VENTILATION PLANNING AND DESIGN OF THE SKYLINE MINES

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This paper describes a case study for the ventilation planning at Utah Fuel Company's Skyline Mines which are located outside Helper, Utah. The Skyline Mines comprise of two longwall coal mining operations and a third longwall mine currently being developed. Future ventilation systems for the three mines were determined by establishing ventilation models developed from data collected during a detailed ventilation survey. The paper discusses the challenge to coordinate immediate ventilation requirements with long-term demands. The continuous movement of the longwall, which frequently represented the only open split in the ventilation network, resulted in cyclic ventilation demands dependent upon the length of the tailgate. A series of system configuration changes were recommended based on the results of the network modeling exercise and an economic evaluation of the various options. The scope and cost of the configuration changes, and the measured improvement resulting from these changes is discussed.

INTRODUCTION

Utah Fuel Company's Skyline Mines consist of two established longwall mines, termed the Skyline #1 and Skyline #3 Mines. Each mine is totally independent and comprises of a single retreating longwall with two or three continuous miner sections. A third longwall mine, the Skyline #2 Mine, is currently being developed off the mains of Mine #3. The current length of the longwall panel in Mine #1 is 2865 m (9,400 ft), the width is 223 m (730 ft), the height varies from 2.6 to 3.1 m (8.5 to 10 ft), and the total projected recovery from the panel is approximately 2 million tons. The current length of the longwall panel in Mine #3 is 2256 m (7,400 ft), the width is 213 m (700 ft), the height is 4.1 m (13.5 ft), and the total projected recovery from the panel is approximately 3 million tons. The total annual production from the Skyline Mines is 4 to 5 million tons, which consistently ranks Skyline Mines within the top five US mines for underground production tonnage. Utah Fuel Company is a subsidiary of Coastal Corporation, which mines in excess of 15 million tons of coal per year.

A critical stage in the development of the Skyline Mines has been reached. The final longwall panel from Mine #3 will have been mined by the end of 1997 and longwall production from Mine #2 will commence following this date. The order that the longwall panels are being mined for Mine #3 is such that the development areas are retreating back toward the mine portals and main fan. However, the
Mine #2 workings are being developed from Mine #3, which means that at least two additional continuous miner headings have to be ventilated off the present Mine #3 ventilation system.

To aid in the planning and development of the ventilation systems for the three mines comprehensive ventilation surveys were conducted for Mines #1 and #3 in March 1993. Accurate ventilation models were constructed from the survey data, and during 1993 and early 1994 these models were used for a series of future design analyses. Based on the results of these analyses recommendations for changes to the ventilation systems of the mines were made.

VENTILATION SURVEY AND BASIC COMPUTER MODELS

In order to accurately model the existing mine ventilation systems it was necessary to evaluate the resistance to airflow for all main airways. This was accomplished by measuring the airflow quantity and pressure differential in each airway. Resistances were evaluated using the Square Law relationship. Ventilation surveys were conducted for Mines #1 and #3 to obtain frictional pressure drop data and corresponding air quantity values for the main airways and leakage routes in the mines. The air quantity survey consisted of the measurement of mean air velocities and airway cross-sectional areas at predetermined locations in the underground. A rotating vane anemometer attached to an extendible rod was used to traverse the airways for measurement of the mean air velocity. Traverses were repeated until three readings were obtained within ±5 %. The airway cross-sectional areas were measured using steel tapes. The air quantities at each station were computed as the product of the air velocity and the airway cross-sectional area. The airflow quantities were checked in the field for adherence with Kirchhoff’s First Law, namely that the sum of airflow entering a junction was equal to the sum of airflow leaving the junction (practically this is within ±5 %). The frictional pressure drops along the main mine airways were determined using the gauge-and-tube technique. The gauge-and-tube (or trailing hose) method allows direct measurement of pressure differential using a manometer connected into a length of tubing, the ends of which are connected to the total pressure tappings of pitot-static tubes. Static pressure drops were taken across regulators, doors and stoppings wherever possible. The pressure differential data were checked for adherence with Kirchhoff’s Second Law, namely that the sum of the pressure drops around a closed circuit equated to zero (practically this is within ±10 %).

