abutment loads during panel retreat and that panel widths be increased to promote caving of the massive roof strata. The report also recommended that the coal block between the 6th East and 7th East entries be left intact to serve as a massive barrier isolating future panels from loading attributable to the previous panels. Development of the 7th, 8th, and 9th East gate roads proceeded in accordance with these suggestions. Cover depth above this barrier pillar averaged approximately 274 m (900 ft), and none of it was overlain by the Castlegate Sandstone.

Longwall Panels in No. 5 Mine

After the 5th East fire, longwall retreat operations were discontinued in the No. 3 Mine and shifted to the west side of the No. 5 Mine in the "D" Seam (figure 49), 131 m (430 ft) above the No. 3 Mine workings in the Sub-3 Seam. The horizontal (map) distance between the No. 5 Mine panels and the No. 3 Mine panels was at least 610 m (2,000 ft). From June 1980 until July 1987, five consecutive adjacent longwall panels, utilizing a new set of face equipment, were successfully developed and retreated in the No. 5 Mine. Five sets of headgate roads were driven for these panels, all using a two-entry design with large block pillars, the first set developed with 25.8- by 25.8-m (85- by 85-ft) pillars and the remaining four sets with 36.5- by 25.8-m (120- by 85-ft) pillars. All five panels were 195 m (640 ft) wide, and retreat lengths ranged from 383 to 784 m (1,260 to 2,580 ft), averaging about 562 m (1,850 ft). Cover depth ranged from 365 to 669 m (1,200 to 2,200 ft), and all five panels were wholly overlain by the Castlegate Sandstone. No major bumps occurred during development or retreat of any of these panels, and with the exception of a significant coal outburst in the bleeder entries inby the panels, the only ground control problems encountered were floor heave, minor roof falls, and small outbursts of coal from the longwall faces and tailgate pillar ribs. Although these outbursts were "small" compared with the outbursts and bumps experienced in the No. 3 Mine, in terms of the relative magnitudes of energy released, they nevertheless remained a potential injury hazard to mine

Figure 49



Bleeder entry in Castle Gate No. 5 Mine. Intact, unfractured ribs and minimal rib sloughage are typical of these workings. "D" Seam height, approximately 3 m (10 ft), is indicated by mine engineer beside rib.

personnel in close proximity to the outbursts, particularly at the longwall faces.

Mine personnel attributed the relative success of these panels, in comparison with the No. 3 Mine panels, to several interdependent factors in addition to the large-abutment-pillar gate road design.

- The entire No. 3 Mine is overlain by the rigid 46-m (150-ft) thick Aberdeen Sandstone, approximately 15 m (50 ft) above the No. 3 workings, whereas the No. 5 Mine workings are about 73 m (240 ft) above the Aberdeen.
- The No. 5 Mine panels were oriented parallel to a major joint set in the immediate roof, thus reducing the potential for interpanel cantilevering and side abutment load override.
- The No. 5 Mine longwall faces were aligned parallel to a secondary roof joint set, facilitating gob area roof caving in each panel.
- Although numerous broad channel sandstones were present in the immediate roof in both mines, it was believed that the channel sandstones in the No. 5 Mine roof were not as stiff as those in the No. 3 Mine roof, thus reducing the potential for the tailgate pillar and face bumping associated with panel extraction directly beneath these units.

Resumption of Longwall Operations in No. 3 Mine

In August 1987, after Castle Gate Coal Co. became the owner and operator of the Castle Gate Mines, longwall operations were resumed in the No. 3 Mine. During 1980 through 1982, while longwall mining was conducted solely in the No. 5 Mine, the 7th East, 8th East, and 9th East gate roads (figure 48) of the No. 3 Mine were developed with large pillars. Since these gate roads met all the criteria established by the MSHA task force (*12*) and since mine personnel believed that the relative success of the two-entry, large-pillar gate road design in reducing bumps and roof falls had been demonstrated in the No. 5 Mine panels (particularly when contrasted with the unfortunate history of the No. 3 Mine panels with relatively narrow-pillar gate roads), permission to mine the 8th and 9th East panels using the existing two-entry gate roads was obtained from MSHA.

Fifth (8th East) Panel

The 7th East three-entry tailgate was driven using 25.8- by 25.8-m (85- by 85-ft) pillars to a distance approximately 471 m (1,550 ft) from the Northwest Mains. The width of the pillars between the two southernmost entries was then decreased 10.6 m (35 ft), resulting in 15.2- by 25.8-m (50- by 85-ft) pillars adjacent to the 7th East panel barrier pillar and 25.8- by 25.8-m (85- by 85-ft) pillars adjacent to the 8th East longwall panel (figure 48). At about 927 m (3,050 ft) from the Northwest Mains (15 crosscuts outby the 8th East panel starting room) the pillar sizes were increased to 25.8- by 36.5-m (85- by 120-ft) and 15.2- by 36.5-m (50- by 120-ft). The 8th East two-entry headgate was developed using 25.8- by 36.5-m (85- by 120-ft) pillars. The panel was 158 m (520 ft) wide, and the original planned retreat length was approximately 1,444 m (4,750 ft). Longwall mining began in September 1987 and was halted in April 1988 at 1,149 m (3,780 ft) retreat (approximately 80% of the planned retreat distance) when the face arrived at an area in the headgate belt entry where a roof fall had occurred during development, and steel arches had been installed to support the roof. Since the face equipment could not be advanced through the arches, and the arches could not be removed without exposing unsupported roof, retreat was discontinued. No major bumps or accidental roof falls occurred during the retreat of this panel; however, loud "booming" noises were heard above the retreating face and the inby panel gob area. The generally favorable ground conditions encountered in this panel are probably attributable to the massive 7th East panel barrier pillar protecting the tailgate pillars and working face from excessive loading. Cover depth above the panel ranged from approximately 243 m (800 ft) above the starting room to 517 m (1,700 ft) above the headgate side of the halt line. The headgate side of the panel, from approximately 243- to 608-m (800- to 2,000-ft) retreat distance, and the entire width of the panel, from about 760 m (2,500 ft) retreat distance to the halt line, were overlain by the Castlegate Sandstone.

Sixth (9th East) Panel (*43*)

Longwall Retreat

The tailgate of the 9th East panel was the 8th East entries with 25.8- by 36.5-m (85- by 120-ft) pillars (figure 48). The 9th East two-entry headgate was developed using 36.5- by 36.5-m (120- by 120-ft) pillars (figure 50). The face width was 185.4 m (610 ft), and the planned retreat distance was approximately 1,064 m (3,500 ft). Panel retreat commenced in June 1988. Beginning in September, when the face had retreated approximately 182 m (600 ft), and continuing through December to approximately 426 m (1,400 ft) retreat, severe bumping occurred at the face and in the tailgate pillars (figure 51). No major bumps occurred during January 1989, and only one significant bump, causing no major structural damage, occurred in early February. Panel retreat progressed rapidly, with high production during this 2-month period. Bumping resumed in March 1989 at approximately 736 m (2,420 ft) retreat and continued until March 23, when the face was halted at 809 m (2,660 ft) retreat because of a face bump. The bumping encountered during September through December 1988 and in March 1989 occurred when the face was beneath ridges and overlain by more than 486 m (1,600 ft) of overburden, including the full thickness of the Castlegate Sandstone; whereas during the relatively bump-free period of January and February 1989, the face was below an area where the Castlegate Sandstone had been eroded by a small canyon, and cover depths were less than 486 m (1,600 ft) (figure 48).

Destressing To Alleviate Bumps

Panel coal destressing by water infusion was initiated in October 1988 and continued until the face was halted. Two infusion procedures were developed and used concurrently. In the first procedure, 51-mm (2-in) diameter holes spaced 9.1 m (30 ft) apart were drilled 6.1 to 27.4 m (20 to 90 ft) into the tailgate panel rib, beginning at least 12.2 m (40 ft) outby the face, and a high-pressure hose with packer was inserted into each hole. Whenever the face had progressed to within 3 m (10 ft) inby a hole, the hose was connected to a pump, and face-support-shield hydraulic fluid, an emulsion of water and soluble oil, was pumped into the hole at an approximate pressure of 27.6 MPa (4,000 psi). Each hole was pressurized for a duration of 30 min or until a minor bump was induced. When the tailgate panel rib holes were drilled with a hand-held drill, maintaining hole alignment parallel to the local seam dip and at midseam height proved very difficult. Because of secondary-support timbers installed in advance of the face, working space and equipment access in the tailgate entry were limited, and use of a machine-mounted drill was not feasible.

Figure 50

Intake entry of 9th East gate road. Even, unfractured ribs are typical of gate road conditions before approach of 9th East panel face and indicate minimal pillar yielding.

The second procedure consisted of drilling three or four holes 6.1 m (20 ft) into the face. These holes were evenly spaced along the face between a point approximately 39.5 m (130 ft) from the tailgate rib and the tailgate panel corner. After all holes were drilled, they were successively pressurized in the same manner as the tailgate rib holes, starting at the hole farthest from the tailgate, then progressing toward the tailgate. This procedure was conducted at intervals of 10 cuts by the shearer, approximately 7.6 m (25 ft). Face-hole destressing was also time consuming and precluded shearer cutting at the section of the face being destressed until the procedure was completed, thus causing significant production delays.

These procedures, utilized concurrently, generally proved effective in destressing the tailgate area of the face, thus alleviating severe unanticipated bumping; however, the difficulty of identifying optimum destressing times and locations, inability to assess the effectiveness of each destressing attempt, limited face destressing duration to avoid production interruptions, and adverse drilling conditions inhibited overall success of the effort.

Mine Idled

The mine was idled in April 1989, precluding complete 9th East panel retreat and full development of the 10th East gate road.

Figure 51

Aftermath of tailgate pillar bump in 8th East gate road. The bump, which occurred outby approaching 9th East panel face, partially filled crosscut with coal thrown from pillars at either side and destroyed solid concrete block stopping.

USBM FIELD STUDY *(43)*

In July 1988, at the request of the mine operator upon startup of the sixth panel, the USBM initiated a research effort into gate road design alternatives suitable for deep, mountainous, bump-prone conditions. The purpose of the USBM study was to characterize ground conditions prior to panel retreat, quantify ground response to panel extraction, verify the performance of the current gate road design, and utilize the results of these efforts to predict probable ground behavior during extraction of the next panel.

The research was conducted in three phases: first, a site investigation of roof, floor, and coal physical properties and in situ stress prior to panel extraction; next, an instrumentation study of pillar loading and entry closures during panel retreat; and last, an analysis of the data resulting from these efforts in conjunction with the panel retreat history in order to formulate engineering parameters for future panels.

The study site selected was in the 9th East headgate and consisted of five 36.5- by 36.5-m (120- by 120-ft) pillars (figure 48). The site was sufficiently remote from the existing face to avoid extraction-related loading effects when in situ stress was measured and to allow installed instruments sufficient time to reach equilibrium before being affected by extraction-induced loading, yet it was close enough for collected data to be incorporated into design considerations for the next panel.

Site Investigation

Determination of In Situ Stresses

Mine personnel had postulated that the absence of faulting in the mine area, in light of major faulting 4.8 to

6.4 km (3 to 4 miles) east and west of the mine, might indicate the presence of unrelieved tectonic stresses in the area. An in situ stress evaluation of both the roof and floor, utilizing the stress-relief overcoring method (*44-45*), determined that the maximum (P) and minimum (Q) horizontal compressive stresses were

Roof: P = 15.01 MPa (2,177 psi), N. 28° W.

Q = 12.50 MPa (1,813 psi), N. 62° E.

Floor: P = 11.96 MPa (1,734 psi), N. 28° W.

Q = 11.17 MPa (1,620 psi), N. 62° E.

These stresses range from nearly three to four times greater than the approximately 3.8 - MPa (550-psi) horizontal stress attributable to overburden loading through Poisson's effect at the 608-m (2,000-ft) depth of the overcoring site (assuming 25 kPa vertical stress per meter of depth (1.1 psi/ft) and an average Poisson's ratio of 0.2 for the overburden strata), indicating the possible presence of significant horizontal tectonic stresses. Since the horizontal principal stresses for both the roof and floor were nearly equivalent to the 15.2-MPa (2,200-psi) estimated vertical stress, however, and the maximum and minimum horizontal principal stresses nearly equaled each other for both the roof and floor, it was concluded that the possible horizontal stress field at the site was not highly biaxial, and overburden-related vertical stress was probably the most significant source of structure loading at the mine.

Physical Property Testing

Laboratory strength testing of roof, floor, and coal samples collected at the study site was conducted in accordance with American Society for Testing and Materials (ASTM) standard procedures. Tests performed on samples from the lower 5.5 m (18 ft) of the roof rock indicated unconfined compressive strengths ranging from 77.9 to 280.0 MPa (11,300 to 40,600 psi), with elastic moduli ranging from 22,345 to 34,621 MPa (3.24 to 5.02 million psi). Samples taken from the upper 4.6 m (15 ft) of the floor indicated uniaxial compressive strengths ranging from 75.9 to 186.2 MPa (11,000 to 27,000 psi) with elastic moduli ranging from 20,000 to 26,897 MPa (2.9 to 3.9 million psi). Testing of the coal samples indicated the coal has an average unconfined compressive strength of approximately 20.7 MPa (3,000 psi) and an average elastic modulus of approximately 4,621 MPa (670,000 psi). Burst testing of the coal showed that it yields nonviolently when loaded between platens made of Teflon fluorocarbon polymer, but bursts violently when loaded between stiff glass platens, demonstrating that when confined between stiff rock units, the coal will fail in a violent, explosive manner. The rigidity and strength of the roof and floor rocks undoubtedly contribute greatly to pillar confinement, which, in combination with the demonstrated burst proneness of the coal, leads to bumping of the coal seam during mining (*46*).

Instrumentation Study

An instrument array consisting of 17 BPC's oriented to measure vertical loads, 3 BPC's to measure horizontal loads, and 18 roof-to-floor closure stations was installed at the 9th East study site, as shown in figure 52. This array was expected to provide a comprehensive chronology of pillar and panel loading in response to face progression, profile load configurations across the pillars and adjacent panel sides, and quantify entry closure resulting from load-induced pillar yielding. It was further believed that correlating ground pressure and closure data directly to relative longwall face position offered the best means of relating retreat progression to loads and entry closure and correlating data from instruments in different locations.

Ground Pressure Data Analysis

The history of midsite pillar and panel loading with respect to relative face position is presented in figures 53-55. The data measured at this pillar were corroborated by BPC data from the four other pillars (figure 52). As the face approached the study site, the onset of the forward abutment load became apparent at -91.2 to -121.6 m (-300 to -400 ft). In each instrumented pillar, the cell nearest the extracted (9th East) panel indicated forward abutment loading prior to the cell farthest from the panel. Also, the cells nearest the 9th East panel indicated greater loading rates and higher peak loads than did those more remote from the active panel.

