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Edited by Joseph Hirschi

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Preface

When I, Joseph Hirschi, began my first postcollege job working at an underground coal mine, the newly hired, inexperienced miner, 40 h training I was required to complete included a tour of the mine complex. One of the stops was where the mainline conveyor belt transferred to the slope belt. The entire mine production passed this point on its way to the surface. This area was kept in immaculate condition with fresh rock dust on the roof, floor, and ribs; there were no spills at transfer points and no float dust underneath belt rollers or on the framework. This was mostly the work of one man, Jack Webb. His primary responsibility was to keep this area clean. Posted on the guarding around the drive motors were hand-painted signs with quotes attributed to Mr. Webb. One read, "A Clean Mine is a Safe Mine." Another read, "A Safe Mine is a Productive Mine." When I first read them, they merely seemed like nice clichés, yet over the course of my career, I have observed and learned the profound truths that they proclaimed. This book expresses that learning in engineering or scientific terms thanks to the contributions of the expert authors, who are specialists in the areas of coal mine operations including productivity, safety, and environmental stewardship. I'm proud to consider them my colleagues and peers and to call them my friends.

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Profitable coal mining means being productive, safe, and environmentally responsible

1

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1.1 Foundations of profitability

For those who keep their finger on the pulse of the coal-mining industry, a few names become very familiar. It doesn't seem to matter whether one is reading a journal article spotlighting the most productive coal mines in the industry, browsing a web page recognizing the most recent Sentinels of Safety award recipients, or reading an email about coal mine operations receiving Excellence in Mine Reclamation awards. The same coal mines and coal-mining companies almost always rise to the top.

Coal mining is a business, and staying in business requires making a profit. The National Mining Association (NMA) is the self-proclaimed voice of the United States (US) mining industry in general and the coal-mining industry in particular. Their mission is to "build support for public policies that will help Americans fully and responsibly benefit from [its] abundant domestic coal and mineral resources" [1]. This is accomplished through being fully engaged in public (governmental) and private (business) processes that impact the mining industry's ability to safely and sustainably extract and process mineral resources in a profitable manner.

The NMA is one of the leading proponents of the CORESafety program, a business partnership among its member organizations, primarily mining companies. This program takes the approach that focusing on mine safety "is the right thing to do," but it is also good business [2]. CORESafety is focused on preventing accidents and eliminating fatalities by establishing the 0:50:5 target for individual mines, entire companies, and the industry as a whole. That standard is to have "0 fatalities and a 50 % reduction in injury rates within 5 years" [3]. They cite US Mine Safety and Health Administration (MSHA) data in postulating that "safer mines are generally more productive" [2].

By its very nature, coal mining impacts the environment in ways that disturb its natural water cycle, decomposition, and erosion processes. While not as easy to statistically measure as are safety standards such as number of fatalities and injury rates, environmental impacts are arguably more visible. Therefore, it is important for the industry to show that these impacts have a productive purpose and will be mitigated in the long term. Regarding environmental sustainability, the NMA assists its

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membership in maintaining a strong commitment to "continuing the extraordinary progress that [has been] made in recent decades, with technological advancements that minimize mining's impact on the environment" [4].

Relationships between productivity, safety, and environmental responsibility are difficult to quantify; however, one does not need mathematical proof when common sense and gut feeling confirm that definite correlations exist. Even the untrained (with respect to mining) mainstream media can pick up on it, as illustrated by the following excerpts from a 2010 New York Times article [5] (company names have been removed at the authors' discretion):

> *Coal mining carries inherent risks, but [a history of] numerous and very public violations and fatalities at [one] mine may leave the impression that all mines are run this way… They [are] not. A comparison [of] safety practices [at the mine with the long history of violations and fatalities] and those of other operators in the coal industry shows sharp differences… And the attention to safety — or the lack of it — has … measurable results: Compared with the industry average, workers [at mines with fewer violations] spent much less time away from work because of injury [i.e., are more productive]; workers [at mines with more violations] spent significantly more [time away from work; i.e., are less productive].*

A coal mining company's profitability is in large measure determined by what its customers are willing to pay for the product it mines. Customers are generally electricity generators, many of which are either publicly traded companies or municipal utilities. These organizations are under close scrutiny by investors, regulators, communities in which they operate, and the public in general. Furthermore, mining companies are in constant competition with other mining companies and other industries for potential employees. All of these entities look at the coal mining company's safety and environmental records as indicators of the stewardship it maintains for its employees, the responsibility it takes for the environment, and the relationship it has with the surrounding community. Consequently, effective health, safety, and environmental programs and procedures can improve mine productivity and thereby business profitability [2].

