A study of the ground control effects of mining longwall faces into open or backfilled entries

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Abstract. Unusual circumstances may require that a longwall retreat into or through a previously driven room. The operation can be completed successfully, but there have been a number of spectacular failures. To help determine what factors contribute to such failures, a comprehensive international database of 131 case histories has been compiled. The cases include six failures where major rock falls occurred in front of the shields, and seven even more serious failures involving major overburden weighting. The case studies suggest two types of room failure mechanism. The first is a roof fall type failure caused by loading of the immediate roof at the face as the fender or remnant longwall panel narrows. The second is an overburden weighting type failure caused by the inability of the roof to bridge the recovery room and face area, and affecting rock well above the immediate roof. The data indicate that the roof fall type of failure is less likely when intensive roof reinforcement (bolts, cables and trusses) is employed together with higher-capacity shields. The overburden weighting failures, in contrast, occurred when the roof was weak and little standing support was used. Weighting failures were not greatly affected by the density of roof reinforcement. In one of the overburden weighting cases, in a Pittsburgh coalbed mine, stress cell, convergence, bolt load and extensometer data have been used to analyze the failure in detail.

Key words: coal mining, ground control, longwall, recovery room.

1. Introduction

Although not standard industry practice, pre-driven longwall recovery rooms and cross panel entries have been used in a number of mines for various reasons, including:

- To speed up recovery of the longwall upon completion of a panel.
- To mine through entries that were driven mid-panel to facilitate ventilation and escape, ("super" longwall panels).
- To extract old barrier pillars that may include crosscuts or crossing entries.
- To mine through areas where geologic features, such as a dike or a fault have been removed prior to longwall mining.

Experience has shown, however, that there are serious ground control risks with this procedure. Several spectacular failures have occurred, where rock falls or severe weighting pressures on the shields required weeks or even months to work through. During these incidents, miners were exposed to extremely hazardous conditions while working in very confined spaces.

In order to help prevent such failures in the future, a comprehensive database of all known examples of longwalls mined into or through pre-driven rooms has been compiled. A total of 131 case histories have been collected from 18 mines in the U.S.A., Australia, and South Africa, covering the period from the mid-1980s through to 1997. The data were obtained primarily from the literature, supplemented by personal communications and experience where applicable. For each case history, every effort was made to obtain information on the geology, the dimensions of the pre-driven room, the support installed, and the results. Table 1 shows the complete database. Some further details on the mines and case histories are provided in the paragraphs that follow.

One particular recovery room weighting failure (Pennsylvania Mine "B"), has been described and analyzed in detail. In that case, the room was instrumented with extensometers, instrumented roof bolts, convergence sensors and vibrating wire stress cells installed in both the longwall panel and in one of the front abutment pillars. The instrument data obtained from Pennsylvania Mine "B" are more detailed than are available from any of the other cases. The instrument data appear to confirm the weighting failure model derived both from individual anecdotal reports and from statistical analysis of the more limited data (both numerical and qualitative) available from the majority of the case histories.

2. U.S.A. Case Histories

Alabama Mine "A": This mine has gained considerable experience with mining through pre-driven rooms in recent years (Hendon, 1998). Successful mine-throughs include:

- Eighteen crosscuts extracted with a 76 m (250 ft) wide longwall face.
- Two "probe entries" driven across a 122 m (400 ft) wide face.
- A number of 42 m (140 ft) crosscuts inside the same 122 m (400 ft) face.

In each of these cases, the face entered the pre-driven room at an angle, generally about seven degrees. Relatively little additional bolting was used to reinforce the roof, which was usually competent siltstone. Standing support consisted of, at most, a single row of fiber cribs on 6 m (20 ft) centers.

There was also one notable failure. At the "pull-out crosscut" of the same 122 m (400 ft) face, a "massive roof fall" occurred at mid-face which required two weeks to clean up. This was the most heavily supported of any of the mined through entries, with a double row of propsetters installed on 1.5 m (5 ft) centers. However, the other difference was that the wall approached the pull-out crosscut much more slowly to facilitate meshing. It was concluded that "substantial standing support was needed at the pull-out point where the face retreat rate was reduced significantly" (Hendon, 1998).

Alabama Mine "B": A longwall was used to extract a barrier pillar which was crossed at right angles by a set of four main entries (Hendon, 1998). The first entry was supported by double rows of propsetters, but a "massive squeeze" developed as the last coal was removed from the fender (the remnant longwall panel). The shield canopies were forced onto the face conveyor and one month was required to get the longwall moving again. The remaining three entries were reinforced with double rows of fiber cribs and propsetters, and were extracted without incident. The roof consisted of 1.5 m (5 ft) of mudstone and coal, overlain by competent siltstone.

Alabama Mine "C": Partial recovery rooms have been used for many years at this mine (Stansbury, 1998). These have been located either in the middle or near the gate ends of the panels. Relatively light roof reinforcement, and no standing support, has been sufficient in the partial recovery rooms.

When a recent panel was extended through three pre-existing entries, the decision was made to fill them with a 0.7 MPa (100 psi) cellular concrete. There were no strata control incidents, but cost considerations made the experience unsatisfactory.

Most recently, cable trusses and concrete pilasters were employed in a full-face recovery room. The coal fender punched into the floor, there was significant shield convergence, and numerous pilasters crushed out, but the face was recovered on schedule. The pilasters were built of solid concrete blocks with approximately 15% wood.

Colorado: On one of the early longwall panels at this mine, a decision was made to recover the longwall from a sub-main entry. The room was supported by a single row of square fibercrete cribs topped with 0.15 m (6 in) wood blocks (Ropchan, 1990). Roof reinforcement included 2.4 m (8 ft) fully grouted rebar on 0.65 m (2 ft) centers with chain link fence. The immediate roof consisted of a weak, highly slickensided shale about 3–4 m (10–12 ft) thick, overlain by a series of weak siltstones, sandstones and shales. The shale-siltstone floor was also weak.

When the fender was between 1–2 m (3–6 ft) wide, the face advance stopped for 6 hr because the pan line was stuck. The roof began to converge rapidly as the fender crushed, and many shields yielded with several becoming iron bound. The pillars in the recovery room punched into the roof with heavy rib spalling and cutter roof failure. All the fiber cribs failed either by splitting or crushing with many showing an hour glass failure configuration. The recovery room was then heavily reinforced with wood cribs, though convergence continued.

The subsequent investigation concluded that the roof had broken at the pillar line with the rock mass moving toward the face as shown in Figure 1 (Pulse, 1990).

Table 1. Database of parameters used in analyzing the performance of longwall mine throughs of pre-driven rooms

Country	State	Mine	No. of Rooms	Soft Floor[1]	Depth, m	CMR, R^2	Seam Height, m	Panel Width, m	Room Length, m	Room Width, m	Shield Capacity, tonnes	RDI[3], MPa·m	Standing Support, MPa	Slow Mining[4]	Out-come[5]
USA	PA	A	1	N	150	40	2.4	244	61	6.1	454	0.37	5.60	N	1
USA	PA	A	1	N	150	40	2.4	183	183	5.2	454	0.43	5.60	N	1
USA	PA	A	1	N	150	40	2.4	183	183	5.2	454	0.37	4.10	N	1
USA	PA	A	1	N	150	40	2.4	183	183	5.2	454	0.37	2.80	N	1
USA	PA	C	3	N	168	40	2.9	270	270	4.9	794	0.22	1.50	N	1
USA	PA	B	3	N	210	40	2.2	250	250	4.9	635	0.53	1.80	N	1
USA	PA	B	3	N	210	40	2.2	305	305	4.9	635	0.29	4.80	N	1
USA	PA	B	1	N	220	40	2.2	305	305	6.7	635	0.88	0.00	N	3
USA	PA	B	1	N	220	40	2.2	305	305	5.2	635	0.72	1.80	Y	1
USA	MD		16	N	190	40	2.6	229	229	4.9	599	0.33	1.20	Y	1
USA	MD		1	N	190	40	2.6	229	229	4.9	599	0.33	1.20	Y	2
USA	MD		6	N	190	40	2.6	229	229	11	599	0.66	4.60	N	1
USA	CO		1	Y	140	35	2.1	168	168	5.2	590	0.62	0.30	Y	3
Australia	NSW	A	1	Y	90	60	3.1	200	200	4.2	590	0.64	0.10	N	1
Australia	NSW	A	6	Y	90	60	3.1	200	200	4.8	590	0.75	0.00	N	1
Australia	NSW	A	1	Y	50	82	3.4	200	200	4.8	590	0.75	0.00	N	1
Australia	NSW	A	4	N	290	50	3	200	200	4.8	590	1.83	0.37		1
Australia	NSW	A	1	N				225	225						2
Australia	NSW	B	1	N	275	45	3	150	150	6.5	617	0.00		Y	3
Australia	NSW	B	3	N	275	45	3	150	150	3.5	617	0.93	0.14	N	1
Australia	NSW	D	2		400	70		200			907				1
Australia	NSW	E	several					50			363				1
Australia	QLD		1	N	190	50	2.4	200	200	5.2	726	0.76	0.14	N	3
South Africa		B	2		200	70	1.9	200	120	5		0.50	0.00	Y	1
South Africa		A	1	N	125	50		200	200	2.6	327	0.42	0.22	Y	2
South Africa		A	4	N	125	50		200	200	2.6	327		3.45	N	1
South Africa		A	1	Y	70	35	3	200	100	5	327	0.55	0.00	Y	3

[*continued*]

Table 1. Continued

Country	State	Mine	No. of Rooms	Soft Floor[1]	Depth, m	CMR, R^2	Seam Height, m	Panel Width, m	Room Length, m	Room Width, m	Shield Capacity, tonnes	RDI[3], MPa·m	Standing Support, MPa	Slow Mining[4]	Out-come[5]
USA	WV		1	N	305	50	1.5	244	244		590	0.55	0.05	N	1
USA	WV		1	N	305	50	1.5	244	244		590	0.52	0.13	N	1
USA	WV		6	N	305	50	1.5	244	244		590	0.52	0.00	N	1
USA	WV		6	N	305	50	1.5	244	244		590	0.42	0.00	N	1
USA	AL	C	3	N	366	67	2.1	265	265		726	0.13	0.69	N	1
USA	AL	C	3	N	366	67	2.1	265	265		726	0.41	0.33	N	1
USA	AL	C	12	N	366	67	2.1	265	53		726	0.54	0.22	N	1
USA	AL	B	1	Y	610	47	2.3	107	107	6.1	590	0.15	0.24	N	3
USA	AL	B	3	Y	610	47	2.3	107	107	6.1	590	0.15	0.72	N	1
USA	AL	A	1	N	610	57.5	2.5	76	76	6.1	590	0.15	0.00	N	2
USA	AL	A	1	N	610	57.5	2.5	76	76	6.1	590	0.15	0.06	N	2
USA	AL	A	16	N	610	57.5	3.0	76	76	6.1	590	0.29	0.06	N	1
USA	AL	A	2	N	610	68	3.0	122	67	6.1	590	0.09	0.00	N	1
USA	AL	A	6	N	610	68	3.0	122	67	6.1	590	0.32	0.00	N	1
USA	AL	A	1	N	610	68	3.0	122	67	6.1	590	0.32	0.24	Y	2
USA	OH		1	Y	150	45	2.1	152	152	4.9–5.5	340	0.19	0.18	Y	3

[1]Soft Floor. Y=Soft. N=Normal or not noted as soft by the original source.
[2]CMRR=Coal Mine Roof Rating.
[3]RDI=Reinforcement Density Index. The product of the support capacity and the support length, divided by the tributary area affected by the support and summed for all support types. In the case of trusses the length of one anchor is used. The index does not apply to standing supports.
[4]Slow Mining. Y=Slow mining. N=Normal or rapid mining or rate unknown.
[5]Outcome. 1=Successful outcome. 2=Failure due to face break or face fall. 3=Failure due to major overburden weighting.