Vertical load response of the row of cells across the midsite pillar is shown in figure 53. The hydraulic tubing for cell 5V was damaged by mobile equipment when the face was about 30.4 m (100 ft) outby the BPC location, and 6V did not respond to ground loads in the same way as other cells the same distance from the active panel (possibly because the 6V cell may have been emplaced in the hard siltstone midseam parting). The cells in this pillar, nevertheless, indicated forward and side abutment loadings similar to those in the other pillars; i.e., the cells nearest the active 9th East panel (9V, 8V, and 7V) showed steeper loading rates and greater total load changes than the cells farther from mining (6V and 5V). Although the pillar was subjected to high-magnitude loading, over 69 MPa (10,000 psi) for cell 9V, no yielding or sloughage of the pillar ribs occurred, indicating high lateral confinement of the pillar. This premise was corroborated by the high horizontal confining loads indicated by cells 1H, 2H, and 3H (figure 54).

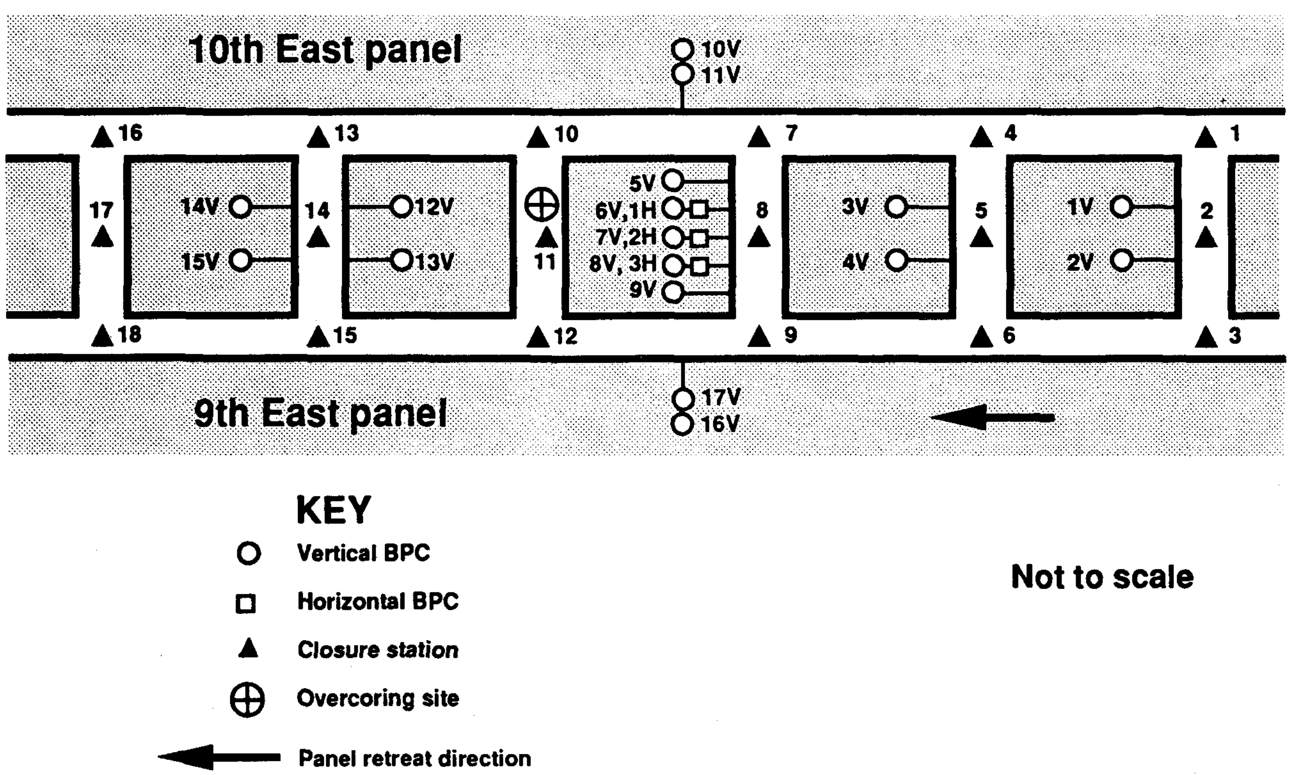
The cells in the active panel headgate rib (16V and 17V) indicated dramatic increases in both loading rate and total load as the face approached the cells, the greatest loading taking place within 6.1 m (20 ft) of the face (figure 55). The data were corroborated by observations that holes drilled into the tailgate panel rib showed no crushing or deformation until the face had approached to within 6.1 m (20 ft) of each hole. As the face approached within 2.4 m (8 ft) of cells 16V and 17V, the indicated loads exceeded 69 MPa (10,000 psi) (off-scale on the pressure recorder chart) and remained at that level until the cells were cut out by the shearer, indicating that no coal yielding occurred in advance of the face. The absence of any face yield zone and concentration of the forward abutment load at the face indicate high confinement across the seam, which almost undoubtedly accounts for the violent bumping and outbursts experienced at the face throughout most of the panel retreat.

Override of the headgate pillar by the side abutment load was demonstrated by the gradual load response of cells 10V and 11V (figure 55) placed at 9.1 m (30 ft) and 12.2 m (40 ft) into the tailgate rib of the next (10th East) panel.

No substantial changes in BPC loading rate or load magnitude were directly attributable to bumping, probably because bumps generally occurred at the tailgate side of the panel, at least 152 m (500 ft) from the instrumented pillars.

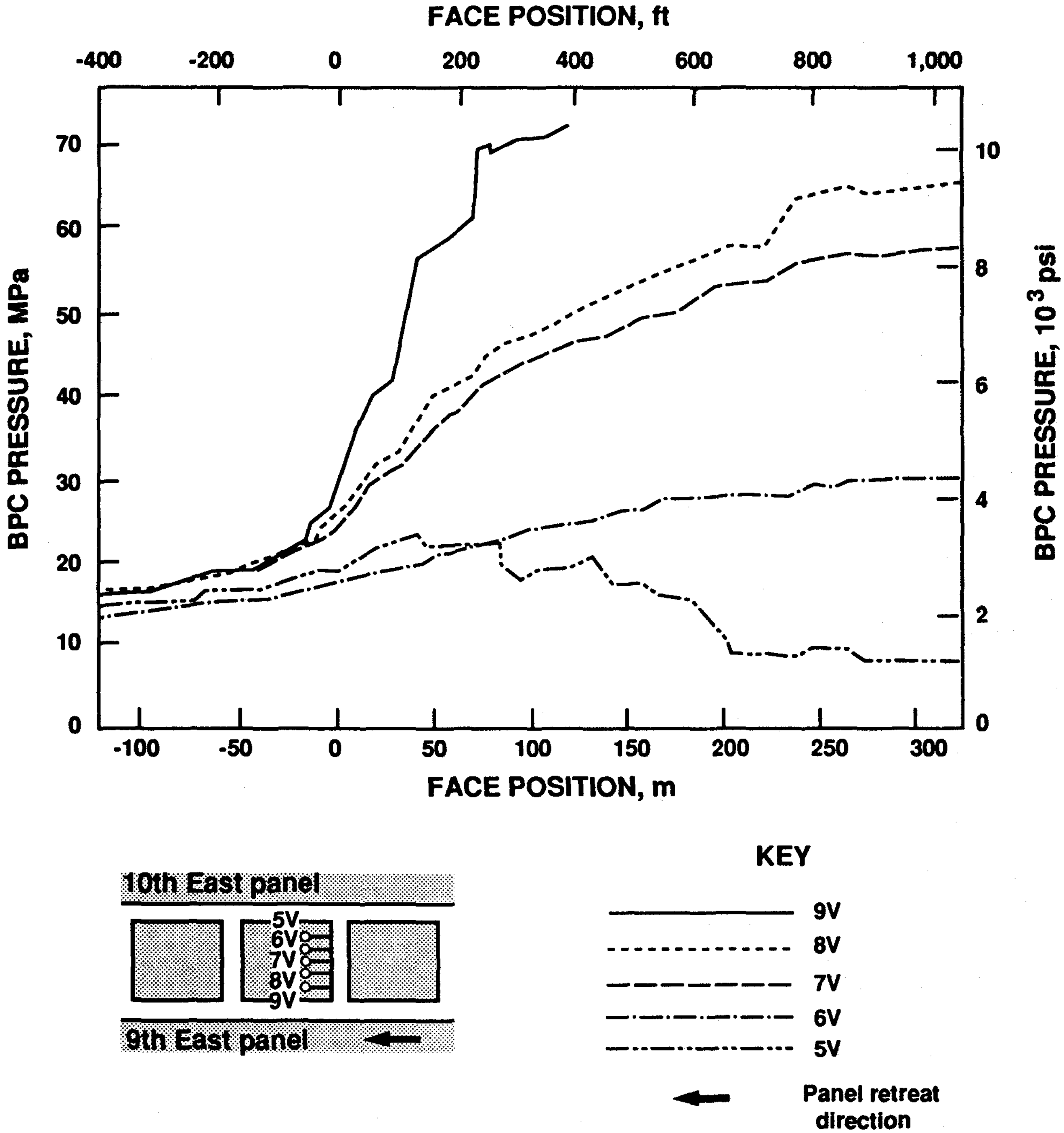
Figure 56 profiles the load configurations across the midsite pillar and adjacent panel ends for several face positions and clearly shows the high-magnitude asymmetrical loading across the pillar and load carryover onto the 10th East panel.

Figure 52



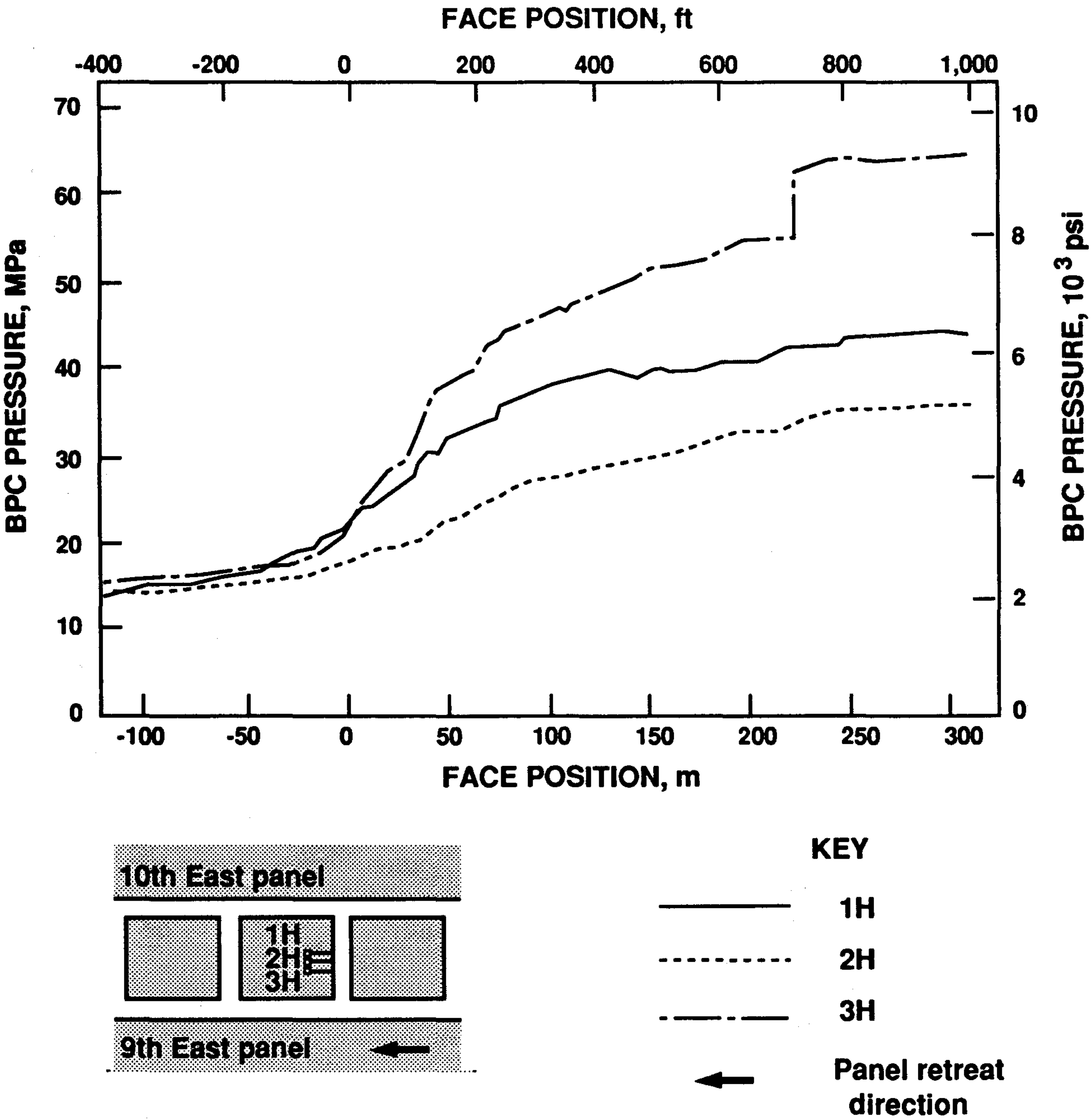
Study site instrumentation in 9th East gate road, Castle Gate No. 3 Mine.