1.2 Statistical comparison of safety versus productivity

In the United States, MSHA regulations require all coal mines to regularly report production; employment; and accidents, injuries, and illnesses [6]. This information is readily available on MSHA's website [7]. The authors initially attempted to validate the hypothesis "the safest mines are the most productive mines" by analyzing safety and productivity data over a period of years; however, the volume of data is so massive that statistical interpretation is akin to an amateur stargazer identifying constellations in the night sky.

Due to time constraints for publishing this book, rather than continuing to work on digesting a massive amount of data, it was decided to limit our analysis to data from one calendar year for the top 25 producing coal mines in the United States. For each mine, data for underground operations, surface operations, nonpreparation plant surface facilities, and preparation plant facilities were considered separately for a total of 321 sites. A simple linear regression analysis was performed on these data using the Python scikit-learn library [8]. It had an R^2 value of 0.024 and did not indicate the trend we were hoping for (see Fig. 1.1).

After cleaning the data to show only total injury rates above zero, that is, null values were dropped, it was scaled and grouped into four clusters using the K-means algorithm. This algorithm works by separating samples into groups of equal variances through minimization of inertia and within-cluster sum-of-squares criteria. It is a three-step process commonly used with large data sets. In the first step, initial cluster centroids are chosen, and each data sample is assigned to one of the centroids. Then, the K-means algorithm creates new centroids based on the mean value of all samples assigned to each centroid in the first step. Finally, the algorithm processes differences between old and new centroids and repeats the first two steps until centroids no longer move significantly [9, 10]. Clusters are described in Table 1.1 and shown in Fig. 1.2. Manual clustering could have improved the statistical accuracy of the data (i.e., increased R^2 values), but this was not performed as it could be considered biased.

After clustering, ordinary least squares (OLS) linear regression analyses were performed separately for Cluster 1 and Clusters 2–4 combined as shown in Figs. 1.3 and 1.4, respectively. Happily, correlation trends were now in the desired direction; however, R^2 values improved to only 0.066 and 0.275, respectively. The model for Clusters 2–4 indicated that an increase in productivity of approximately 1 ton per

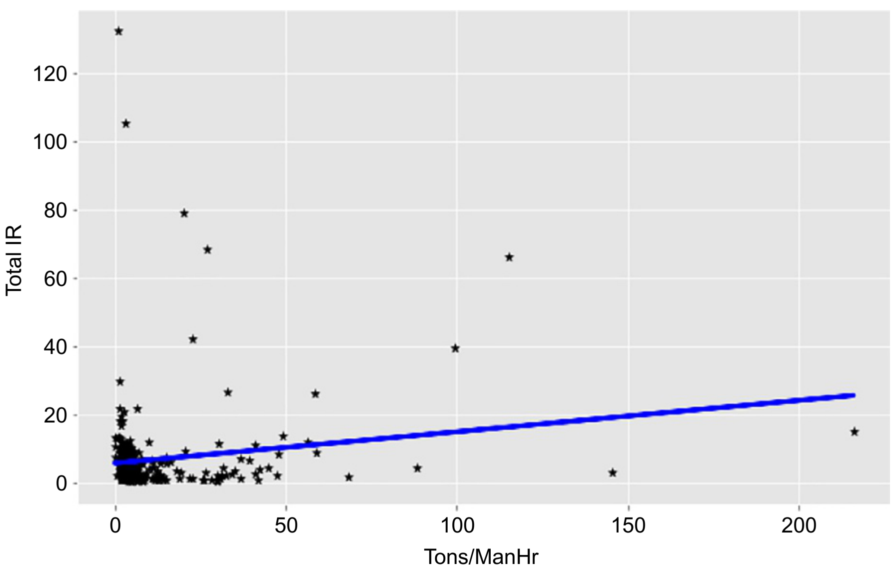

Fig. 1.1 Simple linear regression analysis of total injury frequency rate versus tons per manhour for 321 sites, $R^2 = 0.024$.