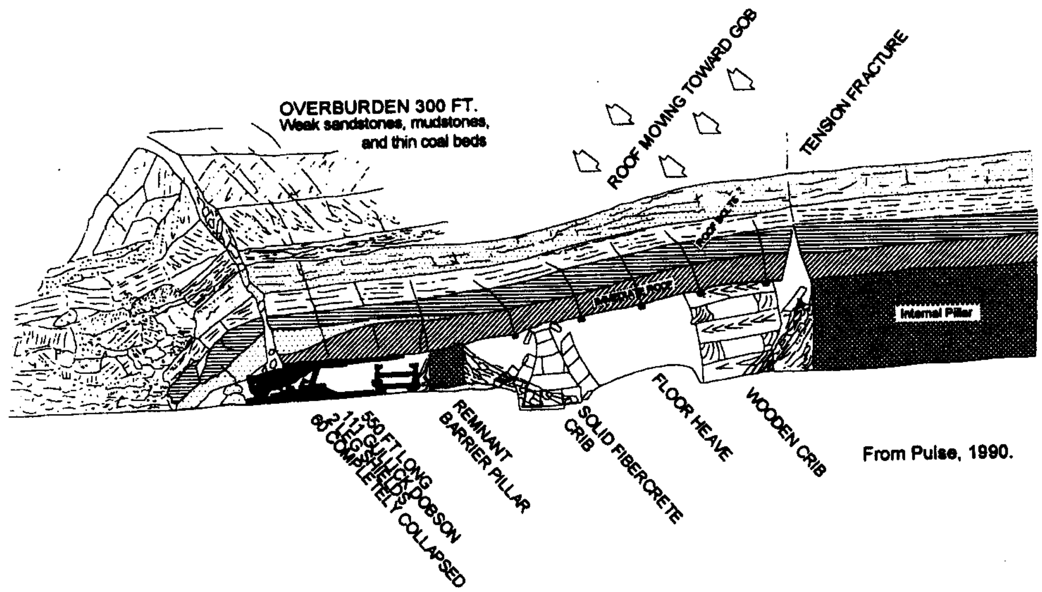


Figure 1. Typical physical behavior of a pre-driven room under weighting type roof failure. From underground observation of an actual recovery room failure. Note the fracture observed to develop at the front edge of the abutment pillar.

Investigators noted a "tensile fracture" which resulted in the "roof moving toward the gob." With the roof beam apparently pivoting around the pillar rib, the largest roof movements were experienced at the face, and a significant portion of the abutment load appeared to have shifted onto the shields. Under such conditions the face could not move the last few feet into the room.

The longwall was eventually recovered from the location where the face was halted, just short of the recovery room. It was necessary to grout and heavily support the room to allow the fender to be mined out with a continuous miner and provide room for the recovery. The recovery operation took about two months.

Maryland: This mine has used 23 recovery rooms, typically 4.9 m (16 ft) wide; but six of the rooms were 11 m (36 ft) wide. The only failure was one of the standard rooms where roof falls necessitated two weeks of remedial action. The extra support in this case was a row of concrete cribs on the longwall panel side of the recovery room and a row of wooden cribs on the abutment pillar side. The damage occurred when the face was 11 m (35 ft) from the room. Mining rates were slow, because of wire meshing activity.

The wide recovery rooms were designed so that the face would not have to slow down for meshing (Wynne *et al.*, 1993a, Wynne *et al.*, 1993b). The room was developed and supported in two passes. Supplemental support included eight rows of concrete donut cribs, three rows of truss bolts, and two rows of 5 m (16 ft), 25 mm (1 in) diameter roof bolts. All the wide rooms were reportedly mined without serious incident (Wynne, 1998).

Ohio: A single recovery room was attempted at this mine in the Pittsburgh coalbed sometime in the early 1980s. The recovery attempt resulted in a weighting type failure. Two 4.9–5.5 m (16–18 ft) entries were driven across the longwall face. The standing support consisted of large wooden posts and cribs in the recovery room and a large number of steel posts in the second room. The standing support density was estimated from examination of photographs of the room taken shortly after the failure. From a telephone conversation one of the authors (Oyler) had with an engineer who worked on the recovery project, the primary intrinsic support is believed to have consisted of 2.4 m and 3 m (8 and 10 ft), 16 mm (5/8 in) diameter mechanical bolts on 1.2 m (4 ft) centers. No secondary intrinsic support was used. Longwall census data give the mining height as 2–2.2 m (78–87 in) and the shield capacity as either 310 or 340 tonnes (two different shield types were used during the period in which the recovery attempt was believed to take place). In the telephone conversation the floor was noted as being soft with the shields digging into the bottom, possibly due to floor heave. National Institute for Occupational Safety and Health data on nearby Pittsburgh coalbed mines where thin competent sandstones are often present in the bolting horizon give estimates of the Coal Mine roof Rating (CMRR) (Molinda and Mark, 1994) between 35 and 55. The immediate roof at this mine was described as 1.8–2.4 m (6–8 ft) of drawrock and slickensided shale with a poor quality sandstone at 2.4–3 m (8–10 ft), suggesting a much lower roof strength. An estimate of 45 has been used for the CMRR in this report. The face was angled with the headgate about 14 m (46 ft) ahead of the tailgate. The angle is clearly apparent in photographs. Finally, mining was delayed at some point by as much as 1.5 shifts. At the time the weighting failure took place the shields had already advanced into the recovery room at the headgate. Because it is not possible to determine which portion of the face is shown in the photographs, it is not clear how far the face had advanced when the failure took place. It took several months to recover the face.

Pennsylvania Mine "A": A total of four recovery rooms were successful at this Pittsburgh seam mine (Bauer *et al.*, 1988; Bauer *et al.*, 1989; Bauer and Listak, 1989; Listak and Bauer, 1989). Three different types of concrete supports were used. The first room employed 1.2 m×1.8 m (4 ft×6 ft) flyash concrete piers. The piers were designed to "replace the load-bearing capacity of the coal" by providing a support resistance of 5.5 MPa (800 psi). To reduce costs, fibercrete cribs with 0.3 m (1 ft) wood cap blocks were used on the next room. Again, the cribs were placed to give a support density of 5.5 MPa (800 psi). The concrete cribs were hard on the shearer and stageloader, however, and so the last two panels employed poured concrete cylinders. The concrete was pumped from the surface to fill 1 m (3 ft) diameter cardboard tubes. The top 0.2–0.3 m (8–12 in) above the concrete cylinders was wedged with wood. The support density was reduced on the fourth panel following good results from the third. No ground control problems were encountered in any of the recovery rooms.

Pennsylvania Mine "B": Three different techniques have been used at this Pittsburgh seam operation. Little information is available on the results from the first recovery rooms. The mining company considered the rooms successful, but did not have full confidence in the technique because of floor heave and the failure of some of the donut cribs and several later panels were recovered short of prepared rooms, using conventional techniques.

More recently, a set of three entries were driven across a longwall panel so that the panel could be lengthened (Chen *et al.*, 1997). The entries were filled with a low strength cement-flyash mixture. There were few ground control problems in mining through the entries, although the wood and steel left in the rooms caused equipment problems.

A full-face pre-driven recovery room was less fortunate. No standing support was used, though the roof was heavily reinforced with 2.4 m×22 mm (8 ft×7/8 in) mechanically-anchored resin-assisted bolts, 3.7 m (12 ft) cable bolts, mesh over the entire roof, two rows of T-5 channels running parallel to the entry and 15 mm (0.6 in) cable trusses on 1.2 m (4 ft) centers. The room was driven 5 m (16 ft) wide and later widened to 6.5 m (22 ft). Supplemental bolting was performed to provide support of the additional mined width. During longwall operations the mining rate averaged more than 15 m/d (50 ft/d) as the panel approached the room. When the fender was 3 m (10 ft) wide, the room began to deteriorate and most shields went on yield immediately after being set. The face entered the recovery room at the headgate and tailgate areas, but the roof converged to the floor over much of the entire mid-panel section. It took several weeks to advance the shields through the collapsed roof, with remedial efforts including the use of polyurethane grout and the installation of cribs.

This recovery room was extensively instrumented with roof extensometers, load cells on roof bolts, strain gauged roof bolts and cable trusses, and vibrating wire stress cells installed in an abutment pillar and in the panel. The data obtained from these instruments will be discussed in detail in a later section of this paper.

The face was then advanced to a second entry that had been mined at the same time as the original recovery room. This room was supported by two rows of donut cribs, a row of 0.76 m (30 in) wooden cribs, and cable bolts. The longwall recovery was successful, although the donut cribs were heavily damaged and a large amount of convergence took place.

Pennsylvania Mine "C": A three-entry system was driven across one panel to allow it to be extended beyond an adjacent, shorter panel. The entries were driven at a 30 degree angle to the panel, each supported by a single row of 1.2 m (4 ft) diameter poured cement cribs on approximately 2.4 m (8 ft) centers (Bookshar *et al.*, 1998). The body of the cribs was made up of a stiff high strength concrete, and the cribs were then topped with a plastic bag approximately 0.3 m (1 ft) thick and filled with a yielding proprietary cement. The mine throughs were successful, with the largest measured deformations just over 127 mm (5 in) in the tailgate, and much smaller over most of the face.

West Virginia: Eleven faces have been recovered using full-face recovery rooms at this mine (Smyth, 1998). The first two used some standing support, but most have used just roof bolts and cable bolts for reinforcement. The coal is quite thin, leaving little room for convergence. While some shields have been stuck, most face recoveries have been conducted without incident.

3. Australian Case Histories

New South Wales (NSW) Mine "A": Recovery rooms were used on 13 panels in four different coalbeds at this mine (Simpson *et al.*, 1991). All cases but one were successful. The unsuccessful case was the only mine through attempt in the West Borehole coalbed. The available information for the West Borehole case is limited to the panel width and the type of failure. On the earlier panels in the Fassifern coalbed 1.8 or 2.1 m (6 or 7 ft) resin bolts, "w" straps and mesh were used for support. Standing support, consisting of two rows of timber props was used only on the first panel. The rooms were 4.8 m (16 ft) wide, except the first, which was only 4.2 m (14 ft) wide. A panel was also recovered in the shallower Great Northern coalbed using the same configuration. When the first recovery room was planned for the deeper Young Wallsend coalbed, three rows of 10 m (33 ft) fully grouted cable bolts were added, (at least near the headgate; the published report is unclear on whether cable bolts were used across the entire room) and a row of 1.8 m (6 ft) diameter standing supports known as Big Bags were installed on 5 m (16 ft) centers near the panel side rib.