The VNETPC™ ventilation simulation computer package was used to develop and manipulate basic networks for each mine. In order to develop the computer ventilation models skeleton networks were constructed from the main airway infrastructure. Additional branches, representing leakage paths, were added as required. Resistance values for each branch in the network were computed directly from measured data and input to the model. Fans were input to the model at fixed pressures measured during the mine ventilation survey. Psychrometric data were used to calculate the natural ventilation pressures (NVP) existing at each mine. Measurements of barometric pressure, dry bulb temperature and relative humidity were taken between intake and return airways and graphs of specific volume against barometric pressure were plotted for closed loops around the ventilation circuit (including the main fan) which were used to evaluate NVP.
Mine #1 had propane heaters to heat the intake air, which resulted in very little additional heating of the air as it passed through the mine workings. Hence, the density difference between intake and exhaust air for Mine #1 was very small and NVP was insignificant. NVP was significant in Mine #3 which had no propane air heaters. The dry bulb temperature of the intake air during the survey was about 2 °C (36 °F), which increased to nearly 14 °C (57 °F) as the air passed through the ventilation circuit. The computed NVP for Mine #3 was +87 Pa (+0.35 inch w.g.). This NVP was modeled as a fixed pressure booster fan in the returns.

In order to ensure that the basic network models were true representations of the ventilation systems correlation exercises were conducted for each mine. The overall correlation for a network is defined from the relationship in Equation 1.

\[
\frac{\sum \text{absolute value} \left(t_{\text{differences between predicted and actual airflows}}\right)}{\sum \text{absolute values measured airflows}} \times 100 \quad (1)
\]

The basic models for each mine correlated within 3%.

Ventilation Design Parameters

In bedded deposits with multiple openings and crosscuts, resistance per unit length measurements provide excellent ventilation parameters for future design modeling. Representative airway lengths are chosen for concurrent pressure and airflow quantity measurements. From this data a resistance per unit length is determined. Table 1 lists the resistance per unit length data gathered during the ventilation survey.

Leakage paths in the model usually represent a number of stoppings or seals along an airway. The resistance of the leakage can be determined by measuring the resistance for a typical stopping and calculating the equivalent resistance for a number of these stoppings in parallel. The resistances measured for certain ventilation controls at the Skyline Mines are given on Table 2. Stopping resistance varied considerably according to the condition and installation of the stopping, and a resistance range has been given for the stoppings measured at the mines.
### Airway Type

<table>
<thead>
<tr>
<th>Airway Type</th>
<th>Resistance Per Meter (Ns²/m⁸ × 10⁻⁶)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1 intake - roadway</td>
<td>28.91</td>
</tr>
<tr>
<td>3 intakes in parallel (new)</td>
<td>3.85</td>
</tr>
<tr>
<td>3 intakes in parallel (old)</td>
<td>8.07</td>
</tr>
<tr>
<td>1 inside return</td>
<td>30.01</td>
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<tr>
<td>1 outside return</td>
<td>42.19</td>
</tr>
<tr>
<td>2 returns in parallel</td>
<td>8.84</td>
</tr>
<tr>
<td>2 returns (1 cribbed)</td>
<td>16.99</td>
</tr>
<tr>
<td>1 cribbed return</td>
<td>274.81</td>
</tr>
</tbody>
</table>

Table 1: Resistance Per Unit Length Parameters For Main Airways

### Description

<table>
<thead>
<tr>
<th>Description</th>
<th>Resistance (Ns²/m⁸)</th>
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</thead>
<tbody>
<tr>
<td>Single Overcast</td>
<td>0.0049</td>
</tr>
<tr>
<td>Double Overcast</td>
<td>0.0107</td>
</tr>
<tr>
<td>Triple Overcast</td>
<td>0.0296</td>
</tr>
<tr>
<td>Kennedy Stopping</td>
<td>1090 - 1678</td>
</tr>
</tbody>
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Table 2: Resistance Values for Ventilation Structures

### FUTURE DESIGN ANALYSES

**Skyline Mine #1**

A number of ventilation models representing future production and mine configurations were developed for Mine #1. From these models a series of alterations to the ventilation system were recommended which were deemed necessary for the provision of adequate ventilation throughout the remaining life of the mine (estimated at 6-7 years). Prior to conducting the future design analyses it was feared that major changes to the ventilation system would be required, which included the possibility of mining a new return ventilation raise and/or the replacement of the main mine fan. With a series of configuration changes neither of these relatively high cost solutions were necessary. Figure 1 shows the ventilation schematic established for Mine #1 from survey data. At the time of the survey the mine had a single longwall, with two continuous miner sections. The mine had a single intake fan with split return portals.