Table 1.1 ***K*-means cluster descriptions**

Cluster	Sites	Mean IR	IR std dev	Mean TPMH	TPMH std dev
1—purple	279	4.924	4.105	4.874	3.896
2—blue	4	30.881	28.060	144.108	51.593
3—green	4	96.368	28.532	12.759	12.760
4—orange	34	6.462	9.038	39.401	13.865

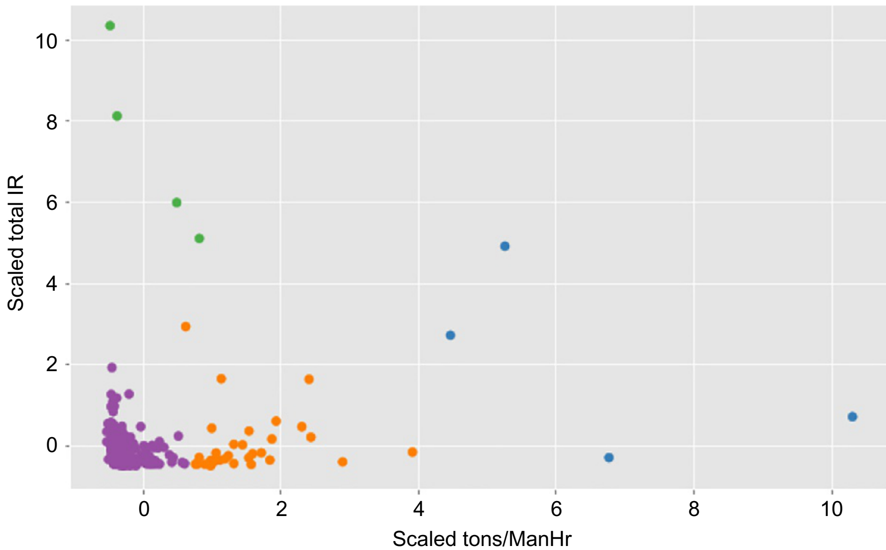

Fig. 1.2 *K*-means clustering of scaled data.

man-hour is correlated with a reduction in the total incident rate of 1 with a *y*-intercept (total injury rate) of about 50 for 0 ton per man-hour. According to this model, safer mines are more productive; however, it accounts for only 27.5% of variability observed in 11.8% of data suggesting a disturbing lack of transparency for the majority of sites.

In an attempt to address the question that remained as to how total incident rate and tons per man-hour are correlated for the majority of sites, a nonlinear (logarithmic) regression analysis was performed on Cluster 1 (see Fig. 1.5) and Clusters 2–4 combined (see Fig. 1.6). The only analysis with a reasonable R^2 value was the logarithmic analysis for Clusters 2–4 with an R^2 value of 0.665. This model suggests that the correlation between increased productivity and decreased safety incidents is asymptotic with total IR $= -32.05 \times \ln(\text{TPMH}) + 126.06$.

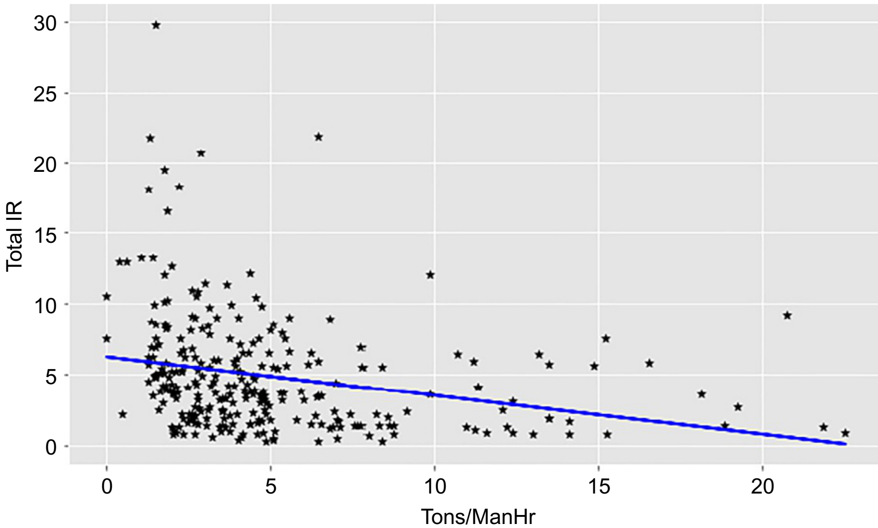

Fig. 1.3 OLS linear regression analysis of Cluster 1, $R^2 = 0.066$.