The mine management attributed the success of the recovery rooms to a large extent to the presence of a soft claystone floor which allowed the fenders to be slowly punched into the floor, with manageable floor heave, and avoiding fender yield. They reported that the fenders typically did not yield until the last few meters and in some cases did not yield at all. In the deeper Young Wallsend coalbed this mechanism was not relied upon, the room was heavily supported by both secondary and standing support, even though the shale floor there was also soft. The sandstone or conglomerate immediate roof above the Fassifern, Great Northern and Young Wallsend probably also contributed to the success of the recovery rooms at this mine. The mine preferred to angle the face by keeping the tailgate back, as much as 8 m (26 ft) in the Young Wallsend coalbed, so that if ground control problems occurred, they would be less likely to affect the entire panel.

NSW Mine "B": Four rooms were mined into, the first being a full face recovery room 6.5 m (21 ft) wide and the next three being narrow (3.5 m or 12 ft) full face entries driven for ventilation. The first recovery room failed, causing the shields to go solid and several months were required to recover them. The room was supported by a standard primary bolting pattern, with spot cable bolting only in areas considered critical, such as gate road and chute intersections. No standing support was used. The motor on the shearer ranging arm broke down when the face was just a few meters from the room and the face remained idle for an extended period. The fender failed when it was 2 m (6 ft) wide. High water flows from the gob and gas flows into the tailgate entry were noted in the course of the mine through.These were interpreted as the effect of the failure and subsequent weighting of an overlying sandstone aquifer and the opening of fractures in failed rock to allow gas flow.

The remaining mine throughs were supported by cable bolts, props and glue injection into the fender. Primary support consisted of 2.4 m (8 ft) bolts and "w" straps on a 0.8 m (2.5 ft) spacing, 10 m (33 ft) cable bolts at a density of 3 cables/2 m of entry and 3 rows of 150 mm (6 in) props on 0.8 m (2.5 ft) centers, over the entire length of the entry. These mine throughs were successful, although there was no necessity to remove the shields since mining continued after the longwall passed through the room.

NSW Mine "C": A recovery room was used to successfully recover a single longwall at this mine. No specific information is available on the supports or on the panel geometry, but it is known that significant secondary support in the form of cable bolts and timber props was used.

NSW Mine "D": Recovery rooms were used on two panels at this mine. The immediate roof in both cases is the massive, competent Coalcliff Sandstone. Both mine throughs were considered completely successful. No information is available on the type of support used in the rooms in either case. Recovery rooms were not used on subsequent panels because the roof lithology changed from competent sandstone to shale and because later panels were also significantly longer, reducing the importance of rapid face moves.

NSW Mine "E": No information is available except that several panels were recovered, the panels were narrow and they were successful.

Queensland: A single recovery room was attempted at this mine (Klenowski *et al.*, 1990). Primary roof support consisted of five rows of 2.1 m (7 ft) resin bolts on 1.5 m (5 ft) spacings with "w" straps. The bolts closest to the ribs were angled over the panel and barrier pillar. Two rows of 8 m (26 ft) cable bolts were also used on 4 m (13 ft) centers. Grouted 1.8 m (6 ft) fiberglass rib bolts were installed in the panel and in the barrier pillar. Standing support was only planned for use on an as-required basis. The design was based upon the results of an instrumented 15 m (50 ft) stub entry.

Convergence was noted to begin to accelerate when the fender was 6 m (20 ft) wide. When the fender was 5 m (16 ft) wide it failed and the shields began to continuously yield. Just prior to entering the room a maximum of 0.42 m (17 in) of convergence took place at one place on the face, where the shearer was hung up under the canopy of a shield and typical convergence may have been 0.26 m (10 in). Less convergence took place in the recovery room. When the shields entered the room the mining height was greater and the available hydraulic fluid was insufficient to set them (due to fluid losses from continuous yielding of the shields). A delay took place until the hydraulic reservoir was refilled. The convergence continued after the room was entered and eventually it was necessary to set timber props, with a total of 392 props finally being used. Of the 440 shield legs on the face, 104 were found to have failed after the mine through due to malfunction of the gas yield valves. One conclusion arrived at by the mining company was that had the shields not failed, the roof convergence would not have been so large and fewer timber props would have been required. However, it is possible that the high rates of convergence caused the damage to the shields.

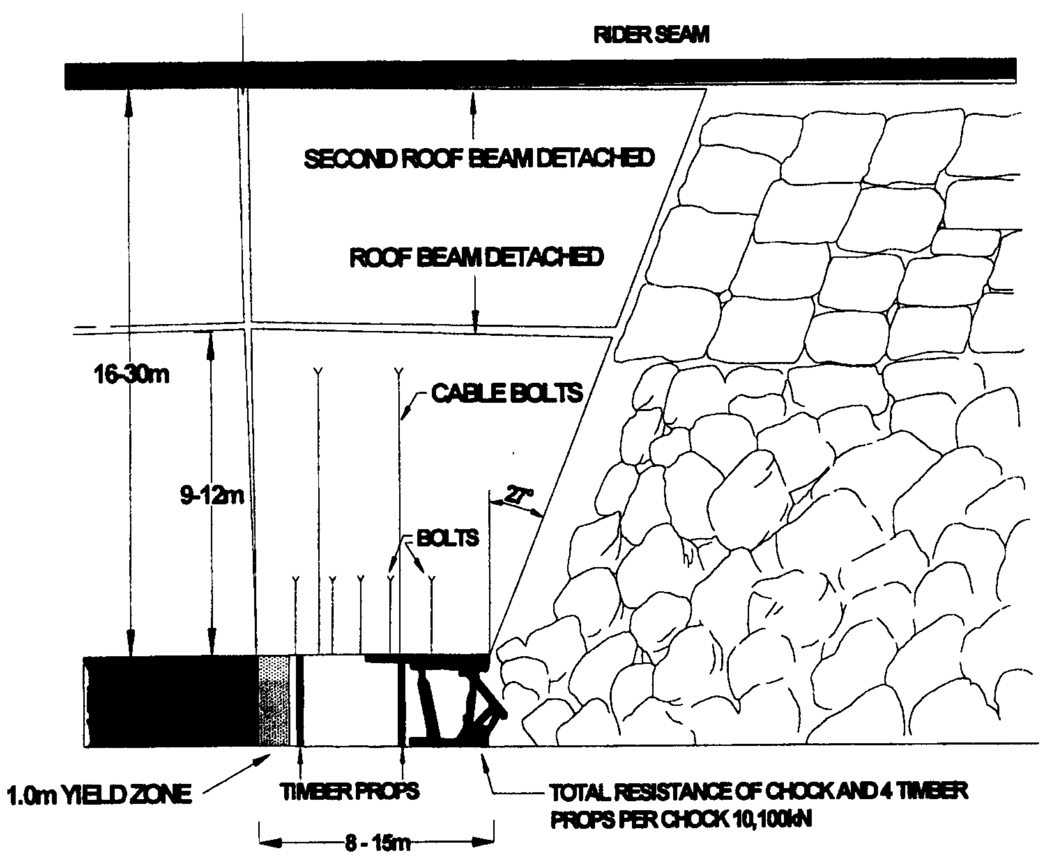


Figure 2. Conceptual model of a weighting type room failure. Based upon observations of an Australian weighting failure. From Klenowski, 1990.

An analysis made after the failure suggested that a "tensile failure" had extended upward to a rider seam located 16 m (50 ft) above the German Creek Seam (Figure 2). Convergence occurred much more rapidly at the face than in the recovery room, indicating that "the pivot point of the failing roof beam was over the barrier pillar" (Klenowski *et al.*, 1990).

4. South African Case Histories

South Africa Mine "A": Six mine throughs were attempted, five to mine through dikes and an experimental recovery of part of a face (van der Merwe, 1988, 1989, 1998). The dike mine throughs used either mat packs (rafts of timbers wired together and laid one on top of the other) or filled entries, except for the third mine through where only timber props were used. For the first and second mine throughs mat packs were used, for the third timber props, for the fourth a back fill material consisting of cement, plaster and sand, and sifted ash as well as timber props, and for the fifth a mix of coal fines and other unspecified materials used to improve the flow characteristics and strength of the mix.

All of the mine throughs except the third were successful. In that case the room was entered 0.3 m (1 ft) too low and the recovery room eventually collapsed before the dike could be blasted away to allow advance into the room.

The sixth mine through was a recovery room test and only covered a portion of the panel. During the mine through the conveyor belt broke and the face sat idle for 8 hrs when the fender was 3 m (10 ft) wide. Apparent water entry from a joint in the panel softened the floor rock and caused the fender to punch into the floor. The floor under the face conveyor heaved and left the face conveyor and shearer too high to allow the shields to be advanced. The recovery room roof also converged until it was impossible to enter, forcing shield recovery in place 3 m (10 ft) from the recovery room, a "lengthy process".

South Africa Mine "B": Two rooms were used successfully to remove a dike crossing both of them at roughly a 10° angle to the face orientation (Minney, 1999). The immediate roof rock was approximately 8 m (26 ft) of competent sandstone directly above the #4 coalbed. The main roof included nearly 50 m (160 ft) of competent siltstone/sandstone and sandstone. The rooms were driven by first mining an entry in the coal alongside the dike and then removing the dike, which was about 0.6 m (2 ft) wide. Primary support in the rooms consisted of three rows of 1.5 m (5 ft) fully grouted resin bolts on 2 m (6.6 ft) centers. Secondary support consisted of 1.8 m (6 ft) tensioned fully grouted resin bolts on 1.5×2.0 m (5×6.6 ft) centers and two rows of 6 m (20 ft) cable bolts. No significant standing support was used in either mine through. The second panel mine through experienced 5 shifts of down time, without significant detrimental effect on either the longwall face or the dike room.

In order to reduce the risks associated with the mine throughs, the panel widths were reduced to some 120 m (394 ft) on the first panel, and to an estimated 140 m (460 ft) on the second. On both panels a 30 m (100 ft) long entry was mined from the room, parallel to the gateroads, toward the advancing longwall face to serve as a tailgate entry and facilitate the removal of the shields from the portion of the panel that would not be mined until the dike was reached and the face could be widened back to 200 m (650 ft).

5. Failure Mechanisms

In the vast majority of cases, longwalls have been successfully mined into pre-driven rooms. Of the 131 cases, only 13 were apparently complete failures. However, the costs associated with these failures, and the hazards they created, were very substantial.

The failures can be divided into two categories. The first includes six cases where the problems were due to roof falls occurring in front of the shields. The second group consists of seven cases involving severe shield weightings accompanied by major convergence.

Roof falls occurred in two situations as the longwalls approached the pre-driven rooms. Some failures developed in the unsupported roof span between the shield tips and the fender (the portion of the longwall panel between the shields and the room), because of the increased span that resulted from extensive coal yielding in the fender. Other roof failures developed because of the large span between the shield tips and the abutment pillar once the fender crushed or was mined out. In both cases a substantial portion of the room could be involved. Additional secondary support was typically required to prevent or control roof fall failures.

