Interaction Between Wall Rock Closure, Cemented Backfill Load, and Reinforcement Bolt Load in an Underhand Stope at the Lucky Friday Mine

T. Williams, D. Denton , J. B. Seymour, and D. Tesarik[1]

C. Peppin and D. Bayer[2]

ABSTRACT

Hecla Mining Company and NIOSH cooperated in a study to document the safety of rock support supplied by reinforced cemented paste backfill at the Lucky Friday Mine. Data are presented on backfill modulus values from laboratory tests and compared with modulus values from in-mine wall closure on the backfill. It is theorized that a compressive zone is created in the backfill between the upper and lower bearing plates on the reinforcing bolts, and that the backfill remains stable as long as the compressive zones from adjacent bolts overlap.

INTRODUCTION

Hecla Mining Company and the Spokane Research Laboratory (SRL) of the National Institute for Occupational Safety and Health (NIOSH) cooperated in a study to monitor reinforced cemented backfill at the Lucky Friday Mine, Mullan, ID. The objective was to determine loads on the reinforcing bolts induced by wall closure and loading of the backfill. The combination of these three elements—backfill response, rate of wall closure, and bolt loading—determines the stability of the reinforced backfill system used at the mine. Laboratory tests of backfill samples were conducted to compare with field results.

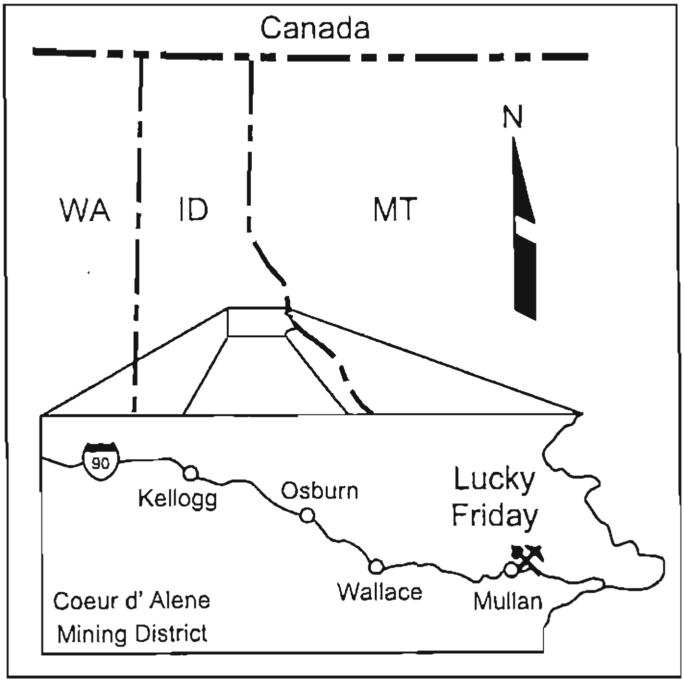

Figure 1.—Location of Lucky Friday Mine, Mullan, ID

The Lucky Friday Mine (figure 1) uses a mechanized underhand cut-and-fill method to mine lead-silver ore from a steeply dipping, 2.4-m-wide vein at a depth exceeding 1.6

---

[1] Spokane Research Laboratory, National Institute for Occupational Safety and Health, Spokane, WA.

[2] Hecla Mining Company, Lucky Friday Mine, Mullan, ID.

km (Scott 1990). Each mining cut is 3.3 m high and extends approximately 75 m along the vein to each side of an access ramp (slot). The broken ore is stored in a muck bay on the opposite side of the vein from the access slot. This creates a four-way intersection where the backfill may span a distance of up to 9 m on the diagonal. In the underhand mining method, the mined-out stope is backfilled with reinforced cemented mill tailings following each cut, which provides a safe stope back or roof for the following cut. Approximately 70% of the stope height is backfilled, leaving a 1-m gap between fills.

Approximately 10% cement by weight of solids is added to the mill tailings. Enough cement must be added to strengthen the backfill rapidly so that mining the following cut can be resumed under the backfill without a long wait. This amount of cement has been selected after years of experience in balancing the cost of the cement and the need to start mining the following cut soon after backfill placement. The backfill is delivered from the surface with a pastelike consistency, which lowers water content and increases the final strength of the backfill (Brackebusch 1994).

The high horizontal in situ stresses at the mine (Whyatt et al. 1995; Whyatt and Beus 1995) result in rapid closure of the wall rock in the mined-out portion of the vein, and this wall closure is the main factor affecting the stability of the backfill. Mining-induced stope closure induces loads in backfill beyond its ultimate strength, causing it to fracture. These fractures generally do not pose a hazard to miners working in the stopes beneath the backfill because any broken material is contained by wire mesh connected to rock bolts used to reinforce the cemented backfill placed in the previous cut.