Figure 53



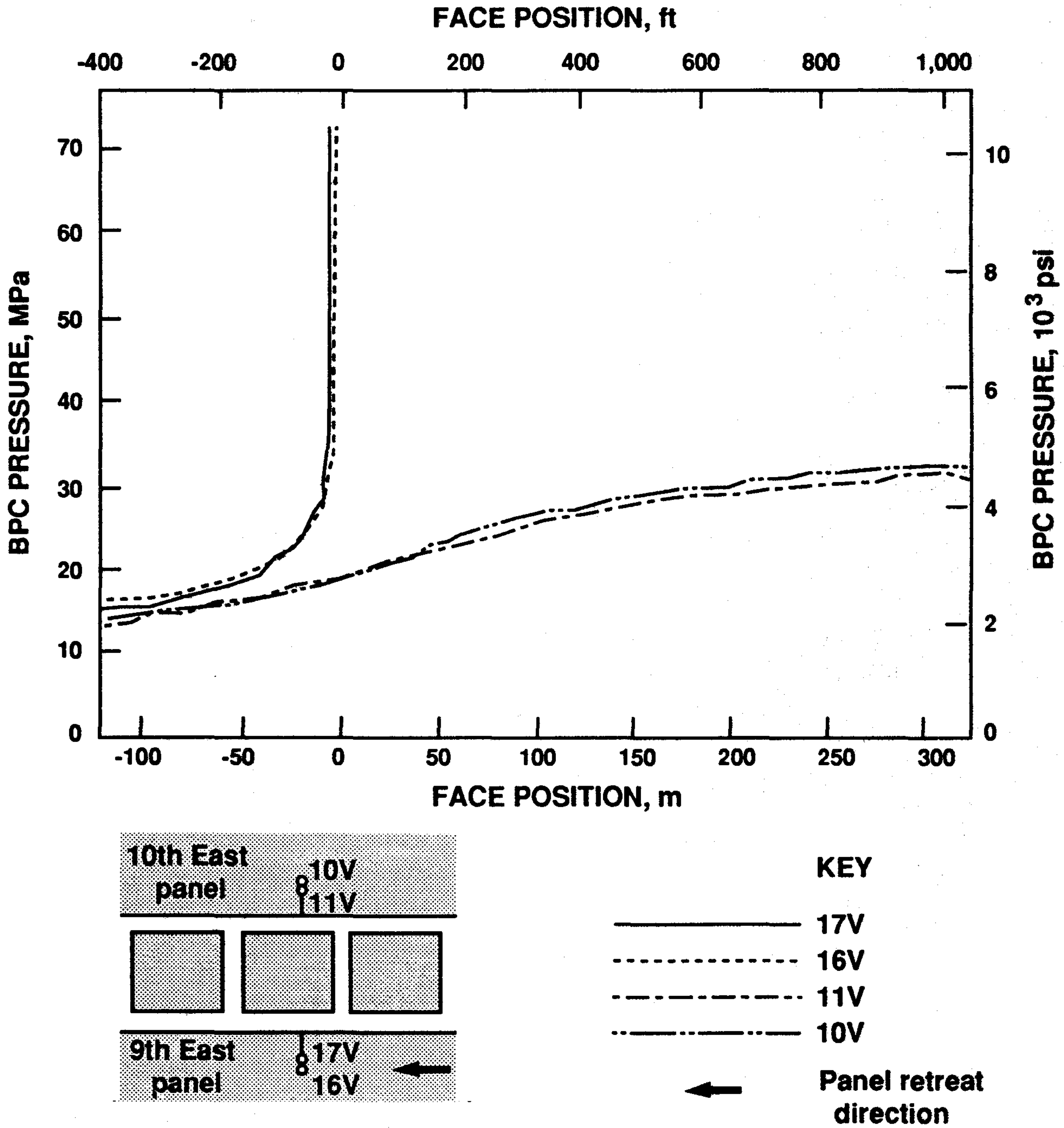
Vertical pressure relative to 9th East panel face position at midsite pillar, Castle Gate No. 3 Mine.

Figure 54



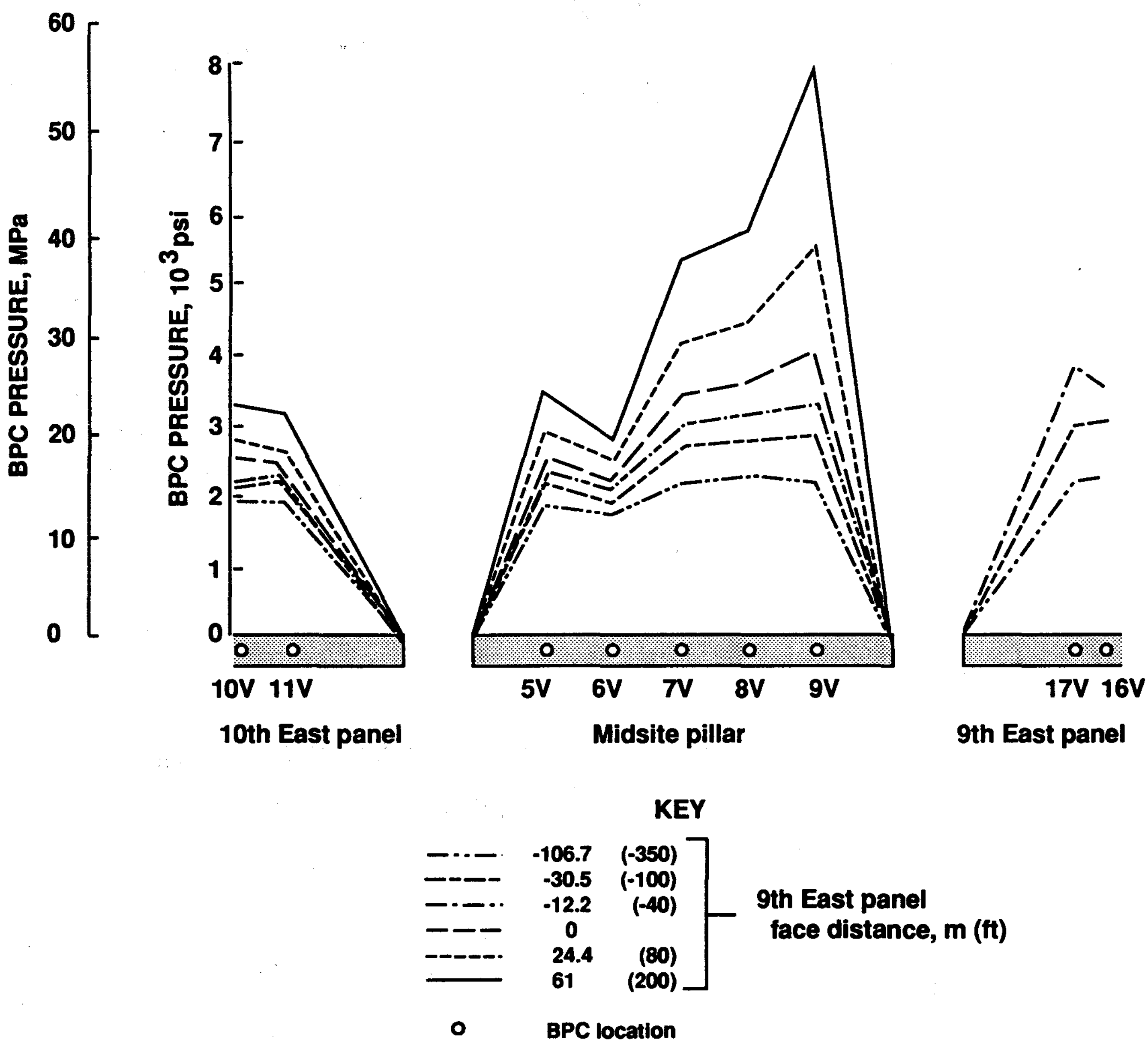
Horizontal pressure relative to 9th East panel face position at midsite pillar, Castle Gate No. 3 Mine.

Figure 55



Vertical pressure relative to 9th East panel face position at 9th and 10th East panel ribs, Castle Gate No. 3 Mine.

Figure 56



Vertical pressure profiles at midsite pillar, Castle Gate No. 3 Mine.

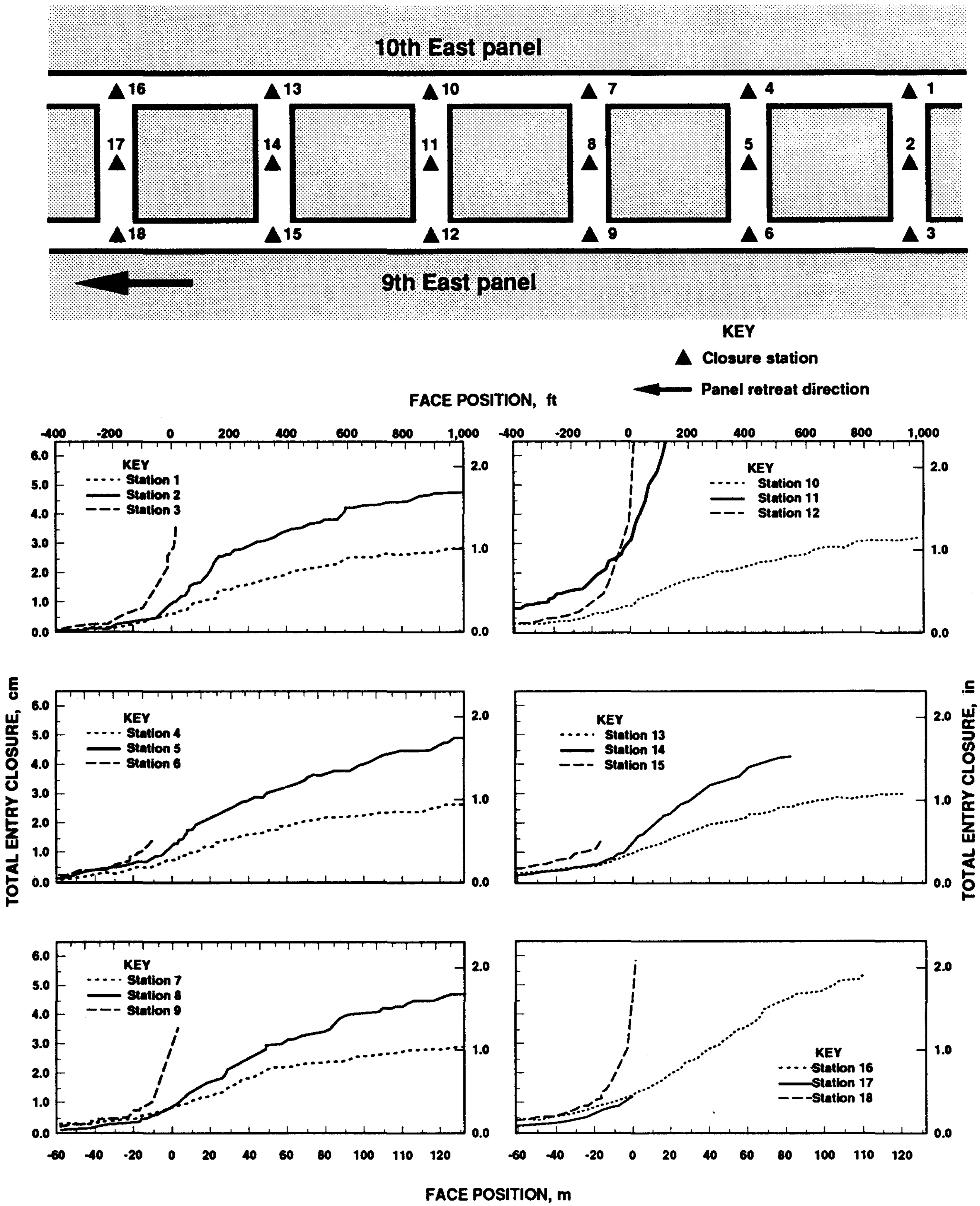
Entry Closure Analysis

Because of the thickness, high compressive strength, and rigidity of the floor sandstone, it was reasonable to infer that entry closure was mostly attributable to deformation of the roof and/or the pillars supporting the roof. Although monitoring at several closure stations (figure 57) was prematurely discontinued because of roof slabbing and falls, and passage of the face destroyed the stations in the entry nearest the active panel, the closure data (figure 58) indicate the same overall trends as the load cell data—the stations closest to the active 9th East (sixth) panel experienced earlier closure onset, higher closure rates, and greater total closure than the stations nearer the 10th East panel. Sudden changes in closure rate correlated well with observed roof deterioration and slabbing.

Figure 57

Castle Gate Coal Co. engineer measuring roof-to-floor convergence at closure station in 9th East gate road.

Figure 58



Entry closure relative to 9th East panel face position, Castle Gate No. 3 Mine.

CASTLE GATE NO. 3 MINE CONCLUSIONS

A summary of the Castle Gate gate road designs and their relative performance is presented in table 1 (*47*). In the absence of quantified data for assessment of gate road performance in the first five panels, mine personnel and geotechnical consultants could utilize only a success or failure criterion to evaluate each successive design. This trial-and-error approach did not expedite alleviation of the bumping problem, and possibly contributed to the major reserve losses incurred during these design attempts. The history of tailgate pillar bumping in the first four panels clearly indicates that a true yield-pillar design had never been achieved (with the possible exception of the 6.1-m (20-ft) 5th East production pillars). The small tailgate pillars were too greatly confined to yield, yet too small to support full abutment loading without failing violently. Calculations using Wilson's confined core pillar design method (*48-49*) indicated that the required width for a pillar to yield in the rigid roof and floor conditions of the mine was approximately 4 m (13 ft).

The 25.9-m (85-ft) wide 8th East pillars bumped violently during tailgate loading when cover depth exceeded 486 m (1,600 ft). Although the 36.6-m (120-ft) wide 9th East pillars under 486- to 669-m (1,600- to 2,200-ft) cover remained stable in headgate loading, the high loads and rapid load responses observed in these pillars indicate that they probably would have bumped if the next (10th East) panel had been extracted. Also, the 9th East panel side abutment loads overriding these pillars onto the 10th East panel, in addition to the probable concentrated forward abutment load in the panel during extraction, could have led to face bumps and outbursts near the tailgate. Larger chain pillars and/or barrier pillars between panels might have been required to preclude violent tailgate pillar bumping and to isolate successive panels from side abutment loading attributable to the preceding panel.

Table 1.—Longwall mining history at Castle Gate mines

Year	Mine and panel	Pillar size, m (ft)				Number of entries		Results
		Headgate		Tailgate		Headgate	Tailgate	
1975 ..	No. 3 Mine, 3rd East.	12.2 by 25.9	(40 by 85)	15.2 by 25.9	(50 by 85)	3	3	Tailgate pillars failed. Bumps were commonplace.
1977 ..	No. 3 Mine, 4th East.	9.1 by 25.9	(30 by 85)	12.2 by 25.9	(40 by 85)	3	3	Violent bumps in tailgate and on face.
1979 ..	No. 3 Mine, 5th East.	9.1 by 25.9	(30 by 85)	9.1 by 25.9	(30 by 85)	3	3	Violent bumps in tailgate and on face. Panel abandoned because of equipment failure.
1980 ..	No. 3 Mine, 6th East.	9.1 by 25.9	(30 by 85)	9.1 by 25.9	(30 by 85)	2	3	Violent bumps as in 4th and 5th East. Panel was ended prematurely because of a fire.
1981[1] ..	No. 5 Mine, 8th West.	25.9 by 25.9	(85 by 85)	9.1 by 30.5	(30 by 100)	2	3	Better conditions; only roof spalling and small headgate bumps.
1982[2] ..	No. 5 Mine, 9th/10th West.	25.9 by 36.6	(85 by 120)	25.9 by 25.9	(85 by 85)	2	2	Minor ground control problems.
1986[3] ..	No. 5 Mine, 11th West.	25.9 by 36.6	(85 by 120)	25.9 by 36.6	(85 by 120)	2	2	Do.
1987 ..	No. 5 Mine, 12th West.	25.9 by 36.6	(85 by 120)	25.9 by 36.6	(85 by 120)	2	2	Do.
	No. 3 Mine, 8th East.[4]	25.9 by 36.6	(85 by 120)	25.9 by 36.6 15.2 by 36.6	(85 by 120) (50 by 120)	2	3	No ground control problems, although loud "booms" were heard.
1988 ..	No. 3 Mine, 9th East.	36.6 by 36.6	(120 by 120)	25.9 by 36.6	(85 by 120)	2	2	Extraction began in June 1988; severe bumping occurred in tailgate and along face.