The weighting type failures resulted in the most severe ground conditions. In the weighting failures, the shields were loaded to the yield point, allowing large roof-to-floor convergences that could be severe enough to cause the shields to become iron-bound. The heavy loading apparently resulted when a new caving break line developed at the abutment pillar rib of the pre-mined room. In the seven cases where the weighting type failures occurred, accelerated rates of convergence were initiated when the fenders were 3 m (10 ft) or less in width. Up until then, the fenders apparently provided enough support to the main roof to prevent the formation of a new caving break line.

Where the standing support density in a room is insufficient, when the new cave line develops, the shields may be called upon to control most of the weight of a cantilever of roof that extends across the pre-mined room. The additional span can double the load that the shields previously supported. As shown in Figure 2, the height of the broken rock requiring support is also greater than normal because of the wider span between the shield tips and the abutment pillar. Further, because the shields are not directly under the main rock mass, they are not in a good position to handle this new load distribution. As the detached block moves down, the main roof is affected. If the main roof is not strong enough to bridge over the detached block, it will subside and add additional load to the face area. Once a weighting failure begins, intrinsic supports (roof bolts) are of little use, because the new caving zone goes above them. On the other hand, where the main roof is strong, it can help the overburden to bridge across the room and face, resulting in manageable face and room conditions, even with minimal standing support.

6. Qualitative Analysis of Factors Affecting Pre-driven Rooms

In this study, every effort was made to obtain information on a variety of descriptive characteristics of all the case histories. The results are summarized in Table 1. The goal was to identify those characteristics which correlated with the failed case histories. Statistical techniques, including Pearson correlation and logistic regression, aided the analysis.

Immediate Roof: Descriptions of the roof geology were usually contained in the literature, and were quantified using the Coal Mine Roof Rating (CMRR). In almost every instance, the published description was supplemented by CMRR data collected by the authors in past visits to the mines.

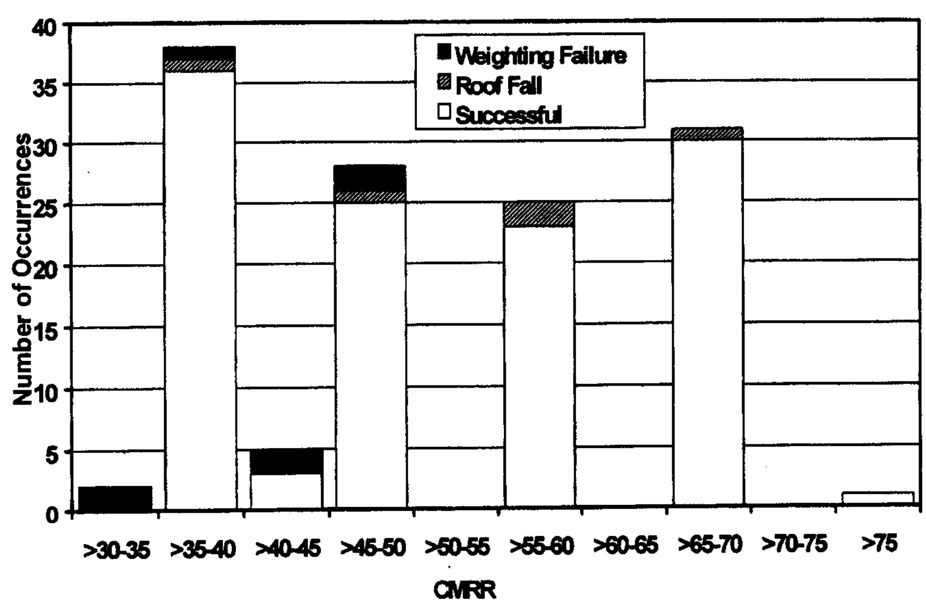


Figure 3. Histogram of CMRR versus mine through occurrences, and indicating results of the mine through.

A very strong correlation between the CMRR and weighting failure was found. All of the seven weighting failures occurred where the roof was relatively weak (CMRR<50). The correlation is less evident for roof fall type failures (Figure 3).

Main Roof: Insufficient data were available on main roof geology for analysis. The authors believe, however, that the main roof geology should be very important to weighting failures. It seems likely that the CMRR is substituting for a characterization of the main roof in many instances.

Floor: Soft floor was reported in 14 of the mine throughs, but these included three of the weighting failures. Under some conditions where the thin, heavily loaded fender is likely to punch into the floor, the potential for failure could be increased, although there were also successful cases where soft floor was credited with delaying fender yield and contributing to the success of the recovery rooms.

Overburden Depth: Case histories were available for a wide range of depths. Roof fall type failures were more likely to occur at greater depth, but not weighting failures (Figure 4).

Seam Thickness: No correlation was found between seam thickness and either type of failure. It is worth noting that while there is less potential for roof weighting when the seam is thin, the tolerance for convergence is also usually much less.

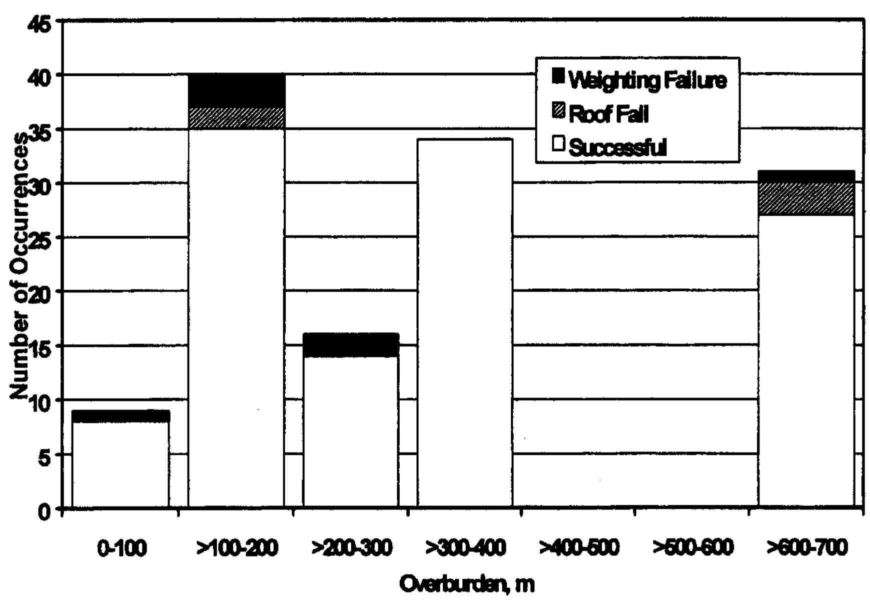


Figure 4. Histogram of Overburden versus mine through occurrence.

Mining Rate: A slow mining rate seems to be a strong predictor of both types of failures. Twenty mine throughs were associated with reports of slower than normal mining, either because they were meshing, or because a breakdown occurred near the recovery room. Four of the weighting incidents, and two of the roof fall failures, were in this group. It is possible that details such as slow mining rates are more likely to be seen as important and reported when a failure occurs. The mining rates may actually have been slow in some of the reported successes, but not considered worthy of comment.

Room Width: There seems to be little correlation between room width and difficulties. The very widest recovery rooms, the 11 m (36 ft) rooms at the Maryland mine, were trouble free. However, it seems reasonable that overall stability will improve as the room width decreases, all other factors being equal.

Room Length: It might be expected that overburden weighting failures would occur more often in longer rooms. No such correlation is apparent in these data, however (Figure 5). Two of the severe weighting failures (one in South Africa and the other in Alabama) occurred on faces that were less than 110 m (350 ft) wide.

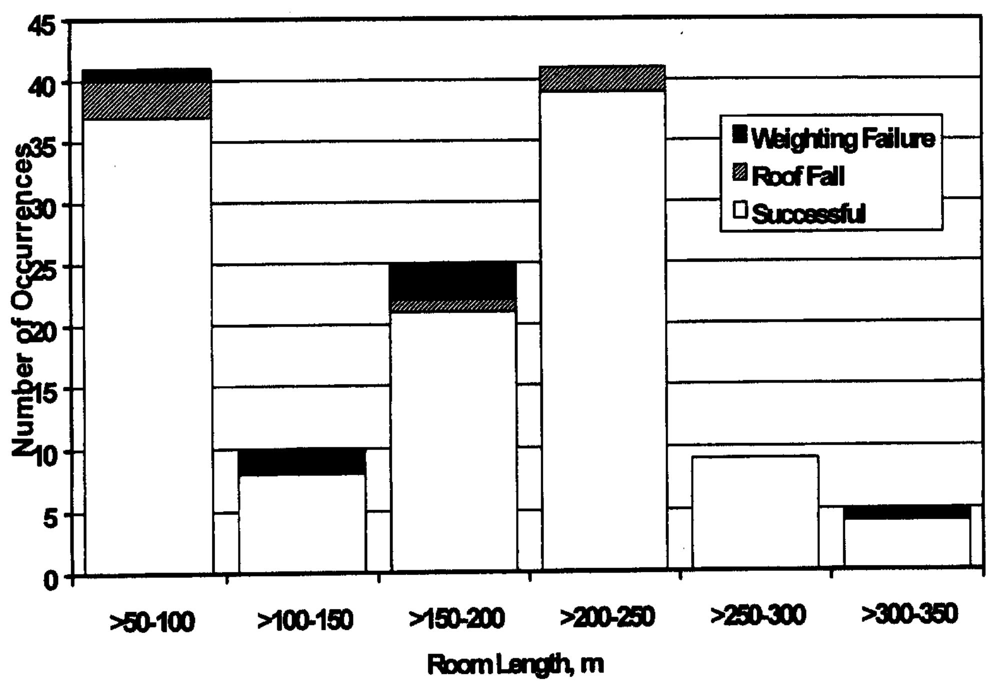


Figure 5. Histogram of Room Length versus mine through occurrence.

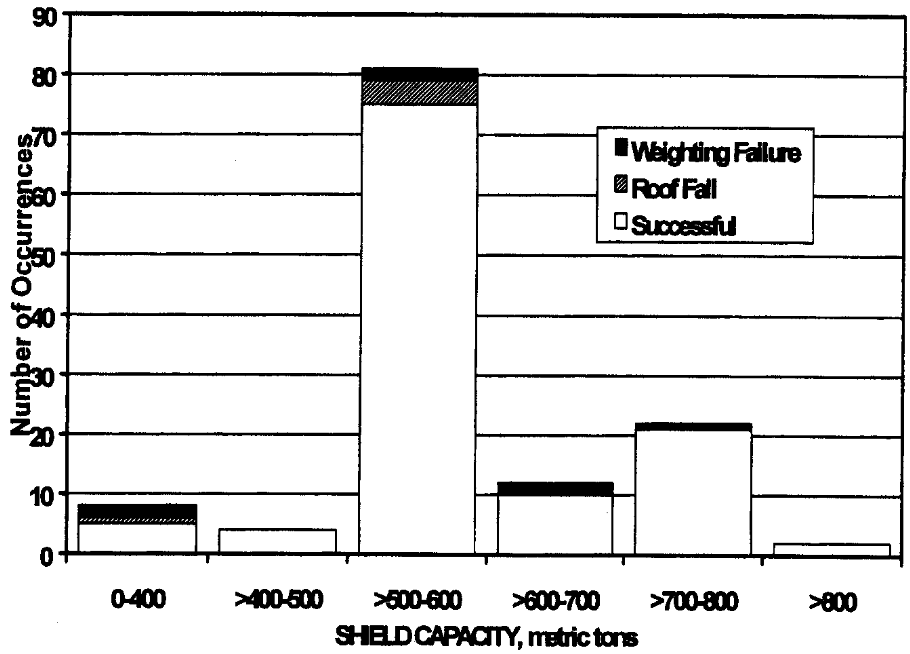


Figure 6. Histogram of Shield Capacity versus mine through occurrence.