Backfill reinforcement consists of 1.8- or 2.4-m-long, No. 7 bolts supplied by Dywidag Systems International, Inc.,[3] of Grand Junction, CO. These bolts have a minimum yield strength of 181 kN and are driven vertically on 1.2-m centers into loose muck on the floor. Prior to placing the fill, a 15- by 15-cm bearing plate and nut are placed at the top of the bolt. When the end of the bolt driven into the loose muck is exposed during mining of the following cut, chain link fencing, a bearing plate, and a nut are installed for ground support. When the bottom nut is tightened, it creates a tension load in the bolt and a compressional load in the backfill between the two bearing plates.

INSTRUMENTATION PLAN

The instrumentation plan was designed to determine closure of the mine openings, pressure in the backfill, strength of the fill, and induced loads on the vertical Dywidag bolts. Because support is provided by the mechanical interaction among these elements, a complete understanding of the engineering parameters of each element was needed to understand how the whole system functions.

Failure of the instruments from deformation and breakup of the backfill was anticipated, so redundant systems were used to measure reinforcement bolt loads and wall closure rates. Stope wall closure and fill pressure had been monitored previously at the mine (Williams et al. 1992; Hedley 1993). Loads on the backfill reinforcement bolts had never been monitored before, however, so instrumented No. 7 Dywidag rock bolts were obtained from Roctest, Plattsburgh, NY, to measure induced load on the rock bolts. Roctest installed a vibrating-wire strain gauge and thermister in one end of the bolt. The instrumented bolts were used to replace some of the vertical bolts placed on 1.2-m centers throughout the stope. The bolts were installed so that the instrumented end was in the fill to protect it from blasting as the next cut was mined.

Anclo 10 load cells were also obtained from Roctest to provide redundant readings on the vertical bolts. They were installed on the lower end of the instrumented bolts (figure 2) after mining had passed. The load cells survived better than the instrumented bolts because the wires were not subjected to fill deformation.

At four locations, wall-to-wall closure was measured in the 1-m-high gap above the backfill with string potentiometers, within the backfill with string potentiometers inside collapsible steel casings, and in the new stope cut with a tape extensometer. All closure readings were taken between 15- by 15-cm bearing plates on 1.2-m-long rock bolts to achieve as much accuracy as possible.

Loads in the fill were measured with 690-kPa, 23-cm-diam Model 3500 earth pressure cells manufactured by Geokon, Inc., Lebanon, NH. A wall closure instrument and an instrumented vertical bolt were installed at the same location as the earth pressure cell. Figure 3 shows the location and types of instruments installed in the stope.

A CR10 datalogger manufactured by Campbell Scientific, Inc., Logan, UT, was chosen because of its ability to record information from a number of instrument types (Seymour et al. 1998; Larson et al. 1995; Larson and Maleki 1996), including vibrating-wire strain gauges and thermistors, load cells, pressure transducers, and

---

[3] Mention of specific products and manufacturers does not imply endorsement by the National Institute for Occupational Safety and Health.