Future development of Mine #1 is away from the fan such that the development areas are becoming increasing difficult to ventilate, and the bleeder system more complex and harder to maintain. The same type of mine fan is used in both mines, which is a 3.4 m (11 ft) diameter, 720 rpm Spendrup. The electric motor for the fan is rated at 447 kW (600 hp). At the time of the survey the main mine fan was intaking 182 m³/s (386 kcfm) at a fan total pressure of 1470 Pa (5.9 inch w.g.). The airflow across the longwall face was 20 m³/s (43 kcfm) and the intake airflow at the end of the
two miner sections was 22 m$^3$/s (47 kcfm) and 26 m$^3$/s (54 kcfm). Although there was not a miner section at the end of the mains, 19 m$^3$/s (40 kcfm) was split around this section. The total quantity intaking the bleeder was 20 m$^3$/s (42 kcfm).

The basic model for Mine #1 was amended using mine plans and schedules to represent the proposed mine layout at future dates. Certain ventilation requirements were established by mine personnel, which are listed in Table 3. By manipulating the various models representing future development of the mine, the system configurations necessary to obtain the desired airflows were determined.

<table>
<thead>
<tr>
<th>Location</th>
<th>Air Quantity (m$^3$/s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Longwall Face</td>
<td>33</td>
</tr>
<tr>
<td>Continuous Miners</td>
<td>24</td>
</tr>
<tr>
<td>Bleeder Seals</td>
<td>7-9</td>
</tr>
</tbody>
</table>

Table 3: Ventilation Requirements for the Skyline Mines

Longwall Ventilation In many longwall operations the longwall tailgate is the only open split in the ventilation network, which is required in order to maximize the airflow across the longwall face. This practice is often required when the mine has developed sufficiently such that the desired airflow requirements across the face are not being met, and it becomes necessary to regulate the remaining splits in the system to force air around the longwall circuit. The problem that can result is increased pressure drops across stoppings with increased leakage throughout the system, and frequently a decrease in the volumetric efficiency of the mine.

At Mine #1 the length of the tailgate varies from a maximum of 2865 m (9,400 ft) at the start of the panel to about 305 m (1,000 ft) immediately prior to recovery. Table 1 lists the resistance per unit length data measured during the ventilation survey, and it can be seen that the resistance of a single cribbed tailgate is about six times greater than that of a regular outside return airway. The resistance of the tailgate branch in the ventilation network varies from a maximum of 0.7873 Ns/m$^8$ (0.7040 P.U.) to a minimum of 0.0838 Ns/m$^8$ (0.0750 P.U.). Assuming that the tailgate is the only route for the air to return from the longwall face, the pressure drop incurred when a quantity of 33 m$^3$/s (70 kcfm) passes through the tailgate is 857 Pa (3.5 inch w.g.) and 91 Pa (0.4 inch w.g.) for the long and short panel configurations respectively. With the present fan motor it is not possible to deliver sufficient quantity across the longwall face with such a high pressure drop in the tailgate (long panel configuration). In the past this has resulted in poor airflow across the face during the mining of certain sections of the panel. The design of the ventilation system is improved, but further complicated by the tailgate not being the only return from the longwall face. A considerable quantity of air can return via the uncaved portion of the tailgate in the GOB and through the bleeder system, depending on the extent that the cribbed tailgate caves. In addition, there is another connection from the longwall to the bleeder system that can provide an alternative route for air to return the longwall. About one-third of the way into the panels (from the set-up room) a dike cuts through the coal seam which necessitates the movement of the shearer around this fault (Figure 1). Because a new set-up room is established in each panel there exists a partially open airway through the old panels, in a direction perpendicular to
the tailgate, to the bleeder system. The tailgate intercepts the set-up room of the last panel, and until
the longwall face retreats past this connection as much as 30-50% of the longwall air can return via
this route (estimated by mine personnel during the mining of previous panels). When the longwall face
has retreated past this point, and the tailgate in the GOB has caved, all the air must return via the open
tailgate. Even though over one-third of the panel has been mined, this can represent the worst-case
scenario for ventilating the longwall.