Shield Capacity: Some correlation was observed between shield capacity and roof fall failures, but higher capacity shields apparently have not helped to prevent weighting failures (Figure 6).

Roof Reinforcement: Roof reinforcement includes all intrinsic support elements, such as roof bolts, cable bolts, and trusses. The Reinforcement Density Index (RDI) is obtained by determining the load capacity of each element per unit area of roof supported by the element and multiplied by the length of the element. The RDI has the units of MPa·m. Where several types of support were used, the ratings for individual supports are summed.

Heavy roof reinforcement was apparently successful in reducing the incidence of roof fall type failures. Roof reinforcement did not appear to be effective in preventing weighting failures (Figure 7).

Standing Support: A characteristic of every one of the weighting failures is a low density of standing support (Figure 8). Standing supports are any supports which are constructed in the room between the roof and floor, such as posts and cribs. The standing support density, with units of pressure (MPa), is determined by dividing the maximum capacity of each support by the tributary area it supports. Where several types of standing supports were used, the standing support densities for individual support types have been summed to determine the standing support

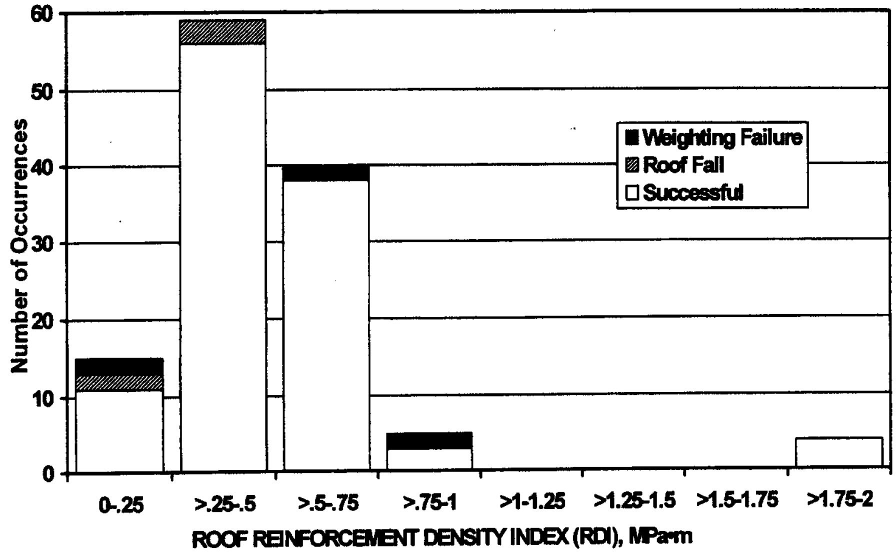


Figure 7. Histogram of Roof Reinforcement Density Index (RDI) versus mine through occurrence.

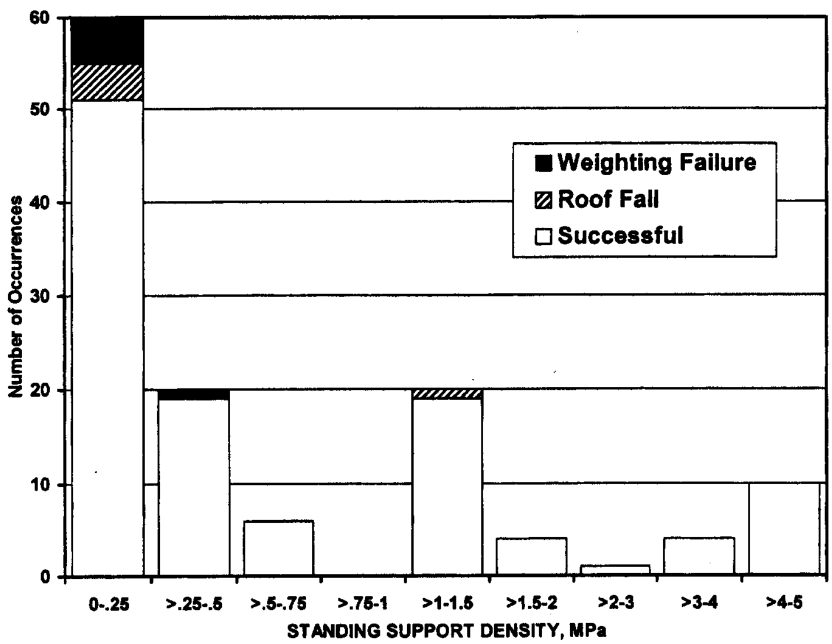
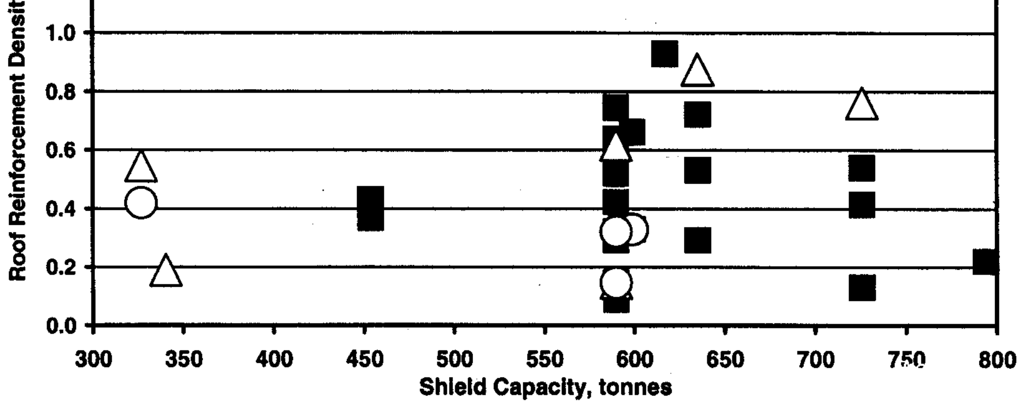


Figure 8. Histogram of Standing Support versus mine through occurrence.

density given in Table 1. In two mines, after a severe weighting failure developed in a room without standing support, adjacent rooms were mined successfully with standing support. These two cases indicate that standing support can be the difference between success and failure. The importance of standing support also lends credence to the proposed mechanism of weighting failure.

Face-Room Angle: Because of the limited data on cases where the room was at an angle to the face, no detailed analysis was conducted on this parameter. As noted earlier, some mines have entered pre-driven rooms at small angles or with the tailgate 6–10 m (20–33 ft) behind the headgate to limit the extent of ground control problems. The evidence that a narrow fender still provides significant support suggests that this approach may be valid. Entering a room at an angle may allow the shields along the leading portion of the face to reach the abutment pillar before the load carrying capacity of the fender is lost over most of the length of the longwall face. Despite the likelihood that entering a room at an angle may improve the probability of a successful outcome, it should be noted that at least one weighting failure (the Ohio case) took place on a face angled at 5° to the room.



7. Multivariate Analyses

Logistic regression was employed to obtain insight into possible design guidelines. Three analyses were performed, analyzing the roof falls, the weightings, and the combined set of failures. Mining rate and weak floor were not included in the analysis because the data were judged incomplete.

Looking at roof falls alone, the two most important variables were shield capacity and the total intrinsic support or reinforcement density index (RDI) (Figure 9). Roof fall type failures appeared to be restricted to those cases with low shield capacities (<600 tonnes) and low RDI values (<0.5 MPa*m). However, this relationship is relatively weak for several reasons. First, there were relatively few roof fall type failures. Second, there were only a few cases where shield capacities were below 600 tonnes. Finally, not all cases meeting those conditions were failures.

For weighting failures, the CMRR and standing support are the two important variables (Figure 10). A highly significant relationship indicates that when the CMRR was greater than 50, little standing support was necessary. For CMRR=40 roof, the successful cases used a standing support density of at least 1.0 MPa (145 psi). For CMRR values in the range of 45–50 standing support densities as low as 0.5 MPa (73 psi) appeared sufficient to prevent or control weighting failures. However, the cost of standing support is small compared to the cost of a weighting room failure, and the observations made from Figure 10 should not be taken as a recommendation to eliminate standing support.

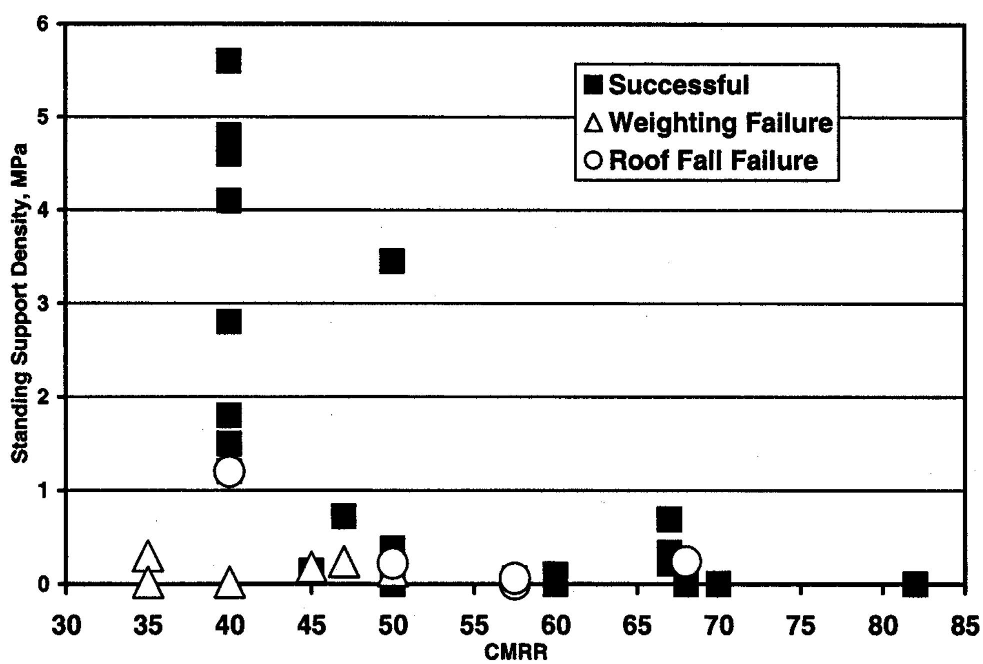


Figure 10. CMRR versus Standing Support Density, indicating results of the mine through.