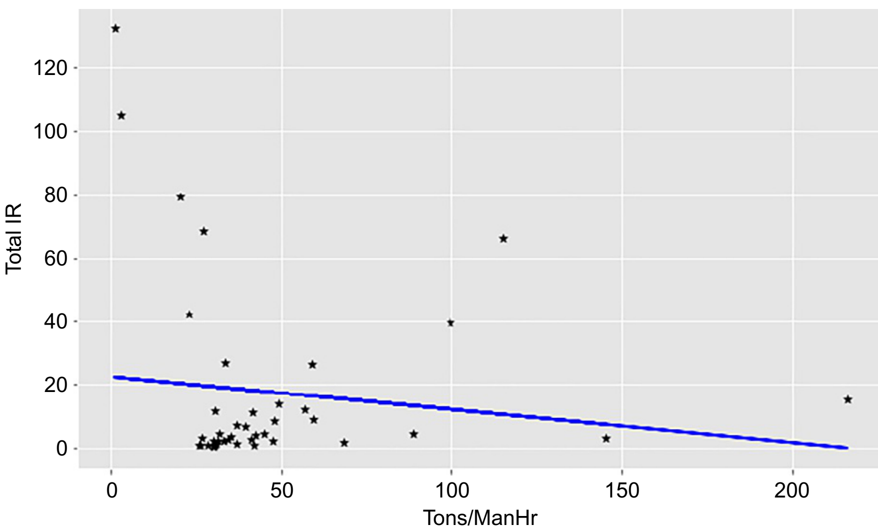

Fig. 1.4 OLS linear regression analysis of Clusters 2–4, $R^2 = 0.275$.

1.3 Importance of environmental responsibility

Statistically evaluating environmental success in coal mining is not as easy as comparing safety and productivity. There is plenty of research supporting successful mine reclamation, but the targets of minimizing environmental impacts during mining and

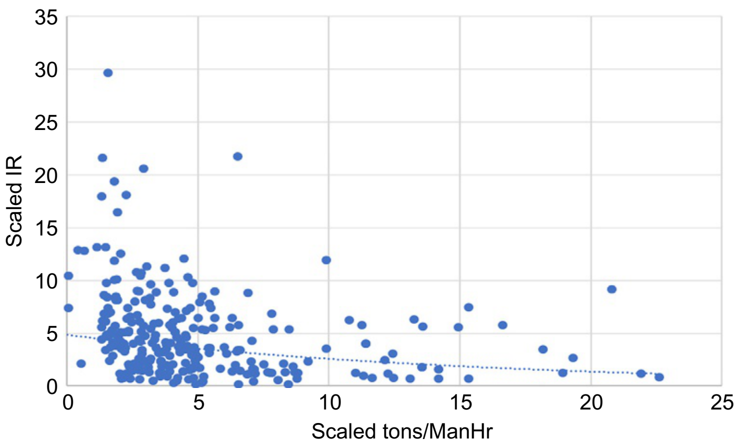

Fig. 1.5 Logarithmic regression analysis of Cluster 1, $R^2 = 0.0846$.

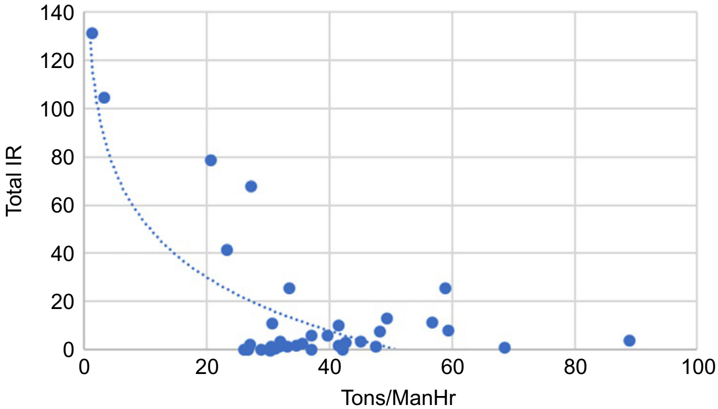

Fig. 1.6 Logarithmic regression analysis of Clusters 2–4, $R^2 = 0.6646$.

returning the land to as good or better use after mining do not have standardized formulas for calculating them like injury frequency rates do. Clearly, reducing safety incidents has a positive impact on mine productivity and profitability. On the other hand, reducing environmental impacts appears to only have a negative impact on the bottom line.

As previously stated, extracting minerals from the earth, by its very nature, is a destructive process; however, it is possible to minimize damages by first systematically evaluating local and industry-wide environmental impacts; then developing methods to counter as many of those impacts as possible; and, finally, implementing methods that prove successful. Regulations have established standardized methods for both safety and environmental performance, but legislation concerning the efficiency of mining has yet to take hold. Obviously, safety and environmental regulations differ from country to country, but generally speaking, they are universal throughout mining, as would be expected for an industry that is unavoidably hazardous and inevitably causes some environmental damage.