[1]Longwall production was moved to No. 5 Mine.
[2]Mining continued using 9th West design until December 1984, at which time mine was idled.
[3]Mine was reactivated in May 1986, and 11th West longwall panel began operation in October 1986.
[4]No. 5 Mine was closed, and production switched to No. 3 Mine.

SUNNYSIDE NO. 1 MINE

LOCATION

The Sunnyside No. 1 Mine portal is located adjacent to Utah Highway 123 approximately 1.6 km (1 mile) northeast of the small town of Sunnyside (figure 59).

GEOLOGIC SETTING

Stratigraphic Units

The principal stratigraphic units of the Sunnyside area are, in ascending order, the Mancos Shale, the Blackhawk Formation, the Castlegate Sandstone, and the Price River, North Horn-Flagstaff Limestone, and Colton Formations (*25*) (figure 60). Although Young (*17*) performed the primary classification of the Blackhawk Formation in the Book Cliffs region, subsequent descriptions of Sunnyside area stratigraphy (*25, 50*) have followed the classification established by Maberry (*51*), who redefined two of Young's members on the basis of lithology and unit mapability. According to Maberry, four Blackhawk members are present in the Sunnyside area: the Kenilworth, a Lower Mudstone, the Sunnyside, and the Upper Mudstone (figures 60-61). The units significantly affecting Sunnyside ground conditions are the Sunnyside and Upper Mudstone Members of the Blackhawk Formation and the Castlegate Sandstone, together with the balance of the overlying strata.

Sunnyside Member

The Sunnyside Member, which is 30 to 58 m (100 to 190 ft) thick in the Sunnyside area, is composed of interbedded sandstone and siltstone at the base, grading upward through fine-grained sandstone to thick-bedded, medium-grained sandstone at the top (*25*). The 30- to 46-m (100- to 150-ft) thick upper sandstone (Young's "Sunnyside Sandstone") constitutes the near-immediate floor of the Sunnyside Mines. In areas where the sandstone directly contacts the base of the coalbed, floor heave is negligible, and minimal sloughage at the lower portion of the ribs indicates some degree of basal pillar confinement.

Upper Mudstone Member

Maberry (*51*) described the Upper Mudstone Member as consisting of "a basal thin discontinuous sandy siltstone, overlain by the thick Sunnyside coal bed and 250-300 feet of dark laminated shale and massive mudstone with lenticular bodies of sandstone and siltstone throughout."

Basal Siltstone

The thin discontinuous siltstone lying between the Sunnyside Sandstone and the Sunnyside Coalbed makes up the immediate floor throughout much of the mining area. Where high-magnitude loads are conveyed directly to the floor through nonyielding large pillars, the siltstone often either shears and/or separates from the underlying sandstone, then shifts laterally away from the pillar or buckles near the center of adjacent openings. In either case, the result is significant floor heave in nearby entries (figure 62).

Sunnyside Coalbed

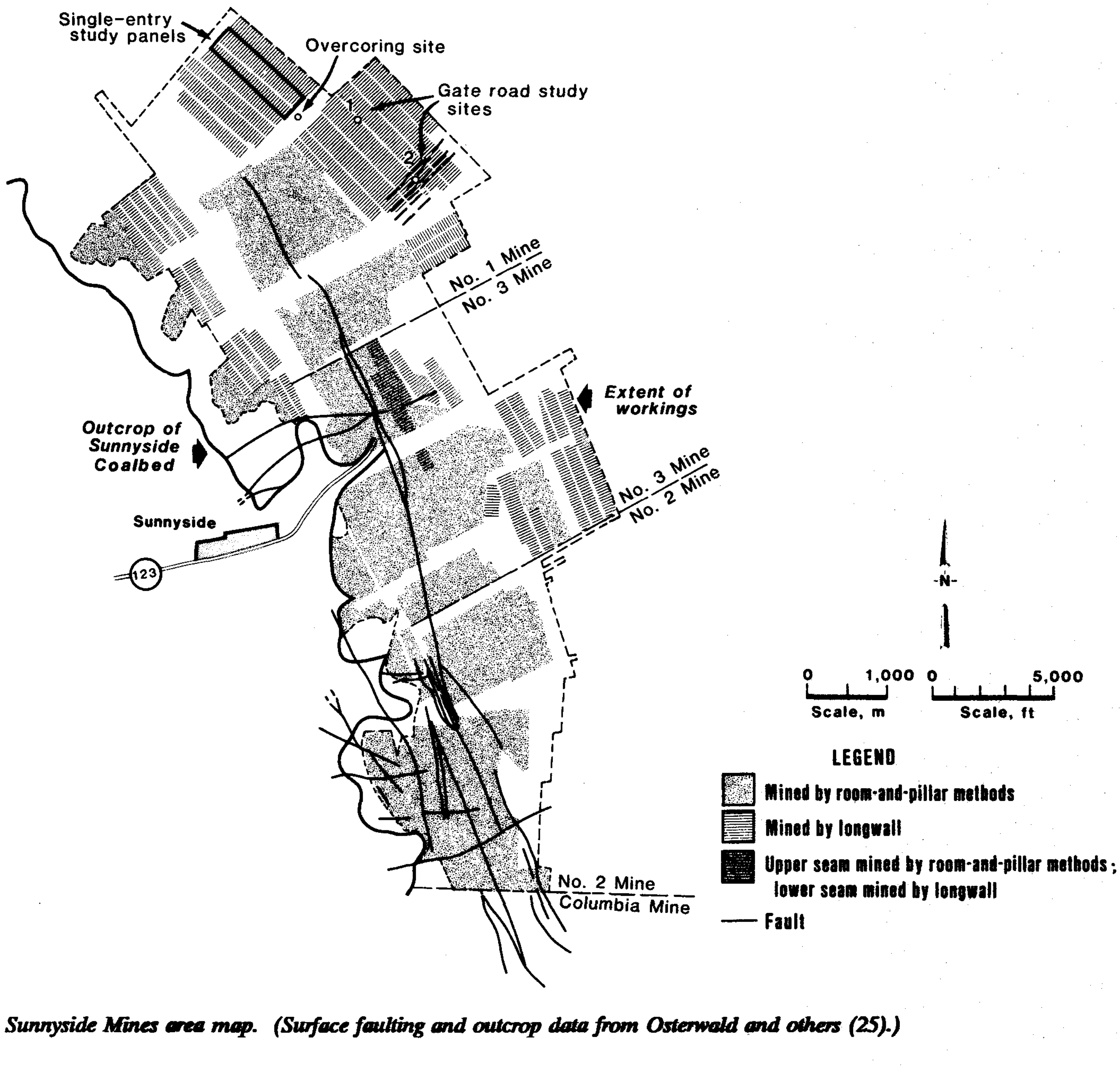
The Sunnyside Coalbed is as much as 7 m (24 ft) thick in a single unit, and it is commonly separated into two minable seams by a split that ranges in thickness from 0 to 23 m (75 ft) (*25*). Because of the thickness of the split, especially in the older workings near the outcrop(s), miners and early geologic investigators believed the coal to be two different beds, termed the "Upper and Lower Sunnyside Seams." Detailed surface mapping and correlation of underground sections, however, later established that the seams are splits of the same bed (*51*). In the area of the Sunnyside No. 1 Mine (figure 59), the lower seam is 1.25 to 2.75 m (4 to 9 ft) thick, while the upper seam is generally 1.0 to 2.2 m (3 to 7 ft) thick, occasionally thickening to 2.75 m (9 ft) (*26*). Although both seams can be characterized as hard coal, the lower seam is usually unfractured, forming competent ribs, whereas the upper seam is often vertically shattered into thin, irregular fragments, particularly on pillar corners, where the fractures extend nearly the full seam height. When the two seams are mined together, their characteristic appearance is generally maintained (figure 63) (*50*).

In the northern part of the No. 1 Mine, the rock split between the upper and lower seams varies from 0 to 11 m (35 ft), averaging about 1.8 m (6 ft), whereas in the southern part of the mine the split is not present (*26*). In areas where the rock split thickness is 1.8 m (6 ft) or less, both seams and the rock were sometimes mined together when primary entries were developed or old haulageways were rehabilitated, particularly in the older workings (*7*). Only the lower seam is extracted during panel mining, unless the rock split is absent, in which case the entire bed thickness is mined to the full machine cutting height, except for top coal left to maintain roof stability.

Although only the lower seam (or both seams together) is mined in the No. 1 Mine, both seams have been worked separately throughout the No. 3 Mine and in parts of the No. 2 Mine, entailing the ground control problems commonly associated with multiple-seam mining.

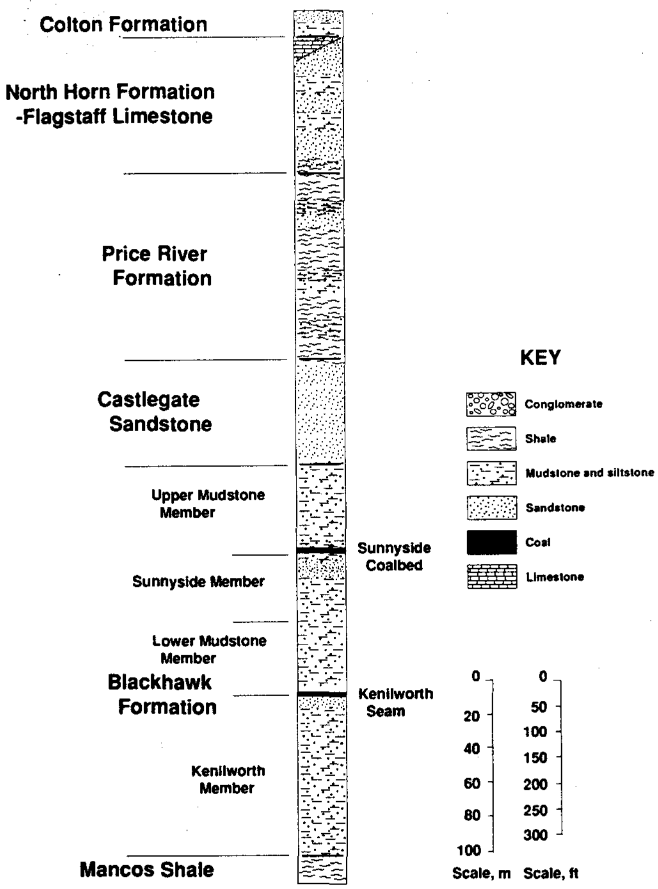
In addition to the two main seams, the Sunnyside bed includes several unminable thin seams at various locations throughout the locality. In the northern part of the No. 1 Mine, the mined seam is underlain by a small seam less than 1 m (3 ft) thick (*26, 52*) and is overlain by a persistent rider seam approximately 1 m thick throughout most of the longwall area (*26*).

Figure 59



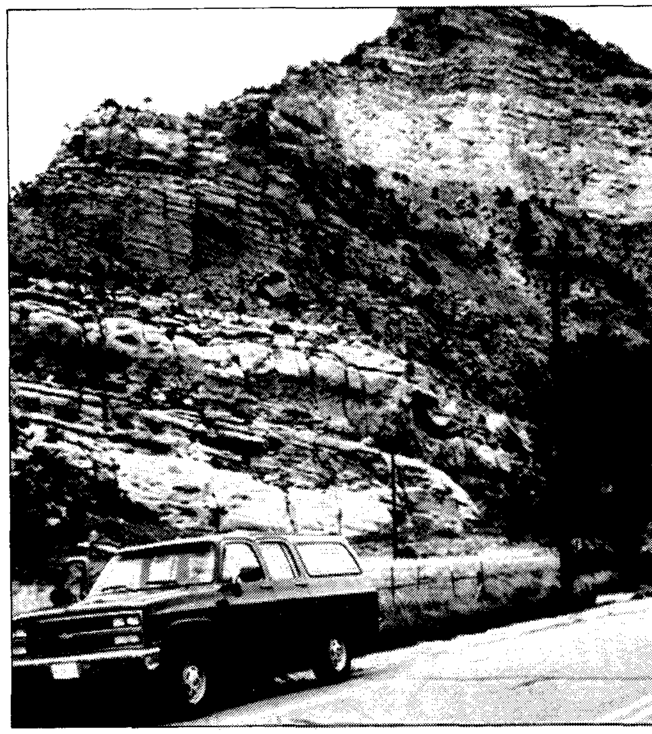
Sunnyside Mines area map. (Surface faulting and outcrop data from Osterwald and others (25).)

Figure 60



Stratigraphic column, Sunnyside No. 1 Mine.

Figure 61



Book Cliffs strata at Sunnyside No. 1 Mine. Mancos Shale lies at foot of escarpment (immediately above open door of vehicle). In Sunnyside area, Blackhawk Formation is composed of four members: the Kenilworth, which forms two prominent light-colored sandstone layers directly above vehicle roof; Lower Mudstone, which makes up dark, thinly bedded cliff between second and third sandstones; Sunnyside, which is third prominent sandstone and forms low cliff and ledge at left skyline; and Upper Mudstone, which makes up intermediate slope extending upward to foot of high, light-colored cliff formed by Castlegate Sandstone. Thinly bedded lower Price River Formation strata form apex of escarpment. Sunnyside Coalbed lies directly upon Sunnyside Member.