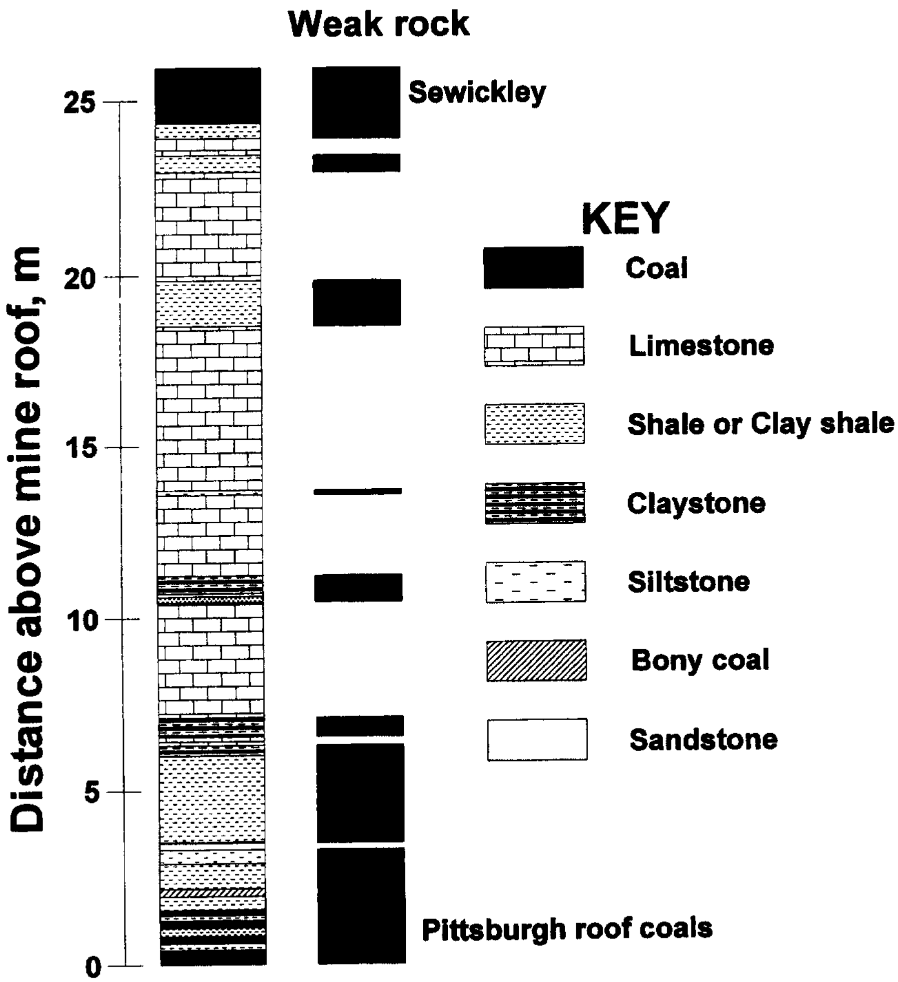


Figure 11. Log from a corehole in the longwall panel 23 m (75 ft) in front of the recovery room at Pennsylvania mine "B". The bars beside the log indicate weak rocks including coals, shales, claystones, and siltstones.

8. Field Study of Recovery Room Performance

As described earlier in this report, a full-face recovery room without standing support was attempted, unsuccessfully, at Pennsylvania Mine "B." The roof collapsed into the room in a weighting failure that extended across the entire face. A large array of instruments had been installed in the recovery room, and the data from these instruments illustrate and provide insight into the weighting failure mechanisms presented in this paper.

Figure 11 shows the roof lithology at the site, obtained from a corehole drilled in the panel 23 m (75 ft) from the room and roughly at mid face (Figure 12). The log extends to the Sewickley coalbed 26 m (85 ft) above the mine roof. The rock

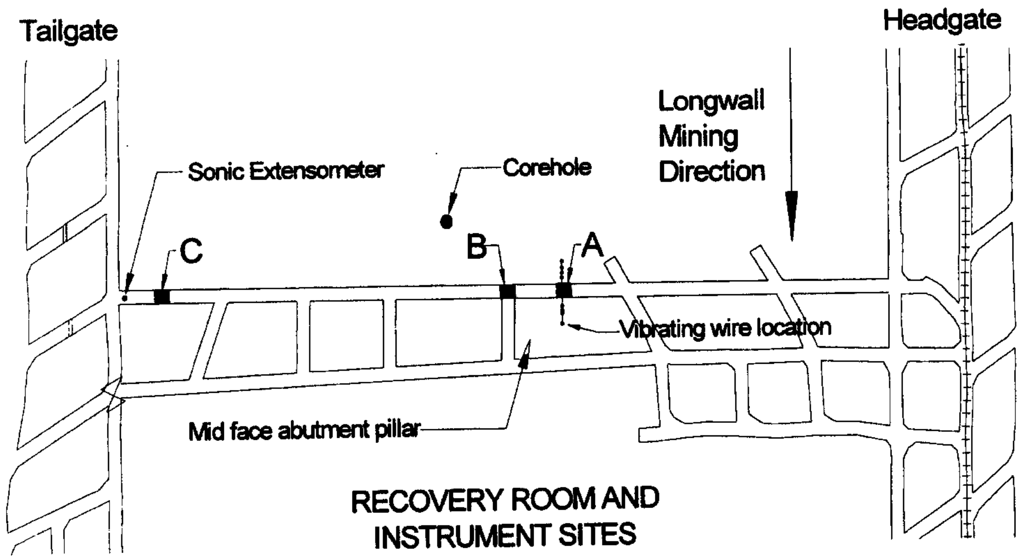


Figure 12. Full face recovery room at Pennsylvania mine "B", showing instrument locations. The face width is 305 m (1000 ft). The entry width is 6.7 m (22 ft).

types are primarily limestones, shales and clay shales. Other coreholes in the vicinity confirm the lateral persistence of the lithology, although the claystones are thicker in some areas. The rocks in Figure 11 have been divided into weak and strong rock types, assuming that the limestones and sandstones are strong and the other rock types are weak. From this characterization it appears that the first 7 m (23 ft) of the roof consists primarily of weak rocks, while the remainder of the section is primarily strong rock, and suggests that the immediate roof may have the greater influence on the outcome of a mine through. This is consistent with the good correlation observed between CMRR and outcome. However, another possibility is that a separation plane formed at the bottom of the Sewickley coalbed leading to the detachment of a 25 m (82 ft) thick roof beam similar to the one described in the Queensland case (see Figure 2) and as suggested to one of the authors by an observer of the Ohio case.

The room was instrumented at four sites (Figure 12); at both an intersection and a mid pillar site at mid face (sites A and B), at a mid pillar site near the tailgate side of the panel (site C), and in the room just off of the tailgate entry. The instruments installed included roof extensometers at all four sites (of three different types installed to depths between 5.8 and 6.7 m [19 and 22 ft]), load cells (hydraulic and strain gauge types) on roof bolts, strain gauges on roof bolts, strain gauges on roof trusses, roof to floor convergence sensors and vibrating wire stress cells installed in the panel and in the abutment pillar. Instrument installation began three months before the face reached the recovery room, and all instruments were installed and connected to data recorders at least two weeks before the room was reached, when the fender was still over 244 m (800 ft) wide. The only exceptions were that

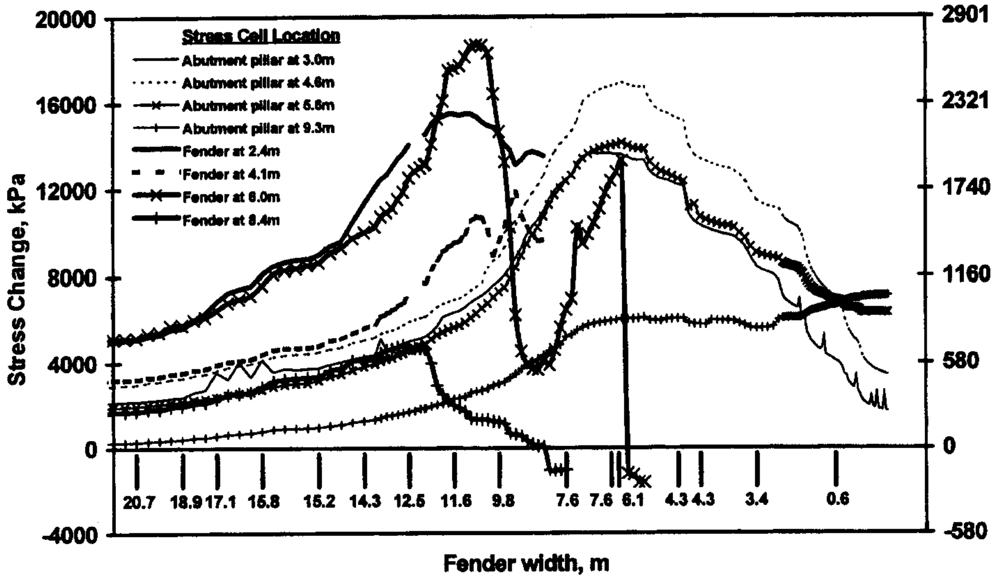


Figure 13. Pennsylvania mine "B". Stress changes measured by vibrating wire stress cells installed in the longwall panel and mid face abutment pillar.

the sonic extensometer, and four strain gauged bolts at site C were read manually throughout the entire recovery attempt.

8.1. PENNSYLVANIA MINE "B" DATA ANALYSIS

Mid Face Stress Changes: A total of eight vibrating wire stress cells were installed to monitor the changes in vertical stress in the fender and in the mid face abutment pillar. A plot of the changes in stress with time, beginning when the fender width was 22.5 m (74 ft), is shown in Figure 13. A coal modulus of 3450 MPa (500000 psi) has been assumed in the computation of the stress changes. Measurable stress changes were first noted in the deepest panel cell (at 8.4 m or 27.5 ft) when the fender width was about 91 m (300 ft). The cells loaded at a consistent, but very low loading rate until the fender width was less than 58 m (191 ft). At that time the 6 m (20 ft) panel cell read a total increase in stress of 1.1 MPa (160 psi). The other seven cells (including the deepest panel cell) indicated smaller stress increases at that time. A significant increase in the rate of loading of all cells took place after the fender width was reduced to 58 m (191 ft) and the rates of loading remained roughly constant until the fender was 22.5 m (74 ft) wide. Up to that time, although the loading rates were different from cell to cell, each change in loading rate could be observed to take place in all eight cells, indicating that the panel and the abutment pillars were behaving as a unit. At that time the loading rates began to increase, and continued to increase until the panel cells failed, were cut out by the shearer or their signals were lost when the cables between the cells and the datalogger were cut. Loading rates in all 8 cells continued to change in unison until the fender width was 14.6 m (48 ft). At that time the shallower three panel cells began to load more rapidly than the abutment pillar cells, and individual panel cells began to fail. The two deeper cells appear to have been destroyed as the shearer reached them, while the shallower panel cells were both lost when the fender width was reduced to 8 m (26 ft), probably when the signal cables were cut by a rib roll.