The mining industry has learned to tolerate environmental regulations; however, for environmental responsibility to become mainstream, these government mandates need to incorporate economic efficiencies that promote profitability. Mining companies are always pushing for incremental efficiency gains, but those will not do the job by themselves. There has to be a huge push toward improving the efficiency of both the mining process and environmental performance as well. Regulations that promote mining efficiency and environmental stewardship will not only protect the environment but also improve the life expectancy of individual mines and the industry as a whole.

Even without regulations promoting mining efficiency, mining companies need to be proactive in pursuing "green" methods of extraction and reclamation. Unconventional (at least from a mining standpoint) practices like recycling have provided significant benefits in terms of their positive effect on both mine economics and environmental impacts for many metal-nonmetal operations [11]. Unlike metals, coal is consumed after mining and processing; however, research needs to explore if there are ways to recycle some of the energy and materials that go into those processes.

The main objective in the reclamation process is to return land that has been disturbed by mining and surrounding areas back to usable conditions. This requires ensuring that any landforms and structures are stable, that watercourses have regained quality levels that meet their intended purposes, and that flora and fauna are growing at healthy rates. Proven methods for removing or properly disposing all waste material, replenishing native soils and vegetation, and establishing ecosystems that are sustainable for years beyond when a mine is no longer operating already exist; however, further research is needed to advance these methods beyond what is currently acceptable. This is because society's commitment to sustainability is ever changing, and reclamation efforts must be constantly pushing the envelope of what is doable. By thinking ahead, mining operations can improve their own efficiencies and do so in a climate where they are better understood and accepted, not only by locally surrounding communities but also by society at large.

Thus, on the environmental side of mining, the focus has to be on integrating proven environmentally sound techniques and methods into the actual mining process itself. Aligning the economic side of mining (i.e., productivity and resource recovery) with the environmental side of mining (i.e., protection and reclamation) is the only sustainable path forward.

1.4 Achieving profitability

Coal's staying power at the forefront of the world's energy mix for more than a century is the result of low-cost price stability; however, that position is under attack from several sides. Critical issues affecting coal mining include maintaining low-cost productivity, addressing health and safety hazards, and being responsible stewards of the environment.

At first glance, each of these critical issues may appear to be an independent or mutually exclusive objective; however, when they are dealt with as such, profitability comes into question.

For coal mining to be successful as a business, profitability is essential. A mine is profitable when revenues exceed costs. Health and safety issues and environmental stewardship affect the cost component of profitability. Focusing on them without sufficient attention to productivity may result in achieving lower costs, but with insufficient production to generate enough revenue to cover those costs. The key parameter affecting both revenue and cost is productivity. Productivity is measured as the rate of output produced per unit of required input. For coal mines, output is generally expressed in tons. Required input includes many components, the largest of which is labor. Thus, input is generally expressed in man-hours. Monetary values are associated with both output and input to derive profitability.

Coal reserves remain abundant, but those with the most favorable mining conditions continue to be depleted leaving mine operators to deal with difficult productivity challenges. Coupled with that, coal mining is still a dangerous occupation despite decades of safety advancements. Although less frequent, fatalities and disasters continue to occur with each one resulting in substantial costs and causing irreparable damage to the image of the entire industry. Coal mining's environmental footprint continues to be reduced as a result of two generations of trained professionals applying the latest technology and scientific information in cooperation with regulatory authorities, but to environmental activists, it is not enough.

In facing these challenges, the tendency is to become narrowly focused on the most pressing issue: to become a specialist in safety, productivity, or the environment. This specialization can lead to the dilemma of "not seeing the forest for the trees." To aid in resisting that tendency, this book takes the approach that the most meaningful advances in profitable coal mining address at least two of the three critical issues that have been identified. The book is divided into three sections. The first focuses on advances that improve productivity and safety; the second focuses on advances that improve safety and environmental responsibility; and the third focuses on advances in environmental responsibility and productivity. It is hoped that this approach will provide the reader with a more holistic perspective on advancements that can adopted in an effort to achieve greater profitability. Further, it is hoped that this approach will serve as a useful guide to those seeking to deliver those advances that will come in the years after this book is published.