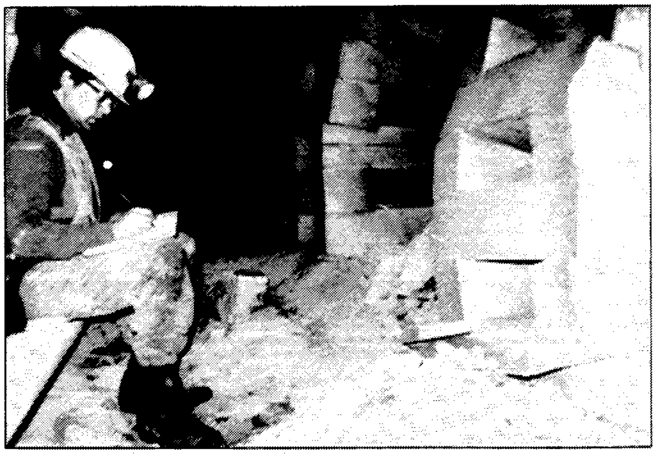
Lower Portion of the Upper Mudstone Member

The lower part of the Upper Mudstone Member forms the immediate roof of the Sunnyside No. 1 Mine. As shown where exposed by roof caving or by core drilling of the roof, the 9 m (30 ft) of strata immediately above the mine seam consists primarily of dark-brown and gray-brown siltstone and mudstone, interbedded siltstone and/or sandstone, and quartzose and calcareous siltstone and/or sandstone (figure 64). The extremely stiff, dark-brown siltstone generally occurs immediately above the mine seam and varies in thickness from several centimeters to several meters. The gray-brown sandy siltstone, which varies in thickness from 0.3 to 2.4 m (1 to 8 ft), is usually found in lenses above the mine seam and is also extremely stiff. Interbedding of siltstone and sandstone occurs frequently throughout the near-immediate roof. The fine-grained quartzose and calcareous sandstones occur irregularly as lenses or channel-fill deposits (*50*, *52*). Although broad channel sandstones are present in sections of the No. 1 Mine, they are not as prevalent, nor do they apparently affect ground conditions to as great a degree, as in the mines of the westernmost Book Cliffs and Wasatch Plateau (*26*).

Castlegate Sandstone and Overlying Units

The upper portion of the Upper Mudstone Member, the Castlegate Sandstone, and the overlying Price River, North Horn-Flagstaff Limestone, and Colton Formations together make up the main roof of the Sunnyside Mines. In the No. 1 Mine area, the Castlegate Sandstone is approximately 61 m (200 ft) thick, and it lies about 46 m (150 ft) above the mine seam (*25*, *50*). Mine personnel have believed that loading and failure of this massive fine-grained sandstone created widespread disturbances on

Figure 62



Floor heave and rolled-out cribs in Sunnyside No. 1 Mine gate road. Loads transferred to floor by oversized chain pillar (to right of photograph) have caused basal siltstone (immediate floor) to separate from underlying rigid sandstone and move laterally toward location of USBM engineer, shifting lower portion of cribs. Floor heave at base of cribs probably resulted from separated siltstone strata overriding intact floor beneath engineer's feet.

active mining sections around the peripheries of mined-out areas (*53*). The depth of cover above the No. 1 Mine ranges from 0 at the outcrop along the Book Cliffs escarpment to nearly 912 m (3,000 ft) over the farthest inby workings. Because of the cliff-forming ridge-and-canyon topography, cover depth on active workings can vary rapidly within a relatively short lateral distance.

Structural Features

The Sunnyside Coalbed dips to the northeast at 5° to 15°, with the steeper dips occurring near the outcrop. It is interesting that many of the extensive northwest-trending faults in the area occur where the dip lessens.

Sunnyside Fault Zone *(25)*

Most of the north-northwest-trending faults in the Sunnyside area are part of the Sunnyside fault zone, which extends from approximately 2 km (1.25 mile) south of the abandoned Columbia Mine to the northern workings of the Sunnyside No. 1 Mine (figure 59). Near the Columbia Mine the zone is as much as 2.4 km (1.5 mile) wide, but it is only 3 m (10 ft) wide in the Sunnyside No. 1 Mine. Individual faults within the zone dip steeply. Average stratigraphic separation is about 9 m (30 ft) in the northern part of the area at the Sunnyside No. 1 Mine, and about 12 to 18 m (40 to 60 ft) in the southern part of the area at the Columbia Mine. Most faults within the zone are nearly parallel to the trend of the zone, but some diverge at small angles and merge with other faults in the zone that are parallel with the zone boundaries. The map relationships of the faults within the zone suggest that the diverging faults may be gash fractures resulting from small components of horizontal movement in which the block northeast of the zone moved relatively northwestward. Thus, the total displacement across the zone may be much greater than the 9 m (30 ft) of stratigraphic separation. At a few localities, strongly fractured zones in the Castlegate Sandstone show no stratigraphic separation and may have resulted from predominantly horizontal motion.

The zone varies widely in its internal characteristics. At some localities in the northern part of the area, no single fault plane is present in the zone, and the separation is distributed along numerous minute fractures across a width of several meters. Elsewhere, as in the main slope of the Sunnyside No. 1 Mine, the position of the zone is marked by gouge, breccia, and fractured rock. In the southern part of the area, where the zone is broad, it consists of widely spaced individual faults.

East-Northeast- to Northeast-Trending and East-Trending Faults *(25)*

Faults that trend east-northeast and east are widely distributed throughout the district but are more numerous near the Columbia and Sunnyside No. 2 Mines (figure 59). These faults trend about parallel to the dip of beds but vary considerably in strike and dip. Most of these faults dip steeply in the northern part of the area but dip less steeply farther south, and at the Columbia Mine they dip 45° or less. East-northeast-trending faults offset faults of the Sunnyside fault zone and hence may be younger than the zone, both in the Sunnyside No. 1 Mine area and in the Columbia area. Separation is less along these faults in the Colton Formation than it is in older rocks, indicating either that movement began during Cretaceous time and continued with decreasing energy into early Tertiary time, or that the fault separation diminished to the east. Stratigraphic separation on individual east-northeast-trending faults is as much as 45.6 m (150 ft) at the Book Cliffs escarpment, but on most faults it is much less. Separation on many of the faults is greatest near the cliffs but decreases eastward away from the cliffs. Movement on most of the east-northeast- and northeast-trending faults apparently was dip slip.

Effects of Faults on Mining Conditions and Hazards

Faulting has had a major effect on mining ground conditions, hazards, and layouts since the earliest days of coal operations in the Sunnyside area. As early as 1915, it was observed that the presence of major fault planes probably

Figure 63

Sloughed rib in Sunnyside No. 1 Mine gate road. Original rib line is seen at roof brow to left of wire mesh. Upper portion of composite seam has fractured and fallen away from roof, covering relatively intact lower part of seam with loose coal.

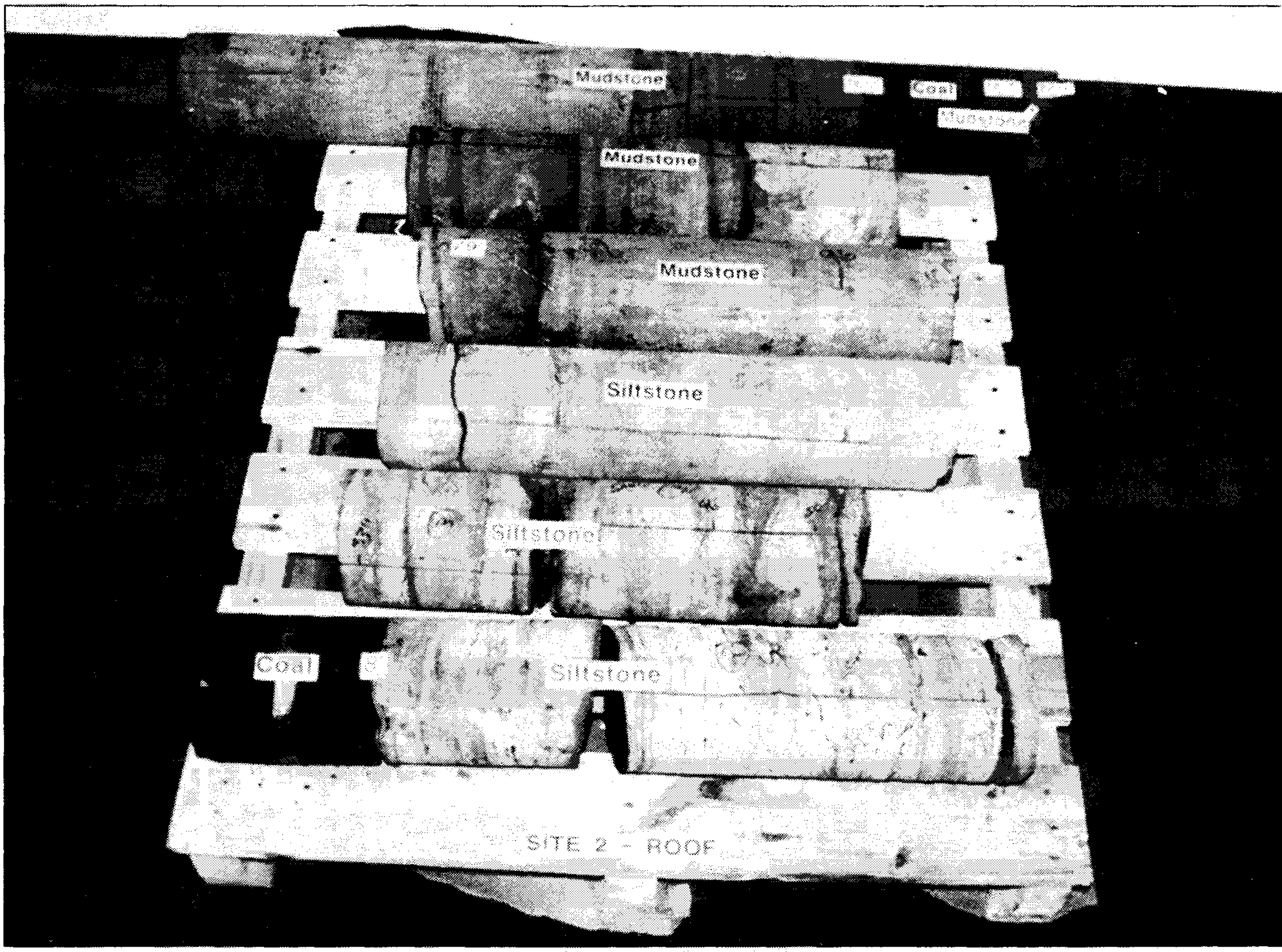
contributed to the occurrence and severity of bumps in the Sunnyside No. 2 Mine (*2*). Roof falls and bumps have long been associated with workings in the Sunnyside fault zone (*54*). Several investigators have noted that many of the most severe bumps occurred in workings that diverged at large angles from faults, whereas workings that intersected faults at small angles were less subject to bumps and required less roof support (*55-56*). Although the longwall panels of the gate road study area in the Sunnyside No. 1 Mine, described in this publication, were a considerable distance downdip from the Sunnyside fault zone (figure 59), the tectonic forces that probably caused the faulting (*25*) may still be active (*56*) and may contribute to adverse ground conditions in the study area (figure 65). Stress-relief overcoring was conducted by the USBM in 1973 approximately 912 m (3,000 ft) west of the future longwall gate road study sites (figure 59). This overcoring indicated major and minor horizontal principal stresses of, respectively, 25.6 and 20.0 MPa (3,718 and 2,898 psi) (*45*) in excess of the approximately 2.1-MPa (300-psi) horizontal stress attributable to overburden loading through Poisson's effect at the 322-m (1,060-ft) depth of the overcoring site (assuming 25 kPa vertical stress per meter of depth (1.1 psi/ft) and an average Poisson's ratio of 0.2 for the overburden strata). The orientation of the major horizontal principal stress, N. 31° W. (*45*), nearly parallels the northwest-trending faults of the Sunnyside fault zone.

SUNNYSIDE NO. 1 MINE OPERATING HISTORY

Room-and-Pillar Mining

The Sunnyside mines were originally opened in 1896 and were operated by the Utah Fuel Co. until 1942, when Kaiser Steel Corp. leased the No. 2 Mine; Kaiser Steel

Figure 64



Immediate roof core from northeast part of Sunnyside No. 1 Mine. Coal in first 20 cm (8 in) in foreground is uppermost part of Sunnyside Bed. Stiff gray-brown siltstone directly contacts coal at bottom and grades upward into stiff dark-brown mudstone at about 2 m (6.6 ft) into roof. A small 17-cm (7-in) coal band terminates mudstone at 4 m (13.3 ft) above mine seam.

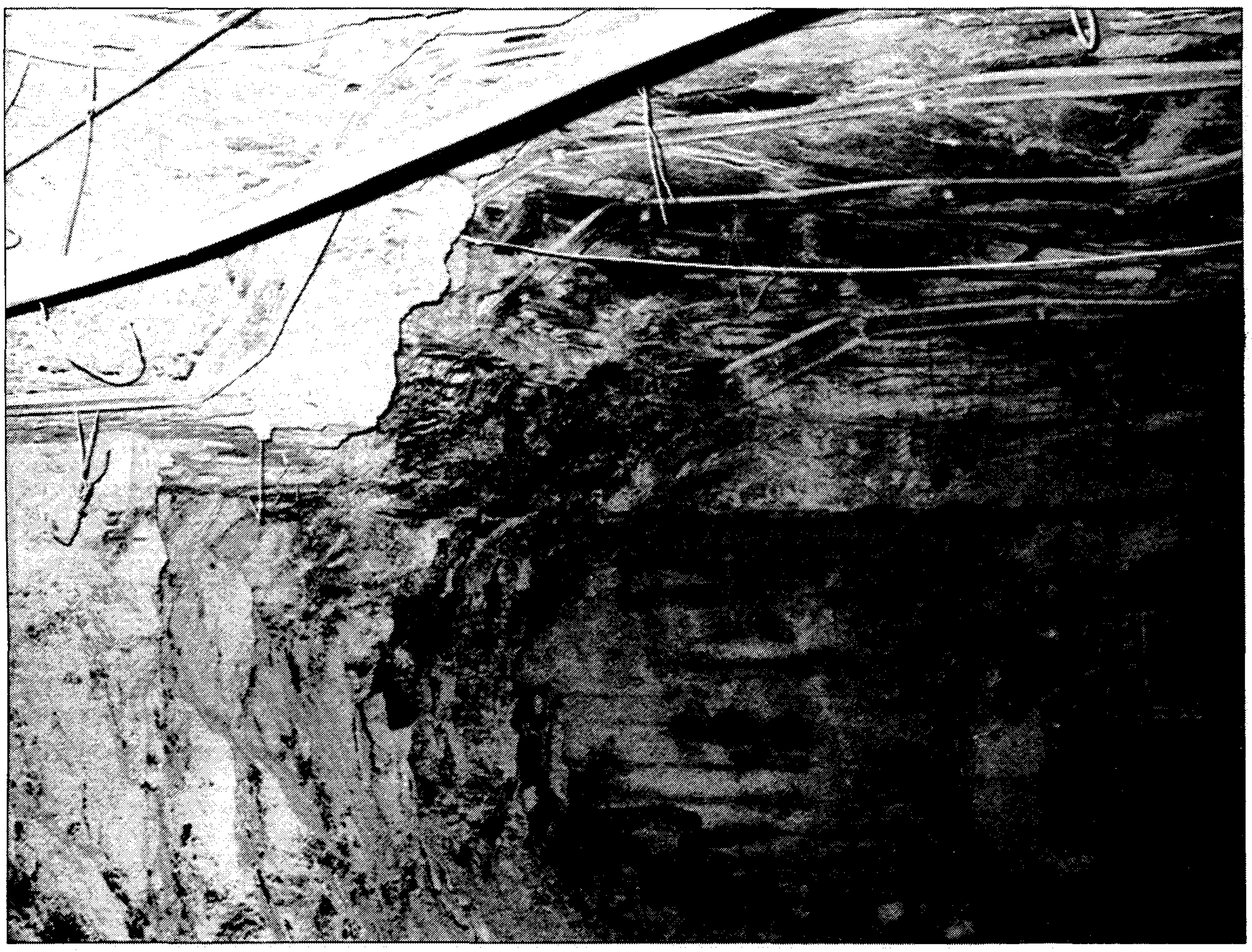
purchased the entire property in 1950 (*57*). Prior to the introduction of longwall mining in 1961, room-and-pillar mining was practiced, first using conventional equipment and later utilizing continuous miners. Rising costs, poor roof, bumps, and interaction between the workings in the closely lying upper and lower seams led to the initial longwall trials in the No. 3 Mine in 1961 (*8, 58-59*).