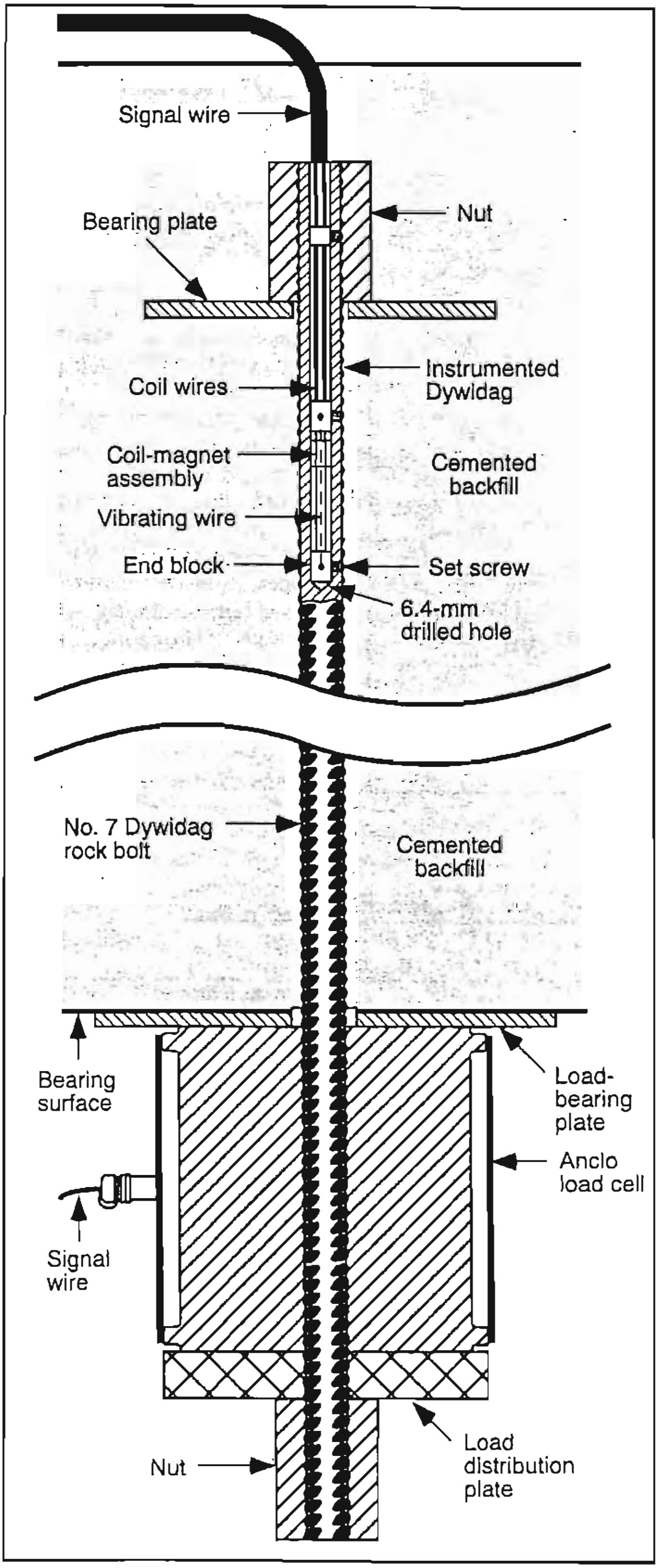

Figure 2.—Instrumented bolt with load cell on bottom

voltage potentiometers. The instruments were initially monitored hourly, then every 2 hr, and, by the end of the project, every 12 hr. The monitoring rate was changed to lengthen the time the system would operate without having to retrieve the data storage canister. An Excel spreadsheet was used for data analysis.

After the instruments were calibrated at SRL, they were installed in the mined-out cut 8 of the 5660-05 stope prior to placement of the backfill. Monitoring began at 7:00 p.m. on October 15, 1997. The stope was filled for 2.3 m of its 3.3-m height. Additional string potentimeters were installed wall-to-wall in the gap above the fill on October 16. The datalogger was then moved to a position in the gap so it would not be damaged by blasting during mining of cut 9.

Mining the next cut (5660-05 stope, cut 9) began by taking up the bottom of the ramp on October 20. The load cells were installed on the exposed ends of the bolts on October 29 after mining had proceeded far enough so that the instrument wires would not be damaged by blasting. Monitoring of the instruments continued to April 2, 1998, as three successive cuts were mined below the instrumented backfill location.

DATA ANALYSIS

Closure Readings

Stope wall closure readings were taken in the mining cut, across the backfill, and in the gap above the backfill to determine horizontal convergence of the stope walls as mining progressed. String potentiometers placed in the backfill showed the vein walls had converged an average of 7.90 cm during mining of cut 9 on the 5660 level, 8.94 cm during mining of cut 1 on the 5750 level, and 8.00 cm during mining of cut 2 on the 5750 level. At the same time, closure measured in the gap area averaged 14.00 cm for cut 9 on the 5660 level, 11.50 cm for cut 1 on the 5750 level, and 9.68 cm for cut 2 on the 5750 level. Increased amounts of closure