The panel cells reached their maximum loads when the fender was between 12.5 and 10.7 m (41 and 35 ft) wide. As the load on the panel cells began to drop, the abutment pillar cell loading rates began to accelerate, indicating that load was being transferred from the fender to the abutment pillar. The abutment pillar cells reached their maximum loads when the fender was 7.6 m (25 ft) wide. The shallower three cells began to rapidly unload at that time, while the deepest cell, at 9.3 m (30.66 ft) into the abutment pillar only slightly unloaded and then began to slowly load up again when the fender width was reduced to 3.3 m (11 ft). By the time the collapse of the recovery room had made further shield advance impossible, the panel side 5 m (16 ft) of the abutment pillar had shed over 80% of the front abutment load developed during the advance of the longwall face.

Mid Face Convergence: The roof to floor convergence sensors installed in the room at sites A and B measured almost no convergence until the fender width was 58 m (191 ft). From that point on the convergence rates gradually increased, with

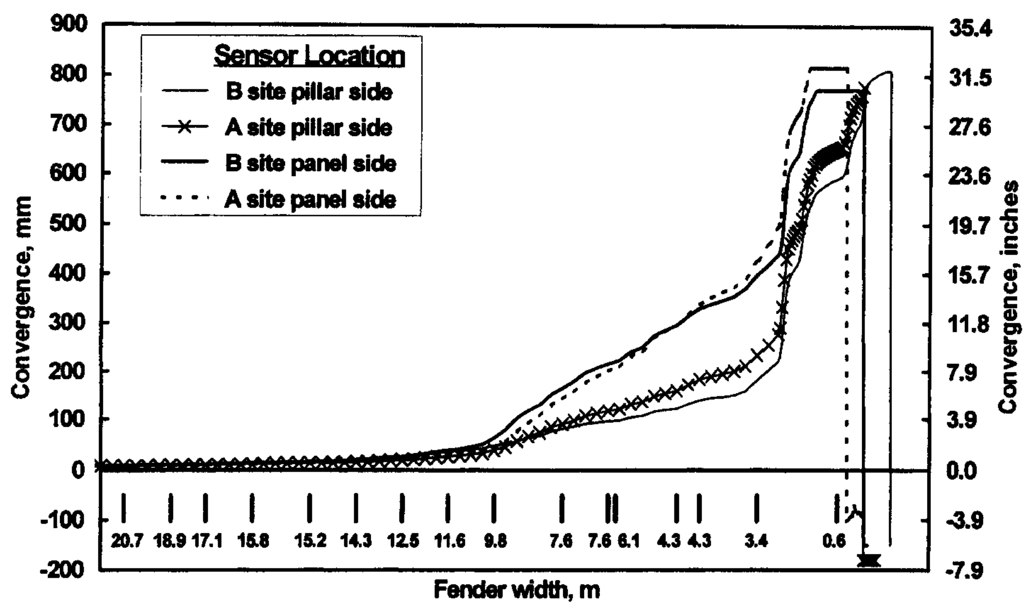


Figure 14. Pennsylvania mine "B". Convergence measured on the panel and abutment sides of the recovery room at sites A and B.

the convergence reaching 2.5 mm (0.1 in) when the fender was 37.4 m (123 ft) wide, 12.5 mm (0.5 in) when the fender was 16.8 m (55 ft) wide (Figure 14), and 30–43 mm (1.2–1.7 in), depending upon sensor location, when the fender was 10.7 m (35 ft) wide. The latter readings were made about an hour after the panel vibrating wire cells reached their maximum readings. Once the panel vibrating wire cell readings began to drop, the rates of room convergence jumped up to 25 mm/hr (1 in/hr) on the panel side, and 12 mm/hr (0.5 in/hr) on the abutment pillar side. The rate of convergence was steady until the fender was 3.3 m (11 ft) wide. Then the convergence rates began to drastically accelerate, and by the time the fender was about 2.4 m (8 ft) wide the convergence rate was as much as 600 mm/hr (24 in/hr), for both the panel and pillar side sensors. The panel side instruments reached their maximum travel within two hours (the shearer had cut into the room at the headgate by that time). The pillar side instruments were destroyed by the failure of the roof in the room after about five hours, before they could reach their maximum travel.

Mid Face Bolt Loads: The responses of the load cells on the bolts at the A and B sites are shown in Figure 15. Bolts AP-1 and BP-1 were 2.4 m (8 ft) mechanically anchored resin-assisted bolts and the remaining bolts were 3.7 m (12 ft) cable bolts. Significant loading of the bolts began when the fender width was 14.6 m (48 ft), and rapid acceleration of the loading rates began about the same time that

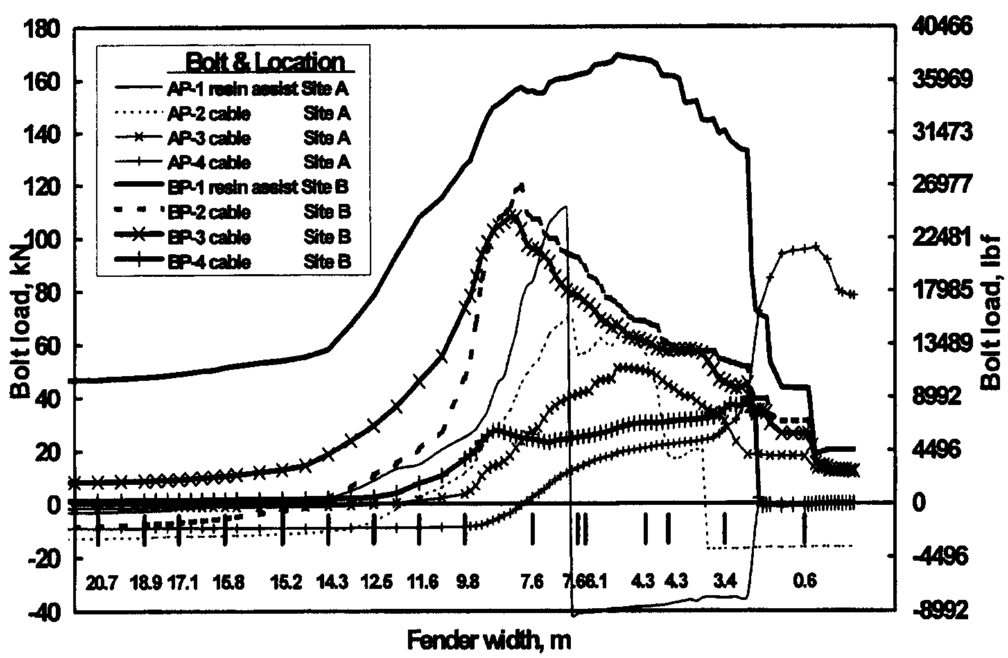


Figure 15. Pennsylvania mine “B”. Loads measured on site A and B bolts.

convergence rates in the room began to accelerate. Most of the bolts reached their peak loads when the fender width had been reduced to between 8.8 and 7.6 m (29 and 25 ft). The exceptions were cable bolts AP-3, AP-4 and BP-4 which reached their maximum loads at various times later in the course of the room collapse. Bolts AP-4 and BP-4 were cable bolts adjacent to the abutment pillar, where the rate of roof convergence was smaller until late in the process of the room failure (although BP-4 was actually installed adjacent to an intersection). Subsequently, most of the bolts began to shed load. By the time the fender width was reduced to 3 m (10 ft) all of the bolts except AP-4, the abutment pillar side cable bolt at site A, had completely unloaded.

Four cable trusses were also strain gauged at the A and B sites. The trusses did not begin to significantly load until the fender width was 2.7 m (9 ft), about the same time that the room convergence rate reached its maximum. At that time three of the four trusses loaded up to over 130–180 kN (15–20 tons), over a period of 1–3 hr. The fourth truss did not load up at all. Of the three trusses that took load, two failed in the following 2–3 hr, when the fender width was about 1 m (3 ft).

Mid Face Extensometers: Two roof extensometers were installed at both site A and site B, located approximately 1/3 and 2/3 of the way across the room. Each extensometer had six anchors, located at approximately 1.2, 2.7, 3.4, 4, 4.9 and 6 m (4, 9, 11, 13, 16, and 20 ft). Figure 16 shows the data from extensometer BE-2

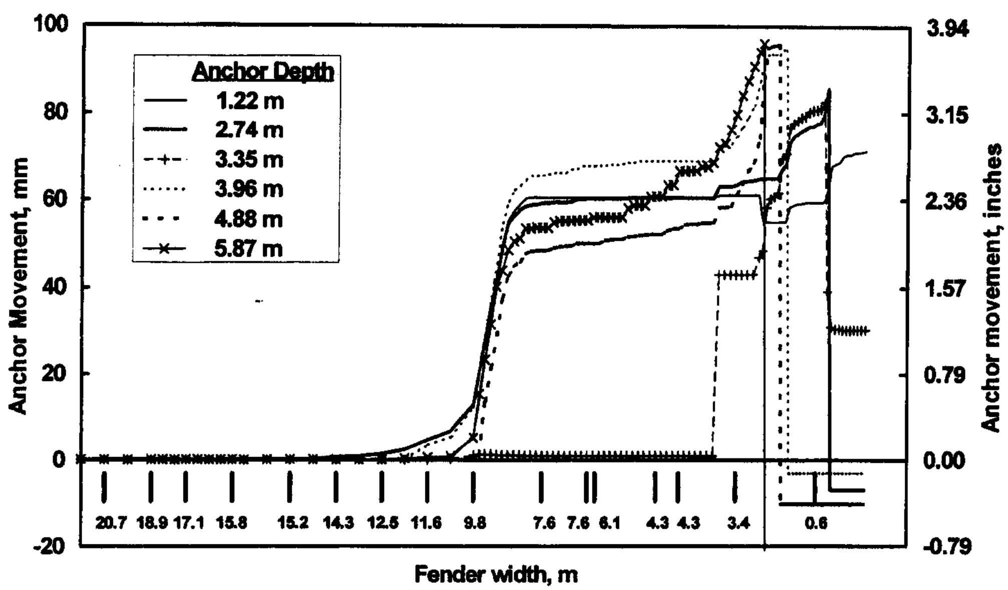


Figure 16. Pennsylvania mine "B". Roof movement (sag) measured by extensometer BE-2 at mid face intersection B (see Figure 12 for location).

located on the abutment pillar side of the room at the intersection site. The other extensometers behaved similarly, except that they generally exhibited slower, more uniform sag rates, and each had more anchors locked or hung up. The extensometers began to indicate slight roof movement when the fender width was 13.4 m (44 ft) (Figure 16), and all four instruments began to measure rapid movement when the fender was 8.8 m (29 ft) wide. This was shortly after the loads measured by the panel vibrating wire cells began to drop and at the same time as the first big increase in the room convergence rates. From the fact that anchors from all depths moved approximately the same distances, it appears that the initial roof movement generally took place below the shallowest, 1.2 m (4 ft), anchor. The panel side extensometers failed when the fender width was 5.2 m (17 ft). The abutment pillar side extensometers (including BE-2) indicated that the roof movement halted for several hours when the fender width was reduced to between 8.5 and 3.4 m (28 and 11 ft). The length of this period was different for AE-2 and BE-2, lasting longer for the latter extensometer. The period roughly corresponds to the period when the convergence rates first greatly increased, but before the final, drastic increase. Extensometers AE-2 and BE-2 again began to indicate roof movement in the bottom 1.2 m (4 ft) of the roof at about the same time as the convergence rates accelerated to their maximum rates. Extensometer BE-2 failed when the fender was about 2 m (6 ft) wide and AE-2 continued to measure skin movement until the recorder was turned off. Prior to the fender width being reduced to 3.3 m (11 ft) none of the four extensometers gave readings which would indicate that separation of the rock layers was taking place between 1.2 and 6 m (4 and 20 ft) At that time the readings from BE-2 begin to suggest that the rock layers between 1.2 and 6 m (4 and 20 ft) were beginning to separate.