Longwall Mining

Since the introduction of the first longwall in 1961, longwall mining has increasingly accounted for the Sunnyside Mines production; continuous mining has concentrated on the development of main entries and longwall gate roads. The Sunnyside Mines "pioneered" the use of longwall in the western United States, introduced the use of yielding chain pillars, and were the site of advancing tailgate and single-entry experiments.

The initial longwall was set up in the No. 3 Mine. The 93.6-m (308-ft) wide face retreated 142.9 m (470 ft); the face was extended to 127.7 m (420 ft) and retreated 165.7 m (545 ft) to the barrier pillar. The initial panel experienced bumps, two especially large, when retreating under the upper seam pillar remnants (*8, 58*). Upper seam pillar failures also resulted in irregular roof action and weight distribution and in some cases, pillar punching into the lower workings. The violence of the bumps was apparently related to the size and location of the pillar remnants; the entries experienced roof problems when the overlying pillars were in line with the entries (*9, 60*). The early faces also experienced problems when mining was under sandstone; hanging roof resulted in excessive

Figure 65



Domed roof fall area in Sunnyside No. 1 Mine near gate road study area. Besides poor roof conditions, horizontal stresses often contribute to occurrence of domed falls.

support loads and, in some cases, the relatively low-capacity two-legged chocks became rigid, incapable of being raised or lowered. Failure of the sandstone bed often resulted in large methane inflows, shattered roof, and floor heave, which required costly entry cleanup and resupport (*10*).[4] The second and third panels, 96.7 m (318 ft) wide by 624.7 m (2,055 ft) long and 142.8 m (470 ft) wide by 775.2 m (2,550 ft) long, respectively, also experienced bumps and roof problems due to the upper seam pillars (*57*). The 197.6-m (650-ft) wide fourth panel was the first to be installed in the upper seam. This panel used a two-entry system driven on 12.2-m (40-ft) centers. Chain pillars, 5.5 m (18 ft) wide by 26.1 m (86 ft) long, were sized so that lower seam mining would not encounter high-pressure areas (*10, 61*). Following the mining of nine panels, a higher capacity face was installed in a panel of the No. 1 Mine. A very soft floor resulted in the early abandonment of this panel.

Two-Entry Gate Roads

Two-entry development began in room-and-pillar panels during the 1950's with the introduction of continuous mining machines (*62*). One entry was used for intake and track, and the other for the return. This technique was used to reduce the amount of ground opened up and to reduce bump frequency and severity. Once development was completed and pillaring was begun, a third entry was started at the inby end of the panel and kept one crosscut ahead of the room-and-pillar work. Rooms turned off

[4]Peparakis, J. (Kaiser Steel Corp.). Recent Experience in Longwall Mining at Sunnyside Mines. Unpublished report, 1970, 10 pp.; available from L. R. Barron, U.S. Bureau of Mines, Denver, CO.

from the third entry prevented disruption of the haulage and provided shuttle car access to the mine cars.

Because the early three-entry gate road systems in both the No. 1 and No. 3 Mines experienced bumps and roof stability problems during longwall retreat (*61*), the fourth longwall panel used a two-entry system with pillars on 12.2-m (40-ft) centers (*10*). This panel retreated without experiencing the bump problems that plagued the previous three panels. Since this design apparently contributed to a reduction in bumps and entry stability problems, the two-entry system was utilized in all but one of the subsequent panels (with the exception of single-entry experimental panels). The only design modifications have been first to widen the chain pillars to 12.2 m (40 ft), then to narrow them to 9.1 m (30 ft); crosscuts remain on 30.5- to 45.7-m (100- to 150-ft) centers (averaging about 38 m (125 ft)).

According to Koehler (*63*):

> The miners stated that there were no noticeable differences in performance of the 40-ft-wide design and the 30-ft-wide design when the pillars were subjected to first panel loads. The difference in performance became quite clear, however, when these two designs were subjected to full tailgate loads. The 40-ft-wide pillars seemed to bump more frequently and with greater force, while the 30-ft-wide pillars rarely bumped at all. In addition, tailgate roof stability throughout the mining cycle seemed to improve when the 30-ft-wide design was used.

Single-Entry Study

The Sunnyside single-entry study, originally begun in the No. 2 Mine, was moved to the No. 1 Mine (figure 59) following discovery of several faults and a minable overlying seam (*62, 64-65*). This study had the objective of determining whether a single entry supported at the center by a fire-resistant partition could be considered equal to or better than the two-entry system (*65*). If successful, the single-entry system would include the following advantages: (1) less ground disturbance, (2) elimination of the chain pillars, (3) elimination of crosscuts and intersections, (4) less dust creation because of less coal exposure and rib sloughage, (5) improved ventilation, (6) improved roof support, (7) reduced electrical and cable damage because of the straight line advance, (8) more secure travel ways, (9) greater extraction ratio, (10) faster development, and (11) controlled subsidence through a reduction in the subsidence gradient (*66*). The study was to investigate three major factors: ground control, ventilation, and safety practices. Particular concerns included (1) ability of the cribs to support the side abutment, (2) potential for high rib stresses, (3) bump occurrences of the ribs and whether the cribs and center partition could withstand bumps, (4) ventilation, and (5) fire containment by the center partition (*66*).

The Sunnyside single-entry study included intensive geological investigations and physical properties testing (*50, 64*). Over 1,200 instruments were installed and monitored. Based on experience gained in the original No. 2 Mine study, the instrumentation was streamlined to four types: crib load cells, roof bolt load cells, closure measurement stations and level survey stations (*67*). Two development headings were driven for the single-entry demonstration. The first heading was 1,368 m (4,500 ft) long; the first 456 m (1,500 ft) was single entry, the second 456 m was double entry, and the last 456 m was single entry. This configuration provided a comparison between the single- and two-entry systems (*62, 64-67*). The first entry was 7.9 m (26 ft) wide and was divided down the center by cribs on 2.1-m (7-ft) centers. A fireproof partition was attached to the cribs, and doors were installed every 30.4 m (100 ft). Development of the first heading required 14 months, longer than anticipated. Delays were attributed to ventilation problems, centerline crib and partition constructions, and inadequate room for the mining equipment (*62, 65*). The width of the second single-entry head-ing was reduced to 6.4 m (21 ft). Use of a bridge conveyor and greater crew experience resulted in a faster development rate (*62*).

During longwall retreat, it became apparent that the first design was inadequate to withstand forward and side abutment loads (*65, 67-68*). The single row of cribs could not support the roof, and in several cases the cribs and roof bolts failed. Stability problems and roof-floor closure up to 0.46 m (1.5 ft) necessitated "brushing" (using a continuous miner to cut and remove rock from the floor) to maintain an adequate travel way and required the installation of additional cribs, arches, steel beams, and posts. One 19-m (63-ft) long section of the first entry was supported with fiber-reinforced concrete cribs. These cribs were very successful; entry height was maintained without additional support, a clean break line was provided, and little or no roof punching occurred (*65, 69*). Structural analysis indicated that the second entry would require a stiffer center support; the entry width was reduced to 6.4 m (21 ft) and was supported along the center axis using wooden posts on 1.2-m (4-ft) centers.

The single-entry demonstration was considered a success by both the USBM and Sunnyside management in terms of reducing ground control problems (*62, 65*). Although the single-entry project demonstrated many potential advantages, there were higher costs associated with heading development and tailgate maintenance and concerns regarding partition integrity in case of a fire. These disadvantages led mine management to conclude that single-entry use, utilizing improved overall layout and partition designs and specialized drivage and haulage equipment, is probably limited to the deeper cover and poorer roof conditions that may be encountered in the future (*62*, *65*).

USBM GATE ROAD STUDY

Instrumentation

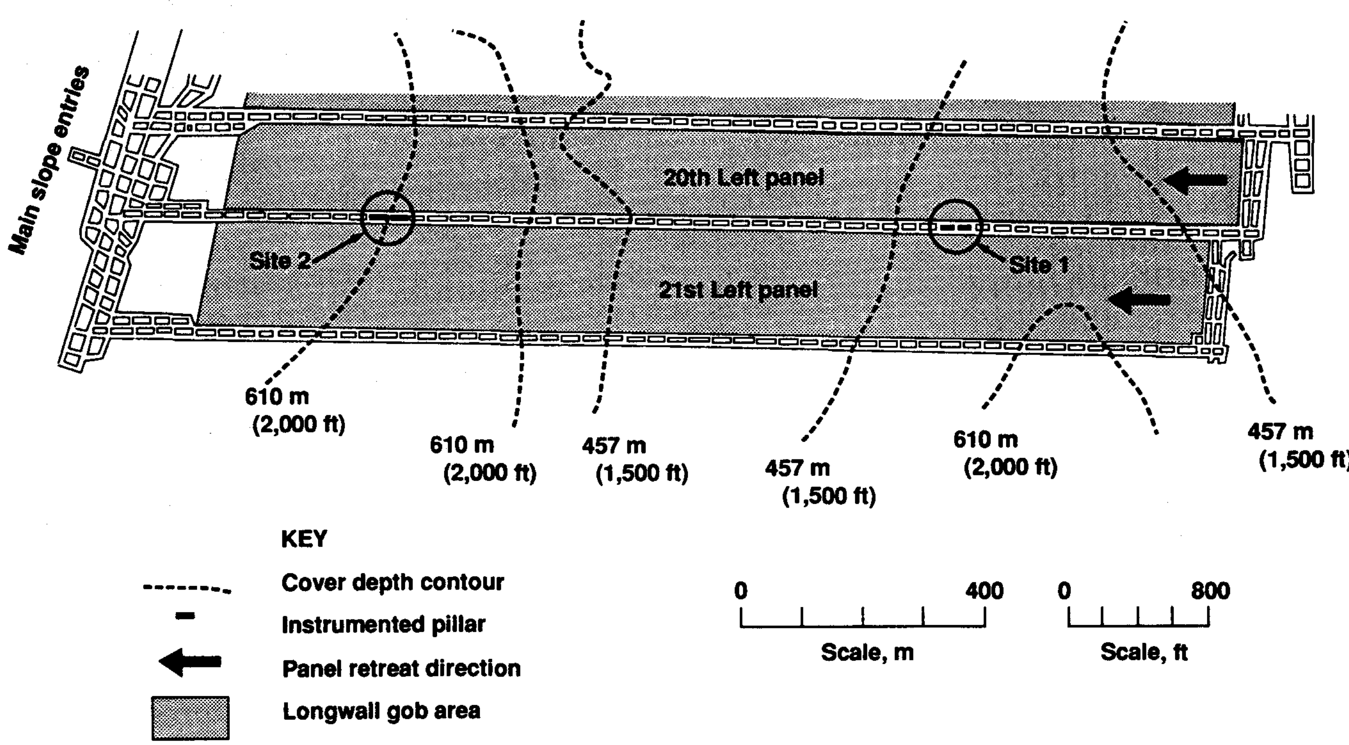
As part of a USBM research program to study the in-mine behavior of various gate road configurations, a field study was conducted at the Sunnyside No. 1 Mine. The mine was selected because of its history of bump occurrences and the apparent success in reducing bumps using the two-entry yield-pillar system. At two sites located in the 20th Left gate road (figure 66), chain pillars and adjacent panel ribs were instrumented with BPC's (oriented to measure vertical pressure) to quantify chain pillar behavior and load transfer onto the adjacent abutments (*47*, *52*).

Data Analysis and Panel Mining

Figures 67 and 68 profile load distribution across the site pillars and adjacent panel ribs at three face positions, showing BPC pressures before, during, and after passage of the 20th Left face. Data at site 1 (figure 67) showed that the chain pillar rib yielded prior to passage of the face. With passage of the face, further pillar destressing occurred, and load was transferred onto the 21st Left panel rib. The site 2 chain pillar (figure 68) indicated a more elastic response to longwall mining. With face passage, the pillar destressed, and load was transferred onto the adjacent panel rib. In contrast to the second-panel rib at site 1, the rib of the 21st Left panel also yielded following load transfer from the chain pillar (figure 69). As the face retreated further outby, additional yielding occurred (*47*, *52*).