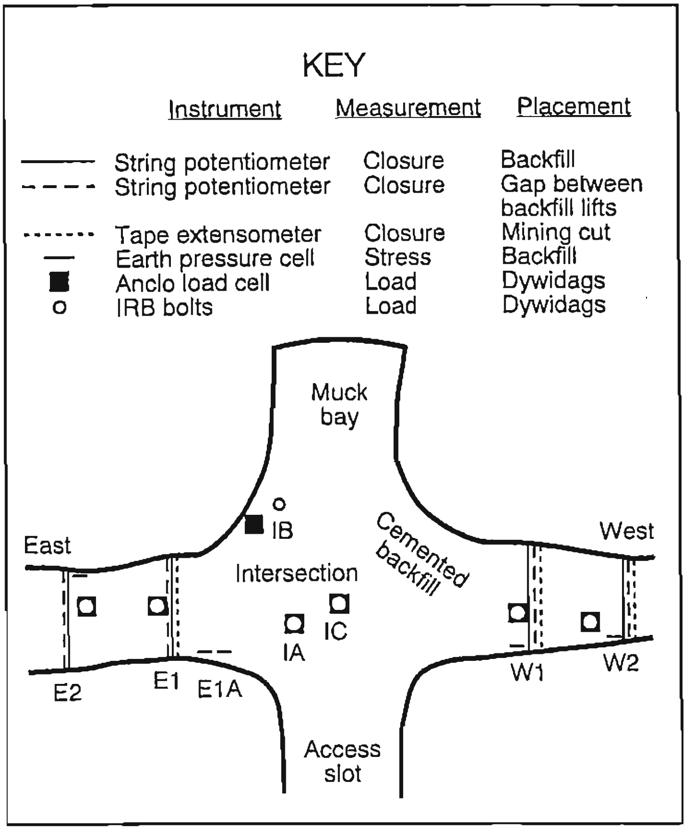

Figure 3.—Instrument types and locations

across the gap were caused by a lack of backfill support for the walls in this area. In the active mining area of cut 9, a tape extensometer recorded an average of 9.42 cm of closure. The measurements in the active mining area do not represent total closure related to mining because measurements began after mining was at least 9 m past the instrument locations.

Visual observations of the top of the fill indicated that the fill buckled rapidly as the walls closed. Buckling continued until the initial 1-m gap was reduced to less than 0.3 m before the fill from above collapsed. Table 1 is a summary of closure readings for the three mining cuts. Figure 4 shows backfill closure as a function of time. Total closure includes some rapid closure as the cut was mined past the instruments and more gradual time-dependent closure resulting from all previous mining in the area.

Table 1.—Closure readings, cm

Instrument and location	5660 level, cut 9	5750 level, cut 1	Cuts 9 + 1	5750 level, cut 2	Cuts 9+1+2
Backfill string pots:					
West 1	7.77	11.28	19.05	12.19	31.24
West 2	*4.65				
East 1	12.14				
East 2	3.78	6.63	10.16	3.81	14.22
Gap string pots:					
West 1	12.40	11.05	23.44		
West 2	11.33				
East 1	16.36	11.86	28.22	9.68	37.90
East 2	15.80				
Slot 1	8.71				
Tape extensometer:					
West 1	9.12				
West 2	12.42				
East 1	*6.76				

* Instrument quit working.

Cemented Backfill Specimen Lab Tests

At the time the stope was backfilled, samples of the fill were collected for compressive strength tests. The 7-day unconfined compressive strengths ranged from 1,703 to 2,744 kPa, with an average of 2,082 kPa, while 28-day unconfined compressive strengths ranged from 2,654 to 3,902 kPa and averaged 3,254 kPa. The laboratory tests agreed with the 2,757-kPa ultimate strength recorded in the stope at approximately 15 days.

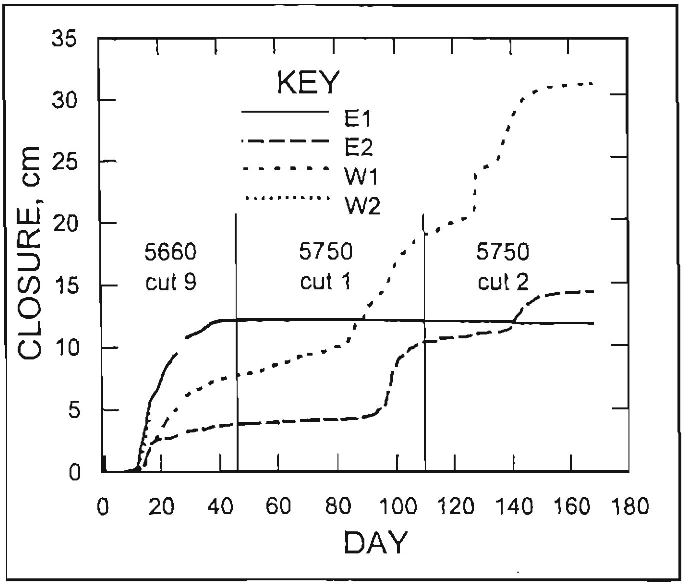

Figure 4—Backfill closure versus time

Movement of the platten head of the compressive test machine was also recorded to determine an apparent modulus for the 7- and 28-day compressive tests. Figure 5 is an example of the stress-strain curve for the tests showing the points for 20% and 50% of ultimate strength. The modulus values for the samples were calculated between the 20% and 50% strength values because this is the straight line portion of the curve and is most representative of the true response of the backfill. To calculate modulus, strength at 20% is subtracted from strength at 50%. The resulting strength value is divided by the value achieved when strain at 20% is subtracted from strain at 50%. [That is, (strength at 50% - strength at 20%) ÷ (strain at 50% - strain at 20%) = modulus.] The five samples tested at 7 days had a range of apparent modulus from 593 to 2,013 MPa with an average of 1,041 MPa. The 2,013-MPa modulus was nearly double the next highest modulus of 1,048 MPa. If the highest value were dropped, average modulus would be reduced to 800 MPa.