Tailgate Site: The results from the instruments installed at site C, near the tailgate, were similar to those just described. Generally the changes noted at the A and B sites took place later at site C, probably due to the support provided by the presence of the tailgate pillars. Significant roof to floor convergence at the C site did not begin until the fender width was reduced to about 3 m (10 ft), and only about one-third of the convergence measured at the A and B sites took place at C before data collection was ended. Only one instrumented bolt at C developed any significant load, the cable bolt closest to the abutment pillar loaded to 186 kN (21 tons) before the datalogger was disconnected. The extensometers at the C site did not indicate roof movement until the fender was about 3 m (10 ft) wide, compared to 9 m (30 ft) at the B site. The extensometer closest to the panel read only minor movement before being lost. The extensometer closest to the abutment pillar read up to 57 mm (2.25 in) of sag, and unlike the extensometers at the A and B sites, the sag appeared to be evenly distributed throughout the roof. However, the sag at the C site began to take place at about the same time as the BE-2 extensometer began to indicate that the entire roof was separating, so the data appear to indicate that the roof in the tailgate vicinity skipped one stage of the process that took place at mid face. The cable trusses at the C site began to load at about the same time as those at sites A and B.

8.2. PENNSYLVANIA MINE "B" SUMMARY

To make the events easier to follow, Figure 17 has been included, with stress curves from the 6 m (19.66 ft) panel and 4.6 m (15 ft) pillar vibrating wire cells, sag from the 5.87 m (19.25 ft) anchor of extensometer BE-2, and convergence from the A site panel side sensor. The extensometer curve has been multiplied by a factor of ten to make it more readable.

As the width of the fender was reduced, the rates of loading of the fender, and the front abutment pillars increased until the coal of the rapidly thinning fender yielded and failed. This interpretation is based upon the stress cell, convergence and extensometer data (Figure 17). The failure took place when the fender was about 10.7 m (35 ft) wide. The yielding of the fender led to an abrupt increase in the rate of loading of the abutment pillar and acceleration of the rates of roof sag and room convergence. At this point, despite the increased room convergence, the fender was probably still providing substantial roof support. Evidence for this comes from the stress measured by the 6 m (19.66 ft) vibrating wire cell which had precipitously dropped upon fender yield but began to increase again when the fender width was about 8 m (26 ft), and continued to increase until the cell was cut out by the shearer. Total failure of the fender as a roof support element probably took

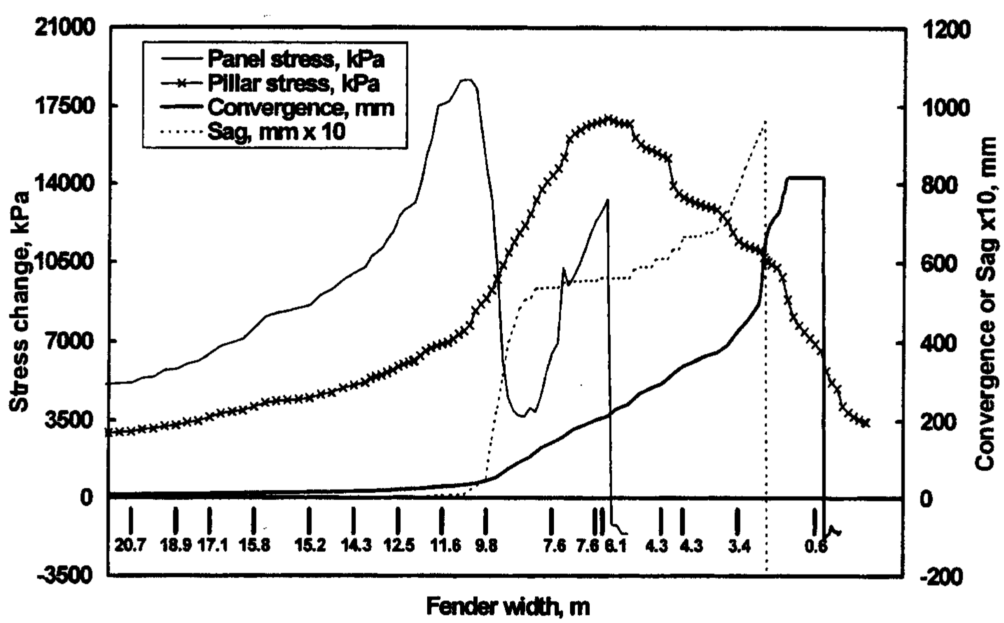


Figure 17. Pennsylvania mine "B". Composite graph of data from the 6.0 m (19.66 ft) panel stress cell, 4.6 m (15 ft) abutment pillar stress cell, the extensometer BE-2 5.87 m (19.25 ft) anchor and the A site panel side convergence sensor. The extensometer (sag) curve has been multiplied by a factor of 10.

place when the fender width was about 3 m (10 ft), about the same width noted by observers in other open entry mine throughs. At the time of total failure of the fender, convergence rates had increased to as much as 600 mm/hr (24 in/hr).

Underground observations (the sounds of thumping and ripping of rock could be heard and seemed to be located above the middle of the mid face abutment pillar) suggest that the roof rock began to break above the mid face abutment pillar when the fender was still 20 m (66 ft) wide. Including the room and abutment pillar, this break up would have been taking place as much as 27–37 m (88–121 ft) ahead of the face. The convergence and sag data suggest that after the fender failed, the roof began to move as a unit at a minimum height of 6 m (20 ft) above the room. This conclusion is based upon the large difference in convergence and sag. At the time the fender was 5.2 m (17 ft) wide, the total sag measured by the top anchor in BE-2 was 56 mm (2.2 in), while the room had converged by 243 mm (9.6 in). Since there was little or no floor heave in the room, the difference can only be explained by a detachment and subsequent 188 mm (7.4 in) of downward movement of a block of roof a minimum of 6 m (20 ft) thick, above the deepest extensometer anchor. The large difference between the panel side and abutment pillar side convergence sensor readings (Figure 14) also suggests that this block was rotating toward the fender about a hinge point somewhere between the rib and a point about 5 m (16 ft) into the abutment pillar. The deeper hinge point is based upon the behavior of the abutment pillar stress cells. This hinging and rotation of the rock above the open entry is consistent with observations made in other cases of weighting failures (Figures 1 and 2).

The behavior of the 4.6 m (15 ft) abutment pillar load cell (Figure 17) is also consistent with the weighting failure model. As the load dropped in the panel cells, the rate of loading in the pillar cell increased, suggesting that the weight of a cantilever of roof from the shields to somewhere above the abutment pillar was being shifted forward from the fender to the abutment pillar. Then, as the new break line formed somewhere between the rib and middle of the abutment pillar, the newly detached block of roof began to rotate toward the gob (Figure 2), and the stress on the front of the abutment pillar began to decrease. By the time the roof in the recovery room had completely failed, almost all of the load developed on the panel side of the abutment pillar had been removed and transferred back to the shields and gob. By this time, the fender was no longer capable of supporting a significant portion of the load.

The minimum load contributed by the detached block may be estimated by assuming a 6 m (20 ft) high, 11.3 m (37 ft) long, 1.5 m (5 ft) wide block acting upon each shield. The length comes from the assumed length of the block just as the shields entered the recovery room and assuming that the block was detached at the rib of the abutment pillar. The estimate assumes that the block extended from the back end of the shield canopies to the rib of the abutment pillar. If the weight of the 6 m (20 ft) high block of roof is computed and it is assumed to be fully detached, the total load per shield would be about 270 tonnes (297 tons). This is well below the 635 tonnes

(700 ton) capacity of the shields. However, the weighting failure model also predicts that as the length of the cantilever increases, the height of detachment can also be affected. There is also anecdotal evidence from the observations of an engineer who was present at the Ohio longwall weighting failure that the height of the break line in that case extended to the Sewickley coalbed, 24 m (80 ft) above the coalbed. The Sewickley coalbed was found at almost exactly the same height above the coalbed in the Pittsburgh mine "B" case. If the roof had detached at the Sewickley coalbed the dead load of the resulting detached block would have been over 1,080 tonnes (1,190 tons).

The apparent sequence of events at the Pennsylvania mine "B" indicates that the weighting failure took place in three phases: (1) As the fender was reduced in width the loads in the fender and in the abutment pillars initially increased. The first phase ended when the fender had yielded, which occurred when it was about 10.7 m wide. (2) The fender load then rapidly shifted to the abutment pillars, and at the same time the roof above the room began to rotate toward the longwall face, hinging about a caving line that apparently began to form between the rib and the middle of the abutment pillar prior to the failure of the fender. The block of roof above the fender and the open room also separated from the main roof forming a detached block a minimum of 6 m (20 ft) thick and possibly as much as 24 m (80 ft) thick. The detached block began to rotate and converge toward the gob, increasing the load on the fender, shields and gob and reducing the load on the panel side of the abutment pillar. During this phase, the fender retained a significant post-failure strength and continued to function as a roof support. (3) When the fender's width was reduced to about 3 m (10 ft), it completely failed and it lost its remaining load carrying capacity. The shields were then loaded to yield and the rate of convergence of the detached block increased catastrophically. Within a short time it was no longer possible to advance the shields and the roof continued to collapse until the complete closure of the room provided support for the detached block and allowed the shields to be dug free, a process which took about a month.

9. Conclusions

As the vast majority of cases attest (92%), longwall mining into pre-driven rooms can be safely accomplished with adequate support for the conditions. However, severe ground control problems from weighting can be encountered where the roof is weak and where the standing support density is insufficient. Although it appears of little value in preventing weighting failures, intrinsic roof support is of significant value in preventing roof fall failures. In the cases of weighting failures, the failure involves the main roof and the establishment of a new caving break ahead of the face along the abutment pillar. The costs associated with these failures and the hazards created are substantial. However even with a weak roof, (CMRR=40), a standing support density of 1.0 MPa (145 psi) appeared sufficient to prevent or control failure and allow the longwall to enter or pass through the rooms. With stronger roof (higher CMRR values) the standing support density required appeared to decrease. This is an observation and should not be considered a recommendation to eliminate standing support solely on the basis of measured CMRR values. Other factors that contribute to this type of failure include soft floors that can result in pillar punching, and slow mining rates.

With a stronger roof, where the weighting type failure does not occur, much lower standing support densities or intrinsic roof reinforcement alone have been successful. However even under these conditions roof falls can still occur either in the unsupported area between the shields and the fender or in the room itself. In the cases where the falls occur in the rooms, they can be controlled by increased roof reinforcement or by the addition of standing support.

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