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Safety and productivity in coal mining—How to make both the top priority

2

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2.1 Introduction

Throughout the recent history of coal mining in the United States (US), advances and changes were made in mining methods, the nature of hazards in mines, markets for coal, technology development and implementation, the geographical extent of coal mining, availability of workers, unionization of workers, federal and state safety agencies and regulations as well as the extent of enforcement, safety practices, and the relationship between management and labor. The evolution of these factors on the pursuit of high-level performance in safety and productivity will be discussed. Certainly, the discussion will reveal the stepwise progress made separately and eventually in tandem on key performance metrics. The joint systemization of operators, enforcement agencies, unions, and miners, collectively, has successively improved their performance toward the ultimate goals of minimizing loss of life and property and maximizing quality of life and productivity, which will be revealed throughout the chapter.

2.2 Coal mine safety and productivity: 1901–2015

In the following subsections, the progress in mine safety and productivity over the period from 1901 to 2015 will be presented. Each subsection covers statistics related to mine safety and production/productivity data, as available. In the earliest years of the overall period, data were only collected on a limited basis, particularly with respect to safety metrics, and hours worked by miners during those years are not publicly available at this time.

2.2.1 Fatalities and estimated fatal incidence rate (IR): 1901–30

No national coal production and safety data can be found for this period, with the exception of the number of miners employed and the number of fatalities that occurred each year [1]. The number of hours actually worked by miners is also missing. Thus, the safety performance measure for this period is the estimated annual fatality rate, as equivalent to the estimated fatality incidence rate (Fatal IR_{equiv}). In the US, the Fatal IR is defined as the number of fatalities occurring in a year divided by the total number

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of annual hours that miners worked, normalized by 200,000, which represents 100 miners working 2000 h in a year. Since work hours were not available for the period, the annual equivalent fatality incidence rate was calculated as the number of fatalities divided by the number of miners times 2000 h, normalized by 200,000.

In 1901, the total number of miners working solely in underground coal mines was 485,544. Deaths in those mines totaled 1574, giving a Fatal IR_{equiv} of 0.324, which is the starting point for comparison purposes. The number of working coal miners increased steadily from 1901 through 1923, when it reached 862,536. In 1923, 2462 miners died in coal mines resulting in a Fatal IR_{equiv} of 0.285. The decrease was not steady during the period, but fluctuated dramatically with a high of 0.476 in 1907, when a record high of 3242 miners died, to a low of 0.235 in 1922. December 1907 was labeled "Bloody December" because of 17 disasters, including the worst one historically in Monongah, West Virginia, where 367 miners lost their lives [2]. Virtually all coal mine fatalities during this period occurred in underground mines; surface mining having begun in 1915, when it accounted for only 0.6% of total US coal production. By 1920, that percentage had only climbed to 1.5%.

The number of miners working in coal mines decreased steadily from 1923 to 1930, when 644,006 miners were employed, again predominately in underground mines. The number of fatalities decreased steadily as well, to 2063 in 1930, with one exception. In 1925, 2518 miners perished. This decrease represents a positive indicator of improvement for the time period.

2.2.2 Safety and productivity performance: 1931–77

Production data are archived and available via the US Mine Safety and Health Administration (MSHA) website [3] for coal mines beginning in 1931, but hours worked by miners are not. For this period, productivity can thus be measured as tons mined per miner on an annual basis (i.e., tons/miner/year). Accordingly, annual productivity and Fatal IR_{equiv} data are presented for end-of-decade years (i.e., 1940, 1950, 1960, and 1970), which will reveal any progress, or lack thereof, made during this entire period [3,4].

Table 2.1 reveals a dramatic decrease in the number of coal miners over the time period. Also dramatic is the increase in the percent of coal mined by surface mining methods, rising from 9.2% in 1940 to 43.8% in 1970. In spite of the decreasing number of miners, production increased by 18.9% over the period. Primarily because of the large increase in surface-mined coal, productivity dramatically increased over the period. Unfortunately, safety performance (i.e., Fatal IR_{equiv}) virtually stagnated from 1960 to 1970 after halving between 1940 and 1950. The number of fatalities in the coal industry fell below 1000 for the first time in 1946. From detailed data not presented here, 1940–44 saw a steady increase in productivity from 831 to 1509 tons/miner/year, but 1945–49 realized a steady decrease. A sustained annual increase of productivity occurred from 1950 at 1164 tons/miner/year through 1969, peaking at 4270 tons/miner/year.