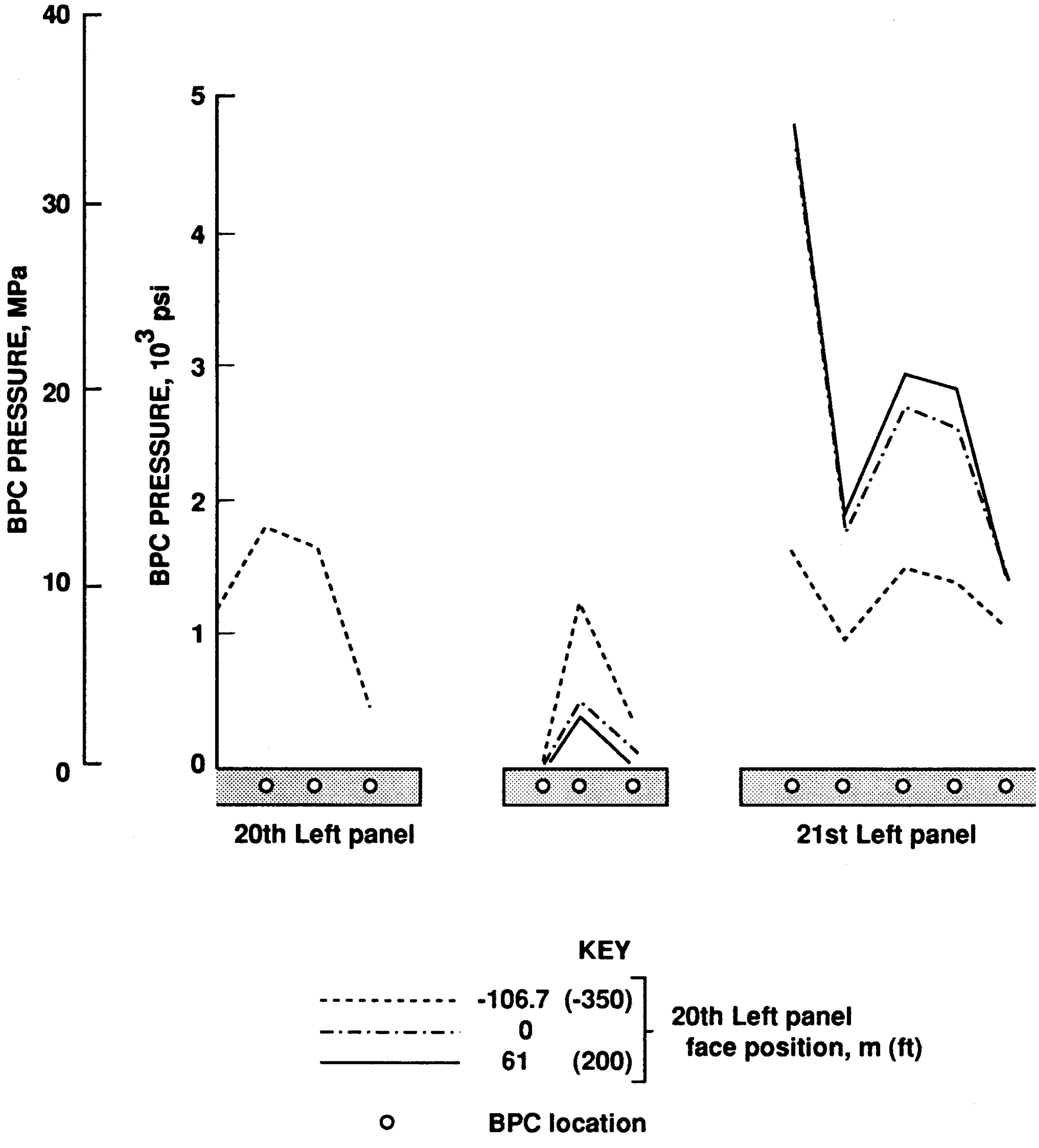
The differences between the pillar and panel responses to loading observed at site 1 and those at site 2 may be attributable to one or more factors. Site 1 was located in an area of the No. 1 Mine where the upper and lower seams are separated by a 0.6-m (2-ft) rock split, and only the lower seam is mined (*70*). In this area, which mine personnel term the "low coal zone," the mined seam height averages about 1.8 m (6 ft). At approximately one-third of the panel length from the starting room, the rock split thins to about 0.2 to 0.3 m (8 to 12 in), and both the lower and upper seams (together with the split) are mined, thus increasing the mining height to about 3 m (10 ft). Site 2 was located within this second area, usually termed the "high coal zone." Not only was the mining height at site 2 approximately 1.2 m (4 ft) greater (and the pillar width-to-height ratio about 40% less) than that at site 1, the structural characteristics of the "composite" coal seam at site 2 probably differed from those of the lower seam at

Figure 66



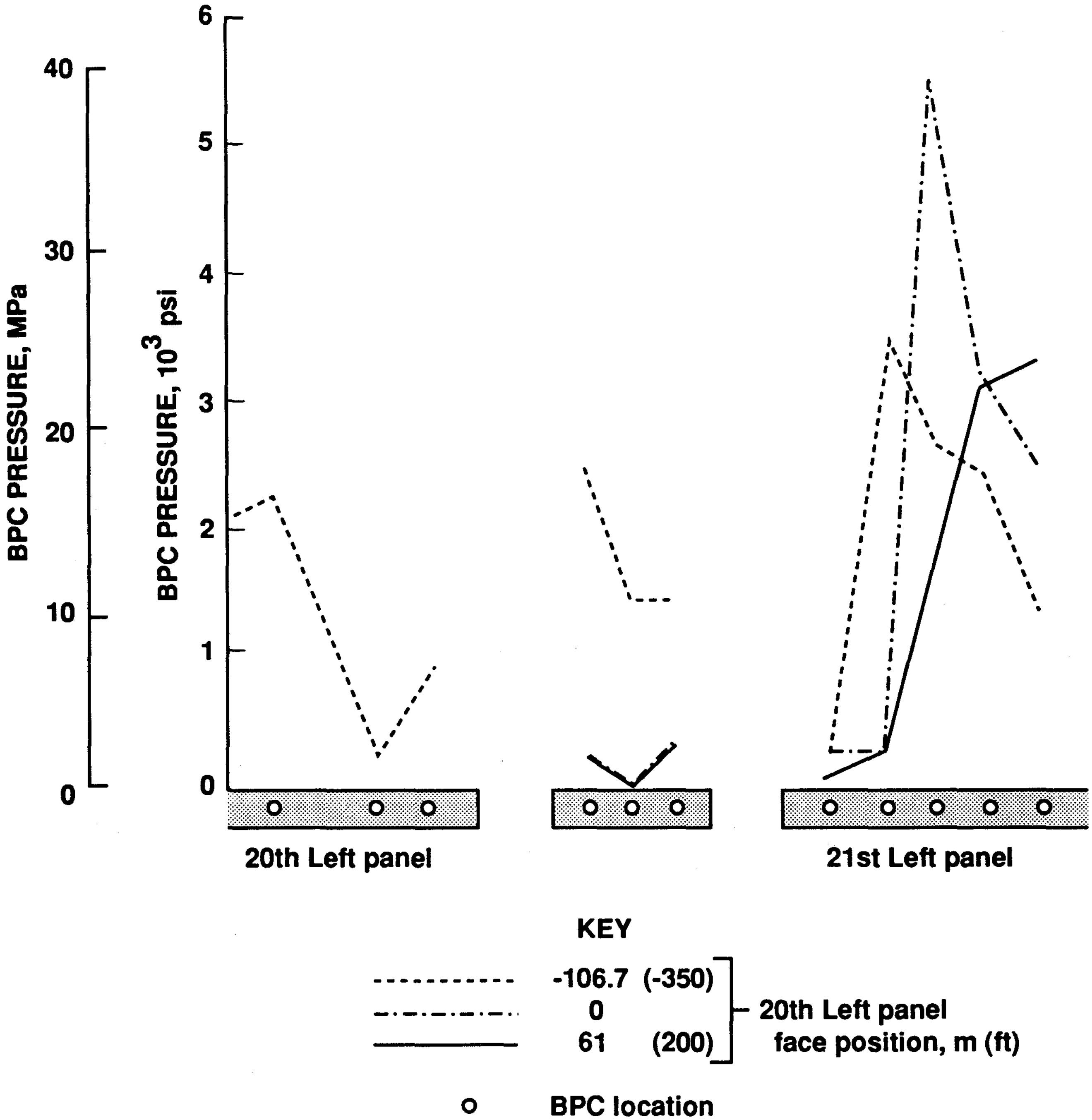
Study area longwall panels at Sunnyside No. 1 Mine.

Figure 67



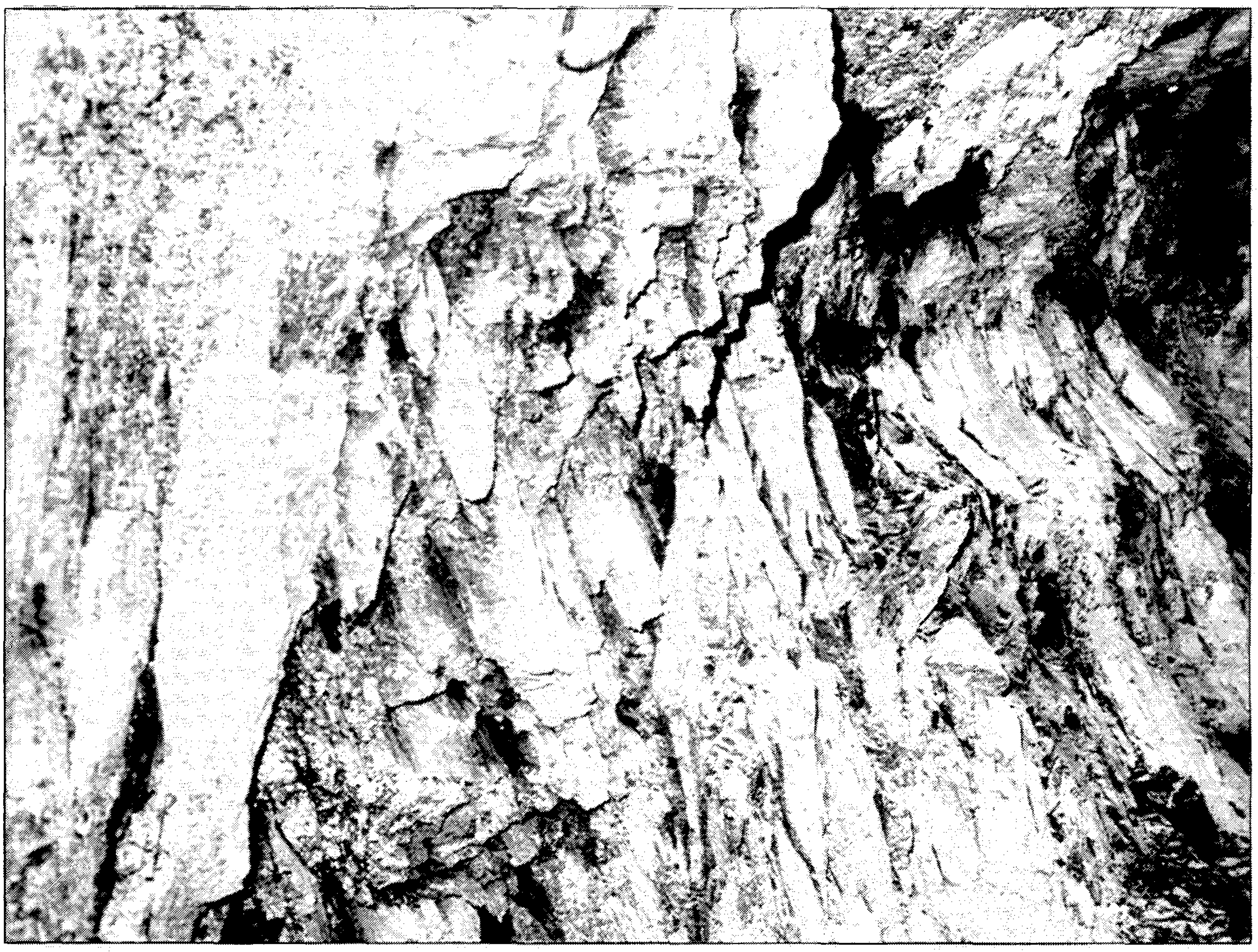
Vertical pressure profiles at site 1, Sunnyside No. 1 Mine.

Figure 68



Vertical pressure profiles at site 2, Sunnyside No. 1 Mine.

Figure 69



Yielded gate road panel rib in northeast part of Sunnyside No. 1 Mine.

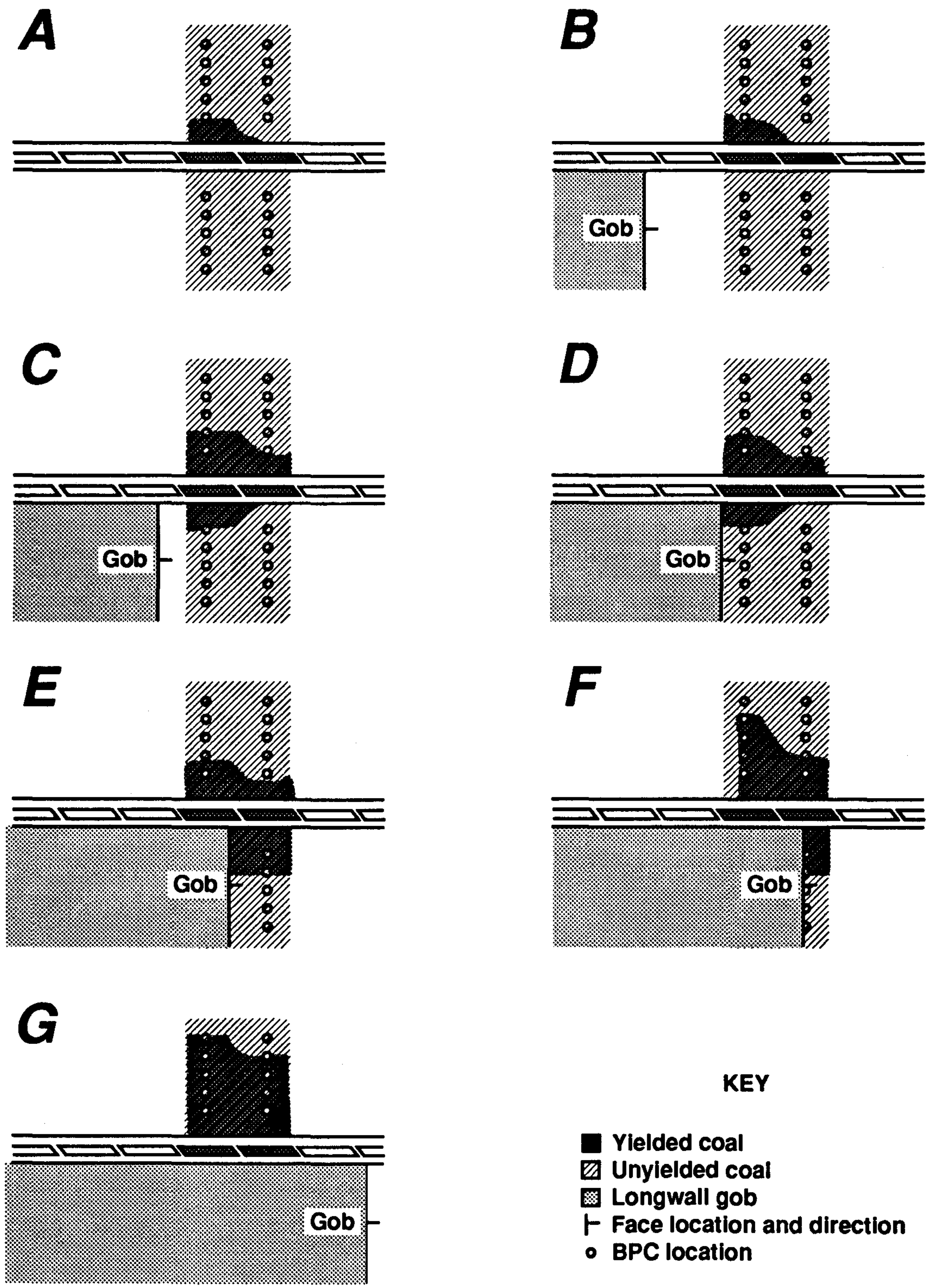
site 1 (especially considering the different fracturing pattern and rib stability that characterize the lower and upper seams of the Sunnyside Coalbed) (*50*). Moreover, the rock split, overlain by approximately 1.8 m (6 ft) of coal (the upper seam) constituted the immediate roof at site 1, whereas the immediate roof of site 2 consisted of the uppermost 0.6 to 0.9 m (2 to 3 ft) of the upper seam (top coal) and the overlying siltstone, sandstone, and mudstone.