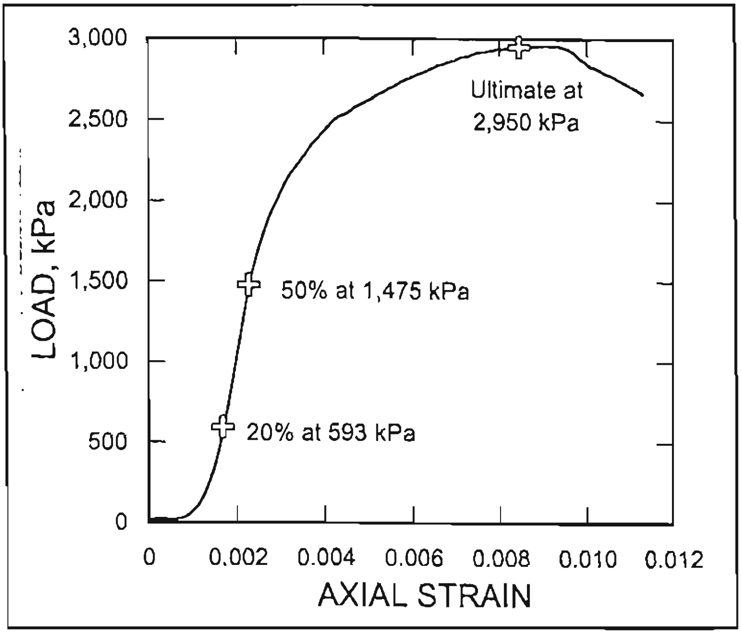

Figure 5.—Axial strain versus load

Apparent modulus for the four 28-day tests ranged from 1,172 to 1,641 MPa and averaged 1,370 MPa.

Backfill Pressure

Readings from the earth pressure cells peaked as soon as mining of cut 9 passed the instrument position (figure 6). Pressure drops were then recorded as each cut was completed. Peak pressures ranged from a low of 1,675 kPa at West 1 to 4,178 kPa at West 2 with an average pressure across the vein of 2,757 kPa at 15 days. The 20%-50% modulus determined at locations E1, E2, W1, and W2 were 681, 1,513, 2,361, and 5,095 MPa, respectively. The average was 2,412 MPa.

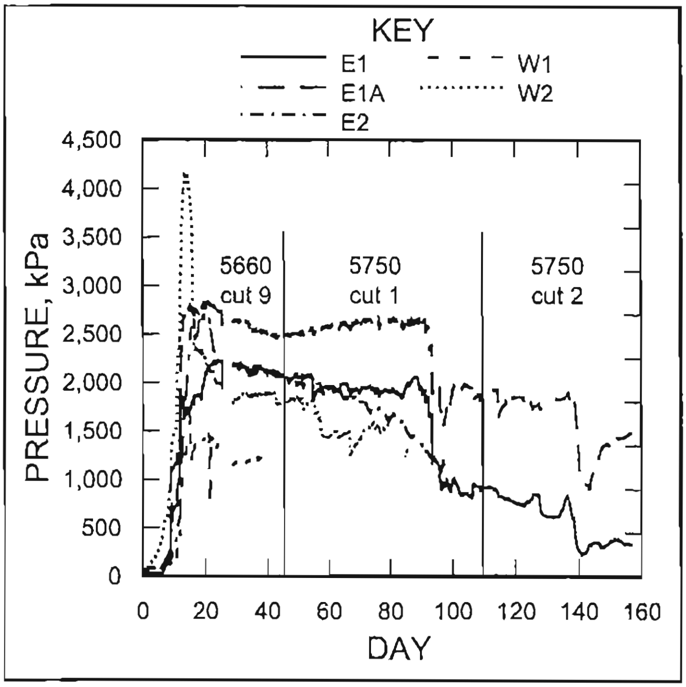

Figure 6.—Backfill pressure versus time

Strain was determined by dividing measured wall-to-wall closure by the original width of the opening. These modulus values were documented 8 to 12 days after the fill had been poured and as mining of the following cut passed the instrument locations. These data show that the ultimate strength of the cemented backfill was surpassed by loading induced by the large amounts of wall closure experienced in the stope. Thus, rock bolt reinforcement is needed to maintain the integrity of the backfill so that it will be safe for miners to work under.

Areas of broken fill were observed in the chain link fencing below the fill, and fill heaving was seen at the top. The fill eventually broke up until it could no longer carry load and support itself. It then collapsed onto the fill below. Because mining was usually two or more cuts below the collapsed backfill, there was no danger to miners.

Instrumented Reinforcement Bolts

The instrumented bolts and load cells were initially calibrated in a Tinius-Olsen testing machine at SRL to 53 kN; however, readings taken at the mine with bolts calibrated to this figure indicated that the yields and ultimate strengths of the bolts had been exceeded. To determine if the initial readings were accurate, two instrumented bolts were tested in tension to failure, and the data from these tests were used to calibrate the bolts. The instrumented bolts yielded at 160 kN and failed at 209 and 221 kN at the gauge location. Gauge response was linear to 124 kN, and the gauge failed at 142 kN prior to reaching the yield point of the bolts.