Ventilation System Adjustments  From the ventilation survey two main areas were established as high
resistance branches in the ventilation. The first location was at the intake portal, and the second area
was the longwall tailgate.

The pressure drop measured at the mine portal was 473 Pa (1.9 inch w.g.), for a total intake
quantity of 182 m$^3$/s (386 kcfm). This pressure drop was measured from a location directly
"downstream" of the mine fan, around a 90° bend and about 152 m (500 ft) down a single intake entry
to a point where the intake splits to parallel entries. This pressure drop was considered too great,
because, of the 1470 Pa (5.9 inch w.g.) fan ventilating pressure, 32% was accounted for in moving
the air past the first five cross-cuts in the mains. To reduce this pressure drop a new intake portal was
designed and installed which incorporated dual intake airways from the surface portal inbye. A right-
angled bend with explosion door was maintained (a standby fan was considered too expensive), but
the shock loss associated with the bend was reduced by ensuring that the transition from the fan drift
to the mains was much less severe and by maintaining both new airways free of debris. Another
benefit of the new portal was the reduction of leakage through the roadway equipment doors. These
high pressure interlocked doors provide the only egress for equipment into the mine, and had become
worn. The leakage measured during the survey was 8.5 m$^3$/s (18 kcfm) which has been reduced to
about 2.5 m$^3$/s (5 kcfm) with the new portal doors.

The second location where high pressure drops were measured was along the timber-cribbed
longwall tailgate. The timber cribbing was not providing a sufficiently large enough area for the air to
move through, and the resistance of the tailgate was considered too high. In addition to the poor
ventilation performance, escalating timber costs have made the use of wood roof support much less
attractive than was formerly the case. Because of these reasons Utah Fuel Company has recently
started testing and using alternative mine roof support methods. The most commonly used support at
the mine site is now Corrugated Confined Core or 3C supports. The 3C support has three separate
components consisting of a corrugated metal pipe, fill material and end caps to shore against the roof.
The supports used at the Skyline Mines are 1.1 m (42 inch) diameter and failure resistant across a
wide yield range. The support collapses like an accordion, without the buckling and associated airway
restriction common with timber supports. Prior to using the 3C supports an analysis was conducted to
determine the expected increase in airflow across the longwall face by using the 3C supports. The
resistance of a tailgate with two rows of 3C supports was estimated to be at least 30% less than that
of a timber cribbed tailgate. To further reduce the resistance of the tailgate at the mine detailed testing
of a cable-bolting system is currently being conducted.
A more minor change to the ventilation system was concerned with the airflow in the development belt drifts. During the development of the longwall headgate and set-up room the belt drift can become extremely long (greater than 3050 m [10,000 ft]) and problems have been encountered with airflow reversing direction in the belt drift due to the high amount of leakage across the stoppings. The Mine Safety and Health Administration (MSHA) decrees that the Skyline Mines must maintain airflow in the belt drift away from the working face, and in order to achieve this the belt drifts were established as low-pressure returns. By installing a belt-to-return regulator at the mouth of the section, installing belt-checks (regulators) periodically along the belt drift, and good maintenance of the stoppings, control of the air away from the face was achieved without losing too much air via leakage.

Skyline Mine #3

Figure 2 shows the ventilation schematic established for Mine #3. At the time of the survey the mine had a single longwall, single continuous miner section and an established bleeder system. The main mine fan was intaking 135 m$^3$/s (285 kcfm) at a fan total pressure of 1445 Pa (5.8 inch w.g.). The airflow across the longwall face was 29 m$^3$/s (61 kcfm) and the intake airflow at the end of the miner section was 26 m$^3$/s (55 kcfm).

The basic model for Mine #3 was amended using mine plans and schedules to represent the proposed mine layout at future dates. Two models were developed for Mine #3 to represent the extent of mine development for the time frames of January 1994 and January 1995. The 1994 model had a single longwall and two miner sections and the 1995 model had a single longwall with three miner sections. The additional miner sections from the basic model case were required for the development of the new South Mains, which represents the start of a new mine (Skyline Mine #2). Mine #2 will be an independent mine, but the initial development of the mine is being conducted from Mine #3. The additional 47 m$^3$/s (100 kcfm) required to ventilate the two miners in the South Mains section had to be incorporated in the Mine #3 ventilation plan.