A series of short en echelon faults, identified at the surface by Osterwald and others (*25*), overlies the area of site 2 (figure 59). At the surface, these faults show up to 1.2 m (4 ft) of vertical separation across a zone 12 m (40 ft) wide. Although the faults (or fault zone) have not been identified in the underlying No. 1 Mine workings, the faulting may have affected the structural stability of the main roof at site 2. Interestingly, these faults are located above an area of the 20th Left panel and gate road in which liquid hydrocarbons resembling crude oil oozed from the roof at several locations (*63*). According to mine personnel, this phenomenon was also encountered in nearby areas of the 21st and 22nd Left gate roads and panels, which are also overlain by this group of faults.

Figure 70 illustrates the progressive yielding sequence suggested by the site 2 data (*52*). With passage of the face, increased yielding of the adjacent panel rib is clearly shown. Yielding of the headgate was confined to the panel corner; in-mine observation indicated that the face remained relatively intact and that coal was dynamically thrown from the face (*52*). Use of yielding chain pillars apparently succeeded in transferring high-stress concentrations away from the chain pillars and edges of the entry systems, thus reducing the potential for bumps.

In-mine pillar behavior was compared with yield-pillar widths calculated using Wilson's (*71*) and Chen's (*72*) methods. Calculations indicated that the 9.7-m (32-ft) wide pillars should yield at the 608-m (2,000-ft) depth, but

Figure 70



Progressive yielding sequence at site 2, Sunnyside No. 1 Mine. (From Haramy and Kneisley (52).)

that a slightly smaller yield pillar, 9.1 m (30 ft) wide, should be used at the shallower depth of 532 m (1,750 ft). These results were generally consistent with the in-mine observations indicating earlier pillar yielding and more extensive yielding of the panel rib. These methods also show the extreme sensitivity of yield-pillar widths to the material property inputs and assumed end conditions.

Data collection was terminated and pressure recorders were removed from the sites when the mine was temporarily closed for economic reasons, precluding instrument monitoring during second-panel mining. Following mine reopening by a new owner-operator, extraction of the second panel, 21st Left, proceeded smoothly, and the infrequent bumps were concentrated along the face within about 30 m (100 ft) of the tailgate corner. Mine personnel observed that the tailgate roof was exceptionally stable, and that noticeable crib loading generally occurred approximately 23 m (75 ft) in advance of the face. From a ground control viewpoint, 21st Left retreat was relatively uneventful (*63*).

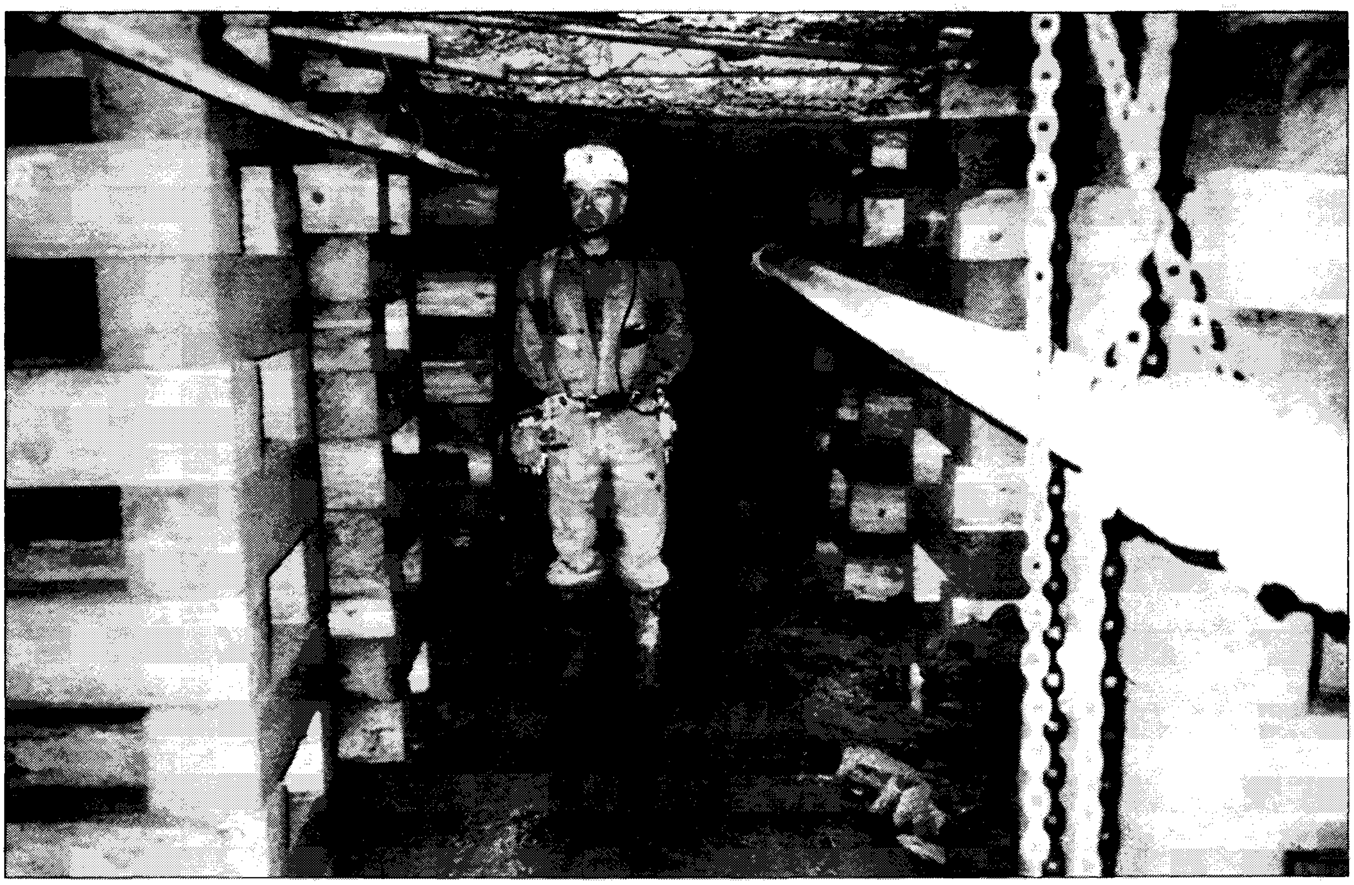
SUNNYSIDE MINES SUMMARY

The Sunnyside two-entry configuration, while requiring significant supplemental support in the tailgate, two rows of cribs on 2.4- to 3.0-m (8- to 10-ft) centers (figure 71), to maintain the return airflow and to provide a secondary escapeway, has proved successful in reducing the frequency and severity of bumps in the vicinity of the face (*10, 53, 62*).[5] From 1945 to 1965, there were 19 fatalities due to falls and bumps; from 1965 to 1984, two fatalities occurred, one because the victim proceeded inby the permanent roof supports.[6] From a ground control viewpoint, this configuration apparently succeeds because of lower pillar and

[5]Heers, R. G. (Independent consultant, formerly of Kaiser Steel Corp.). Two Entry System for Development of Longwall Panels. Prepared for MSHA, U.S. Department of Labor, Public Meeting, March 12, 1985, Denver, CO; available from L. R. Barron, U.S. Bureau of Mines, Denver, CO.

[6]Work cited in footnote 5.

Figure 71



Typical secondary-support cribbing of two-entry Sunnyside tailgate.

abutment stresses and a 36% reduction in exposed roof area. This system also reduces the number of intersections by 33% (*11*).

Regarding 9-m-wide (30-ft) versus 12-m-wide (40-ft) yield pillars, Sunnyside experience is that the narrow pillars seldom bump, whereas the wider pillars bump more frequently and with more force (figure 72). Additionally, improved tailgate roof conditions are reported when the narrower chain pillar is used (*73*).

The Sunnyside two-entry system has greatly reduced the coal bump problem and has improved ground-related mining conditions; however, in order to meet ventilation and escapeway requirements, use of the two-entry system necessitates certain precautions during both development and retreat. At Sunnyside, maximum ventilation efficiency is achieved during development by providing a separate intake entry and using the belt entry for return air. During longwall retreat, two distinct intake air courses are provided by the isolated belt entry and the intake escapeway. Two return air escapeways are provided via the tailgate entry and bleeder entries (*11*).

Figure 72

Aftermath of gate road chain pillar bump in northeast part of Sunnyside No. 1 Mine. Crosscut has been mostly filled with loose coal thrown from pillars during bump. Pillars were oversized at 16.8 m (55 ft) wide owing to headgate entry misalignment.

SUMMARY

Attempts to design longwall gate road systems to cope with adverse conditions in the four mines clearly met with widely varying degrees of success. In light of the operators' experience with the performance of the systems, corroborated by the results of the USBM gate road stability investigations, the two-entry gate road systems utilized in the Sunnyside No. 1 and Star Point No. 2 Mines succeeded in alleviating their particular ground control difficulties, facilitating safe, efficient recovery of adjacent panels. In contrast, the systems used in the 6th Right gate road of the Wilberg Mine and throughout the Castle Gate No. 3 Mine either failed to mitigate their problems or led to additional complications, resulting in (1) hazardous conditions at the face and/or the tailgate escapeway, (2) the need for massive secondary support to maintain tailgate entries, with the possibility of entry impassability should the resupport prove inadequate, (3) minimal longwall productivity, and (4) premature panel termination with consequent major reserve losses.

The Sunnyside two-entry yield-pillar gate road system resulted from experience gained through many years of mining in a bump-prone environment and evolved in response to a particular set of conditions: deep cover, 450 to 900 m (1,500 to 3,000 ft); precipitous topography; massive sandstone in the upper roof; highly elastic coal capable of storing large amounts of strain energy; and variable immediate roof lithology and competency.[7] During 33 years of longwall mining, numerous experiments with different gate road and panel configurations have confirmed that pillar width and overall gate road span (distance between panel ribs) are the primary controllable influences on gate road system performance at Sunnyside (*73*). Utilizing narrow yielding pillars eliminated tailgate pillar bumps, decreased the frequency and severity of face bumps, and alleviated occasional floor heave (*73*). The system has generally performed as designed for the range of roof and floor lithologies and qualities encountered in the Sunnyside mines to date. Variations in entry support requirements due to localized roof quality changes are dealt with by matching secondary support stiffness to site-specific roof and floor conditions (i.e., using single- or double-row cribbing with four- or nine-point cribs) in the tailgate entries.

In the case of the Star Point No. 2 Mine, a successful change to the two-entry yield-pillar gate system (*29*) resulted from an engineering analysis of gate road design based upon the documented poor performance of the three-entry stiff-pillar gate roads together with measured gate road component parameters, such as physical properties of the roof and floor, roof sag and separation, primary- and secondary-support loading, entry closure, pillar and panel stresses, and lateral dilation of pillars and panel ribs. Employing this system dramatically improved gate road stability, reducing floor heave, rib sloughage, roof falls (during both gate development and panel retreat), and tailgate support requirements, and alleviated concentrated load transfer onto underlying lower seam workings (*41*).

Two-entry yield-pillar gate systems were utilized at the Wilberg Mine in an effort to mitigate longwall-abutment pillar loading observed in three-entry abutment-pillar gate roads overlain by an extensive channel sandstone. Although generally successful throughout the rest of the mine, when the two-entry system was used in the uniquely adverse 6th Right gate road geologic conditions—weak channel-margin sandstone, siltstone, and mudstone roof, which had a decided propensity to fail and cave, together with a weak interbedded immediate floor—the chain pillars primarily yielded following development, with total pillar yield occurring during first-panel mining. The artificial secondary support thus became the primary load-bearing structures remaining in the tailgate entry of the second panel. The weak roof was unable to withstand deflection caused by pillar compression; loads imposed by the failed roof exceeded the load-bearing capacity of the secondary support and crushed chain pillars; and the weak floor heaved severely, resulting in areas of the entry becoming impassable during first-panel mining (*38*). The two-entry yield-pillar design has since been successfully employed in several gate roads of the Wilberg and Cottonwood Mines.

Although the bump-prone environment at the Castle Gate No. 3 Mine was generally similar to that at Sunnyside, the Castle Gate immediate roof and floor were much stiffer than any ever encountered in Sunnyside mining. When the two-entry yield-pillar gate road design, successful at Sunnyside, was adopted at Castle Gate, roof-floor confinement prevented yielding of gate pillars, panel ribs, and panel faces, resulting in severe tailgate pillar and face bumps. Consecutive gate road designs using incrementally narrower pillars and gate road spans never achieved an actual yield-pillar system. Consultants utilized a number of design parameters, such as overburden depth and composition, panel and gate road widths, and pillar width-to-height ratios, in conjunction with state-of-the-art pillar design methods to arrive at their gate road system recommendations, but they failed to take into account the effects of confinement on in-seam mine structures. Effective yield zones could not develop at gate pillar ribs, along panel ribs, or at the face; therefore, the wide-abutment-pillar two-entry gate road design advocated by the consultants, when used under deep cover on both sides of the final

[7]Work cited in footnote 5.

panel, could not prevent dangerous tailgate pillar and face bumps, intense loading of headgate pillars, or load override onto the scheduled next panel (*43*).

Although the four mines share a common geologic setting and general set of mining conditions, immediate roof and floor lithologies and the structural characteristics of the roof, floor, and coal seams are different in each mine, and often vary widely in a single mine. Gate road system designs that successfully alleviate adverse ground conditions and enable safe, productive longwall operations in one mine may prove inappropriate, or even disastrous, in another, and may not be the optimal design for a different area of the same mine. Site-specific quantitative data for evaluating the intrinsic parameters and mining performance of gate road structural components should be collected and analyzed during gate road development in order to foresee potential problems when adjacent panels are extracted, and throughout panel retreat to assess the performance of the current gate road system and the need for design alterations in future systems.

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