Six of the nine instrumented bolts placed vertically in the fill exceeded loads of 142 kN, which was the limit of the vibrating-wire gauge. Information from most of the instrumented bolts was eventually lost because deformation of the backfill apparently broke many of the signal wires. The data show that the vertical reinforcing bolts in the backfill did a good job of resisting fill deformation and provided a safe back for miners.

Load Cells

Data from the load cells indicated that the bolts were under loads from 41 to 179 kN after cut 9 was mined. Figure 7 shows the relationship of bolt load to closure as measured by the fill string pot. The figure indicates that bolt load leveled off and

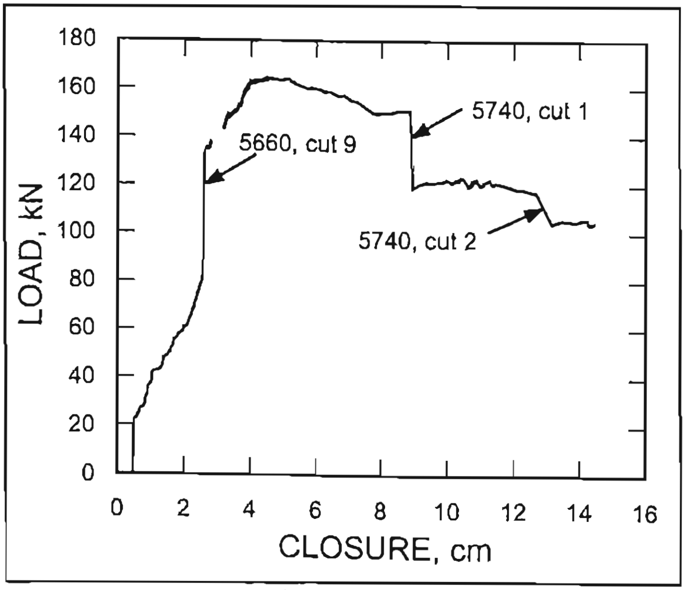

Figure 7.—Bolt load versus closure, site E2

decreased as closure continued following each mining cut. This behavior was caused by fill failure around the bolt bearing plates as the walls converged. Inspection of the bottom of the fill revealed bags of broken fill in the chain link fencing between the rock bolt plates. Visual observation of the top of the 2.4-m-long vertical bolts confirmed that the 15- by 15-cm bearing plates resisted fill deformation and transferred load to the rock bolt. Table 2 is a summary of the loads recorded on the instrumented bolts and load cells.

Table 2.—Loads on instrumented bolts and load cells for three cuts of mining, kN

Location	5660-05 level, cut 9		5750-05 level, cut 1		5750-05 level, cut 2	
	Bolt	Load cell	Bolt	Load cell	Bolt	Load cell
West 1	63.1	41.5	50.3	34.1	NA	25.3
West 2	*142	116	NA	19.4	NA	9.9
East 1	*142	179	NA	157	NA	133
East 2	*142	157	*142	122	NA	104
Intersection A	*142		*142	NA	*142	NA
Intersection B	103	164	53.3	1.8	43.3	NA
Center	*142	113	*142	104	*142	NA

* Instrument out of range or quit working. NA = Not available.

DISCUSSION

A thrust-fault-type of fracture was noticed at the top of the backfill while retrieving the data canister on November 4, 1997. Both 1.8- and 2.4-m-long vertical bolts had been placed in the area. Failure appeared to be above the 1.8-m bolts but blocked by the 2.4-m bolts, the ends of which were visible at the top of the fill. This failure was caused by convergence of the vein walls on the fill. At this time there had been 2.5 to 7.6 cm of horizontal closure measured in the fill and 7.6 to 15.2 cm measured in the gap. Fracturing of the fill caused it to buckle upward and gradually reduce the gap between the fills over time.

On March 1, loads on the vertical bolts on the west side decreased by 9.8 and 4.4 kN. This indicates that as the fill failed nearby, horizontal pressures creating the load on the vertical bolts were reduced. Visual inspection of the intersection on March 3 showed that the fill in the northwest corner of the intersection had collapsed onto the cut 9 fill. It was not possible to determine how far the failure extended along the west side of the stope, but it is thought that, based on instrument response, the failure did not reach the West 1 location. At this time, the active mining face was 9 m below with two backfill horizons between it and the failed backfill, so the failure posed no hazard to the miners.