**Ventilation System Adjustments** In order to meet the ventilation requirements across the longwall and in the miner sections for both time frames certain ventilation system adjustments were necessary. As the number of development areas increased, the total face quantity increased by 47 m$^3$/s (100 kcfm). Because of the increased demand at the fan it was expected that a major change to the ventilation system would be required. Alternatives to replacing the mine fan motor or the entire fan were investigated and a series of more minor ventilation changes were established from manipulation of the ventilation models. The changes to the ventilation system consisted of:

- Belt checks in the longwall headgate, the new South Mains development section and the belt portal connection.
- Rehabilitation of 457 m (1500 feet) of the main returns.
- Better regulation of the intake air through the bleeder.
- Repair and replacement of damaged stoppings and doors throughout the mine.
- Utilizing 3C supports in longwall tailgate.
During the ventilation survey the volumetric efficiency (total face air quantity/delivered air quantity) was approximately 40%. To achieve the design air quantities for 1995 (on the ventilation model) it was necessary to maintain a volumetric efficiency of about 60%. Such a high efficiency will be very hard to obtain in practice and leakage throughout the mine will have to be reduced with the repair of any damaged stoppings or doors.

A concern from the ventilation survey was the high amount of leakage from the main intake to the conveyor belt drift to exhaust the mine via the belt slope (23 m³/s [49 kcfm]). This amount of leakage was not considered acceptable for future, multiple section mining situations and belt-checks were installed in certain locations to limit the leakage via the belt. Installing belt regulators reduces the pressure differential from intake to belt thereby reducing the leakage.

During the mine ventilation survey a high resistance section in the main return was noted where the air passed through a single, steel supported airway. Over this section the resistance of the airway was about four times higher than if two similar parallel returns were available for airflow. A pressure drop in excess of 249 Pa (1 inch w.g.) was measured along the restricted airway which meant that the return was regulating the airflow in the sections off the Mains. The impact of leaving the single return and using dual returns was analyzed for both the 1994 and 1995 models.

**COST DATA AND SYSTEM IMPROVEMENTS**

New Portal Installation for Mine #1

- Date construction commenced: August 23, 1993
- Date construction completed: November 13, 1993
- Cost for construction and engineering: $277,590
- Cost for electrical work: $5,500
- **Total cost for portal:** $283,090

Rehabilitation of Returns in Mine #3

- Date work commenced: September 1, 1993
- Date work completed: January 20, 1994
- Man-hour cost (labor + benefits): $30 per hour
- Equipment cost (capital + operating + maintenance): $70 per hour
- Equipment employed: 1 EIMCO loader, 1 Fletcher roof bolter
- Total man-hours at regular rate: 2120 hours
- Total man-hours at overtime rate (× 1.5): 20 hours
- **Total cost to rehabilitate returns:** $186,000
Cost for Wood Cribs

Number of rows of crib packs per airway: 2
Spacing of crib packs (center): 4.1 m (13 ft 6 inch)
Number of cribs per pack: 70
Size each crib: 20 cm x 20 cm x 1.2 m (8 inch x 8 inch x 4 ft)
Distance 3-man crew can install per shift: 30.5 m (100 ft)
Total cost: $311 per m ($95 per ft)

3C Supports

Number of rows of supports per airway: 2
Spacing of supports (center): 2.3 m (7 ft 6 inch)
Diameter of supports: 1.1 m (3 ft 6 inch)
Total cost per support (including fill): $250
Distance 3-man crew can install per shift: 45.7 m (150 ft)
Cost of machine (for support placement): $500 per shift
Total cost: $249 per m ($76 per ft)

The total cost for the two main system improvements (portal and return rehabilitation) was $469,090. The cost saving of using 3C supports instead of timber crib packs is $62 per m ($19 per ft), which equates to about $176,340 for each cribbed tailgate in Mine #1. If the cost of timber continues to increase (which is very likely) this saving will increase.