As productivity sustained improvement over the period, unfortunately mine disasters (explosions and fires predominately) continued to occur catching the attention of

Table 2.1 **Summary of safety and productivity data: 1940–70**

Year	Number of miners	Number of fatalities	Equivalent fatal incidence rate (Fatal IR_{equiv})	US production (million tons)	Productivity (tons/miner/year)	Percentage of surface mining
1940	533,267	1388	0.260	513	962	9.2
1950	483,239	643	0.133	562	1164	23.9
1960	189,679	325	0.171	435	2294	29.5
1970	144,480	260	0.180	610	4221	43.8

Table 2.2 Delineation of safety and productivity data: 1971–77

Year	Number of miners	Number of fatalities	Equivalent fatal incidence rate (Fatal IR_{equiv})	US production (million tons)	Productivity (tons/miner/year)
1971	533,267	181	0.127	563	3960
1972	483,239	156	0.096	535	3296
1973	189,679	132	0.087	558	3673
1974	144,480	133	0.073	566	3104
1975	224,412	155	0.069	611	2723
1976	221,255	141	0.064	647	2925
1977	237,506	139	0.059	671	2824

the public and the US Congress. The stagnation of improvement in the fatality rate coupled with the major (78 deaths) explosion-type disaster in 1968 at Farmington, West Virginia, led to passage of the 1969 Federal Coal Mine Health and Safety Act [5], which will be discussed later.

Table 2.2 gives detailed fatality and productivity data from 1971 through 1977, when the 1977 Federal Mine Safety and Health Act was passed by the US Congress [5]. Employment in coal mines more than halved over the period, and the number of fatalities decreased steadily as the percentage of surface-mined coal increased toward the 1980 level of 59.3%. Productivity decreased significantly as well. In a nutshell, progress in safety increased while productivity fell greatly. Certainly increased surface mining had an enormous impact on safety performance, and as underground mining diminished dramatically, the number of fatalities did, too. The increase in surface mining would likely have improved productivity, if it were the only factor influencing productivity; however, the coal industry had to adjust to the "1969 Act," which required many additional nonproduction miners to comply with a comprehensive set of new regulations, and this resulted in the decline in productivity.

2.2.3 Safety and productivity performance: 1978–2015

Table 2.3 shows progress in more recent years using additional data and measures from another MSHA source [6] while keeping relative comparisons intact. Remarkable are the sharp decreases in the number of miners employed and the number of fatalities (e.g., 1983 marked the first year in which fatalities fell below 100). Following this decline in the Fatal IR_{equiv} from 1980 to 1990, significant progress stalled until breakthroughs occurred in the second decade of the 21st century. The year 2010 marked the last major disaster, an explosion at the Upper Big Branch mine in southern West Virginia that resulted in 29 fatalities. It is noted that the 2006 Mine Improvement and New Emergency Response (MINER) Act [7] was passed by the US Congress

Table 2.3 **Summary of safety and productivity data: 1980–2015**

Year	Number of miners	Number of fatalities	Equivalent fatal incidence rate (Fatal IR_{equiv})	US production (million tons)	Productivity (tons/miner/year)
1980	253,007	133	0.053	798	3154
1990	168,625	66	0.039	1019	6045
2000	108,098	38	0.035	1079	9982
2010	135,500	48	0.035	1086	8015
2015	102,804	11	0.011	896	8716

because of multiple explosions and a mine fire during the 2001–06 period. This will be discussed further in Section 2.3, along with those safety issues it addressed.

Following a 9-year general decline in productivity, primarily the result of adjusting to the 1969 and 1977 Acts, a 26-year trend of increasing productivity occurred from 1979 to 2004. During this period, productivity increased from 2848 to 10,231 tons/miner/year, as it continued to track with the increased percentage of surface-mined coal, which went from 59.3% in 1980 to 68.9% in 2010. During the period from 2005 to 2012, however, productivity dropped steadily from 9748 to 7395. This occurred largely from multiple influences, including adjustment to the provisions of the MINER Act and a change in workforce demographics (i.e., large numbers of retiring, experienced miners were replaced with a new generation of inexperienced miners).

Excluding the 29 coal miners who perished in the Upper Big Branch disaster of 2010 [5], there were 19 other fatalities that year, which halved the number that occurred in 2000 and would have halved the Fatal IR_{equiv} as well. Thus, 2010s safety performance shown in Table 2.3 belies the progress made during the period 2001–10, which was continued onward through 2015. MSHA initiatives, many of them nonlegislative regulatory efforts, placed considerable pressure on mine operators helping to drive these improvements in safety performance measures. These initiatives will be discussed in detail in Section 2.3.