Figure 8 is an idealized drawing of how the instrumented reinforcement bolts interact with the cemented backfill in the vein portion of the stope as mining progresses. This interpretation is based on conventional theories of rock bolt reinforcement that state that tensioned rock bolts create a self-supporting compressive arch across the opening (Lang 1961; Hoek and Brown 1980; Stillborg 1986; Brady and Brown 1993). The illustration is also based on visual observations, closure measurements, and load readings on the vertical reinforcing bolts. The forces in the fill, created as the bearing plates resist deformation, form a cone of compression in the fill. The fill is self-supporting until the zone of breakup eliminates the overlap of the cones of compression between adjacent vertical roof bolts. The fill then collapses because of gravity.

CONCLUSIONS

Data from an instrumentation project carried out in cut 8 of the 05 stope on the 5660 level of the Lucky Friday Mine indicated that wall closure caused deformation of the backfill and induced loads in the vertical Dywidag bolts used as backfill reinforcement.

An interpretation of the interaction among wall closure, fill deformation, and induced loads in the vertical bolts in the cemented backfill is presented in figure 8 and indicates how the reinforced backfill support system may work. This knowledge is important for designing backfill support systems for other mines to ensure the safety of miners working in underhand stopes.

ACKNOWLEDGMENTS

The authors would like to thank Frank Reed of Hecla Mining Company and Rich Rains, Mark Larson, Paul Pierce, and Mike Jones of the Spokane Research Laboratory for their assistance during the project. Without their help, the project would not have been completed.

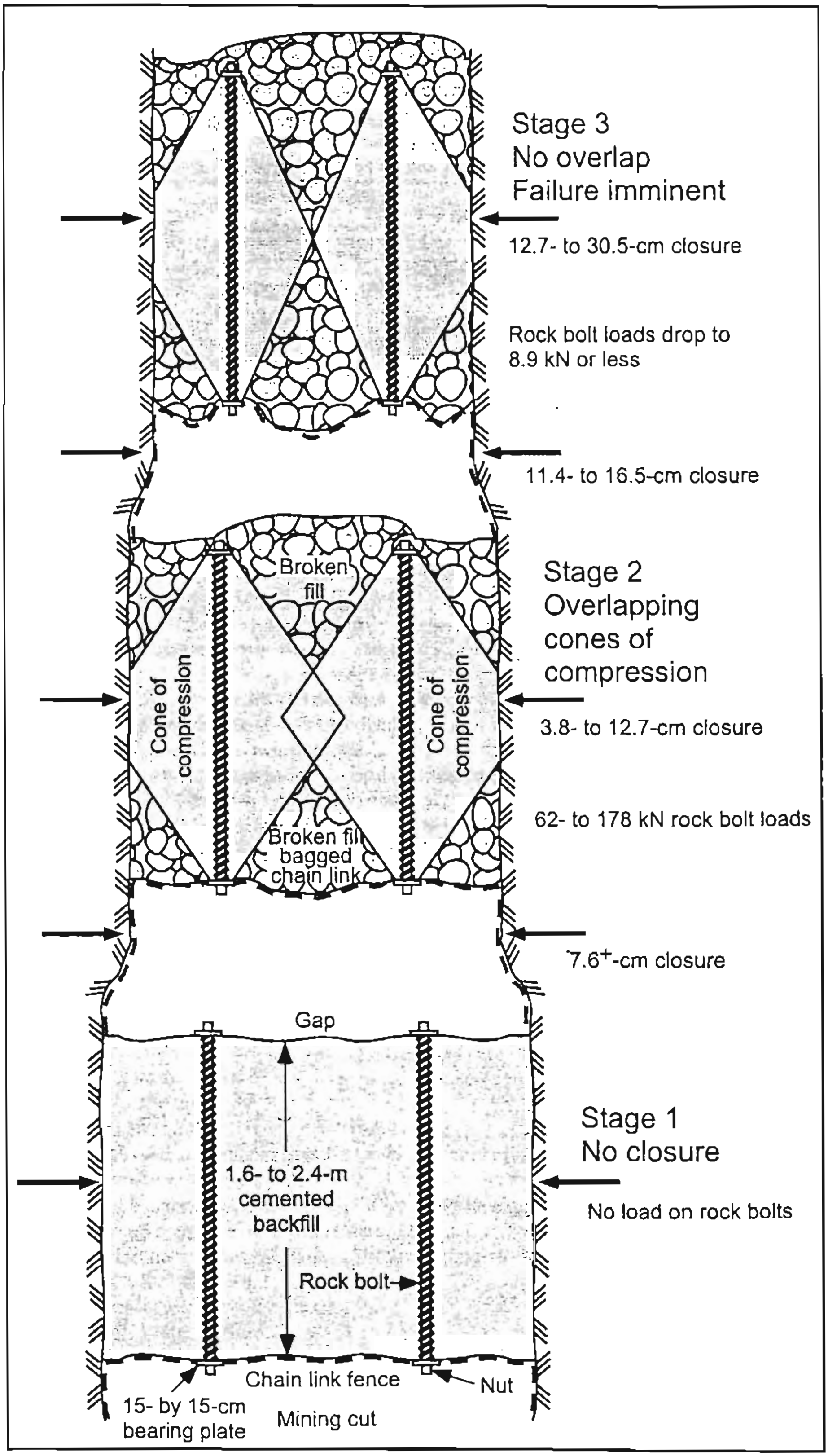

Figure 8.—Theory of backfill support

REFERENCES

Brackebusch, F.W. 1994. Basics of paste backfill systems. *Min. Engin.*, Oct.: 1175-1178.