System Improvements

It is difficult to directly compare the ventilation system in a mine before and after configuration changes due to the airway infrastructure expanding as the mine develops. Table 4 lists the fan operating points measured before and after work was conducted in both mines.

<table>
<thead>
<tr>
<th>Description</th>
<th>Fan Operating Point</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Pressure (Pa)</td>
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<tr>
<td>Mine 1 Ventilation Survey - Feb. 1993</td>
<td>1470</td>
</tr>
<tr>
<td>New Portal - Dec. 1994</td>
<td>1296</td>
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<tr>
<td>VNETPC Prediction</td>
<td>1320</td>
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<tr>
<td>Mine #3 Ventilation Survey - Feb. 1993</td>
<td>1437</td>
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<tr>
<td>Rehab. Return/Min. Leakage</td>
<td>1445</td>
</tr>
<tr>
<td>- New Curve - Jan. 1994</td>
<td></td>
</tr>
</tbody>
</table>

Table 4: Change in Fan Operating Points

It can be seen that the addition of a new portal in Mine #1 lowered the operating point of the fan on the characteristic curve by 174 Pa (0.7 inch w.g.). For Mine #1 it was not thought that the system
resistance would have varied much from the survey date to December 1994, and a ventilation model developed for this date showed a fan pressure similar to that measured during the survey without the new portal. With the new portal the model predicted fan pressure and quantity within 2% of the measured values as shown on Table 4. The increase in airflow resulting from the new portal was 19 m$^3$/s (39 kcfm), which resulted in an increase of almost 9 m$^3$/s (20 kcfm) at the working sections. While this is not a large increase (10.5%) it appears that there is now sufficient air being delivered to the sections to exclude the necessity for installing a raise or new fan. The air power cost for operating the fan each year is reduced slightly ($4,000) but this is not a significant factor compared to the increase in ventilation.

A number of configuration changes were conducted in Mine #3. The changes which had the largest impact were the rehabilitation of the return airway and the installation of 3C supports instead of timber cribs. The fan operating point does not provide a suitable comparison for the system improvement (see Table 4) because the fan blade setting was adjusted between the two measurement dates to provide more airflow in the system for the additional miner sections (Mine #2 development). The measured pressure drop across the section of return that was rehabilitated was 150 Pa (0.6 inch w.g.), which represents a 57% reduction in air power across this section. This saving amounts to about $10,200 per year for this airway, however it is the increased airflow in the sections that justifies the cost of rehabilitation and not the air power saving. Without reducing the resistance of the returns the ventilation analyses predicted that a new motor would have been required for the main fan in order to achieve the desired airflows. The cost of procuring and installing a new motor, and the increased air power cost would have exceeded the money spent on the rehabilitation work. There is no data available yet on the improvement in ventilation resulting from the use of 3C supports in the longwall tailgate. However, even if the resistance of the tailgate is not much lower, the 3C supports still represent a less expensive type of support than the timber crib packs.

SUMMARY

Using detailed ventilation survey data basic computer models were developed to represent mine development at the Skyline Mines. The ventilation models for Mine #1 and #3 correlated to measured airflow data within 3%. From manipulation of the basic models future airway configurations were simulated and certain ventilation system adjustments were implemented.

The cost of installing a new portal in Mine #1 was high ($283,090), however the benefits are significant. The resistance of the new portal configuration is considerably less than the previous design which results in greater than 10% increase in quantity through the fan. In addition, egress into the mine is improved with the new portal configuration and leakage through the airlock portal doors is reduced. The additional fan quantity and reduced leakage result in about 25 m$^3$/s (53 kcfm) more air past cross-cut #5 in the intake airways. The increased fan capability should provide sufficient ventilation for the remaining life of Mine #1 without the need for a return ventilation shaft or replacement of the fan. The cost of rehabilitating Mine #3 was high ($186,000) because the work was extremely labor intensive. However, the improvements (including replacement of stoppings and
installation of belt regulators) have resulted in a more efficient ventilation system capable of supporting the additional miner sections required for Mine #2 development.

Provided that ground conditions allow for the use of 3C support, longwall ventilation can be improved by switching to this support instead of timber cribbing. The savings are substantial, in terms of both material costs and man-hour requirements ($176,340 for Mine #1 tailgate).