Brady, B.H.G. and E.T. Brown. 1993. *Rock Mechanics for Underground Mining*, 2nd ed. Chapman and Hall, London.

Hedley, D.G.F. 1993. Properties of cemented paste backfill at Hecla's Lucky Friday Mine. Report on Stiff Backfill Project of Canadian Rockburst Research Project, 13 pp.

Hoek, E. and E.T. Brown. 1980. Underground excavations in rock. Inst. Min. and Metall., London, UK.

Lang, T.A. 1961. Theory and practice of rock bolting. *Trans.* 220:333-348.

Larson, M.K. and H. Maleki. 1996. Geotechnical factors influencing a time-dependent deformation mechanism around an entry in a dipping seam. Paper in *Proceedings, 15th International Conference on Ground Control in Mining*, ed. by L. Ozdemir, K. Hanna, K. Y. Haramy, and S. Peng (Golden, CO, Aug. 13-15, 1996). CO School of Mines, 1996, pp. 699-710.

Larson, M., C. Stewart, M. Stevenson, M. King, and S. Signer. 1995. A case study of a deformation mechanism around a two-entry gateroad system involving probable time-dependent deformation. Paper in *Proceedings of 14th International Conference on Ground Control in Mining*, ed. by S. S. Peng (Morgantown, WV, Aug. 1-3, 1995). Dept. of Mining Engineering, WV Univ, 1995, pp. 295-304.

Scott, D.F. 1990. Relationship of geologic features to rock bursts, Lucky Friday Mine, Mullan, Idaho. Paper in *Rockbursts and Seismicity in Mines. Proceedings of the 2nd International Symposium on Rock Bursts and Seismicity in Mines*, ed. by C. Fairhurst (Minneapolis, MN, June 8-10, 1988). Balkema, 1990, pp. 401-406.

Seymour, B., D. Tesarik, M. Larson, and J. Shoemaker. 1998. Stability of Backfilled Cross-Panel Entries During Longwall Mining. Paper in *Proceedings of 17th International Conference on Ground Control in Mining*, ed. by Syd S. Peng (Univ. of WV, Morgantown, WV, Aug. 4-6, 1998), pp. 11-20

Stillborg, B. 1986. *Professional Users Handbook for Rock Bolting*. Trans Tech Publi. Series on Rock and Soil Mechanics, Vol. 15.

Whyatt, J.K. and M.J. Beus. 1995. In situ stress at the Lucky Friday Mine (in four parts): 1. Reanalysis of overcore measurements from 4250 level. U.S. Bur. Mines Rep. of Invest. 9532, 1995, 26 pp.

Whyatt, J.K., T.J. Williams, and W. Blake. 1995. In situ stress at the Lucky Friday Mine (in four parts): 4. Characterization of mine in situ stress field. U.S. Bur. Mines Rep. of Invest. 9582, 1995, 26 pp.

Williams, T.J., J.K. Whyatt, and M.E. Poad. 1992. Rock mechanics investigations at the Lucky Friday Mine (in three parts): 1. Instrumentation of an experimental underhand longwall stope. U.S. Bur. Mines Rep. of Invest. 9432, 1992, 26 pp.

MINEFILL 2001:

PROCEEDINGS OF THE

7TH INTERNATIONAL SYMPOSIUM ON MINING WITH BACKFILL

Edited by David Stone, P.E.

Published by the
Society for Mining, Metallurgy, and Exploration, Inc.

This proceedings volume sponsored by

LaFarge Canada Inc.

Cover photo courtesy of The Doe Run Company, Viburnum, Missouri.

Society for Mining, Metallurgy, and Exploration, Inc. (SME)
8307 Shaffer Parkway
Littleton, Colorado USA 80127
(303) 973-9550 / (800) 763-3132
www.smenet.org

SME advances the worldwide minerals community through information exchange and professional development. SME is the world's largest association of minerals professionals.

Disclaimer
The papers contained in this proceedings volume are published as supplied by individual authors. Any statement or views presented here are those of individual authors and are not necessarily those of the Society for Mining, Metallurgy, and Exploration, Inc. The mention of trade names for commercial products does not imply the approval or endorsement of SME.

ISBN 0-87335-211-4