2.2.4 Safety and productivity performance by mining method: 1980–2015

There are major differences in fatality and injury rates as well as in productivity for underground and surface coal mines. Underground mine hazards are more prevalent and have magnified risk relative to surface mine hazards. Thus, a proper assessment of safety performance progress in the coal industry requires separate examination of underground and surface mines in order to assess the progress of each sector. Data are available online [6,8].

Table 2.4 summarizes relevant safety and productivity performance for underground coal mines during the period from 1980 to 2015, as broken down by previously

Table 2.4 Summary of safety and productivity data for underground coal mines: 1980–2015

Year	Fatal IR	NFDL IR	Production (tons)	Employee hours	Tons per employee hour
1980	0.08	12.13	321,018,628	248,546,274	1.29
1990	0.06	12.15	419,899,597	153,347,464	2.74
2000	0.05	8.34	373,318,434	86,160,480	4.33
2010	0.08	3.78	337,348,524	100,975,207	3.34
2015	0.02	3.23	306,338,066	77,062,576	3.98

used decade marker years. Of note, first of all, is the dramatic decrease in employee hours worked, which represents the exposure to potential hazards by workers. This significant change is reflective of the continued percent increase in surface-mined coal and the decrease in the demand for coal nationally (largely from 2012 to 2015).

As seen in Table 2.4, the Fatal IR fluctuated between 1980 and 2010 without significant improvement; the same can be said regarding year-to-year performance (not shown). On the other hand, the NonFatal occurrences with Days Lost (NFDL) IR showed steady and significant progress from 1990 to 2015. During this period, one or two fatalities in a year highly inflated the Fatal IR, but the same was not true with the impact of a few lost-time injuries on the NFDL IR.

Of further note in Table 2.4 is the productivity increase from 1980 to 2000 in underground coal mines; however, that progress was interrupted by several disasters and other multiple-fatality events during 2001–10, as described previously. Further, the ramping up of federal legislation (MINER Act), regulations, and other initiatives, to be described in Section 2.3, helped generate subsequent progress in safety performance while simultaneously impacting productivity negatively. One other important factor was the transition toward employment of younger, inexperienced miners who replaced retiring veterans. In the end, performance achieved on both fronts (safety and productivity) in 2015 shows the net, positive effect of all influences.

Table 2.5 reveals the same safety and productivity performance for surface mines over the period from 1980 to 2015. Patterns of change for employee hours worked,

Table 2.5 Summary of safety and productivity data for surface coal mines: 1980–2015

Year	Fatal IR	NFDL IR	Production (tons)	Employee hours	Tons per employee hour
1980	0.03	3.48	477,063,106	136,112,132	3.50
1990	0.02	3.57	599,443,181	104,682,281	5.73
2000	0.03	2.23	700,497,168	71,886,169	9.74
2010	0.01	1.19	748,703,196	70,322,826	10.65
2015	0.004	0.96	589,702,597	47,573,141	12.40

Fatal IR, and NFDL IR are similar; however, productivity steadily and significantly increased in marker years. It is noted that magnitudes of both the Fatal IR and the NFDL IR were much lower than for underground mines, reflective of the increased high-risk hazards in underground coal mines.

2.2.5 Sustainability study of safety performance of surface and underground mines: 2008–13

Reflective of the need for pursuit of sustainability-related improvements in safety and environmental performances in the US, a major electric power company (American Electric Power) began to examine the performance of its coal suppliers (mines). Because comprehensive data were available on coal mine safety, the company sought comparisons of the suppliers' performances relative to national averages in key safety metrics. Through use of a consultant, national government coal mine safety data were gathered, analyzed, compared, and summarized in an annual report [9] titled "Sustainability Survey of AEP Coal Suppliers." Detailed examples from the study are presented next revealing the importance of safety performance improvements in the public's eye. One additional metric was introduced, the severity measure (SM), which is calculated as the total days lost and restricted work days for an injured miner plus statutory days charged for various levels of disability and death. This important metric, later adopted by the federal enforcement agency (MSHA) in its procedure to identify poor safety performers, will be described in Section 2.3. Results of the survey are presented for large surface mines and large underground mines over the 6-year period from 2008 to 2013. Results in the survey were presented by mine-size category (large, medium, small, and very small) because of the significant differences in the level of resources available, quality of the equipment, size of the workforce, physical size of the mine complex, and other factors like work and safety culture. A summary of overall results was published [10].

2.2.5.1 Fatality and injury metrics

Table 2.6 summarizes the safety performance of large surface mines (employing 100 or more miners) during the entire survey period. Comparisons of metrics are made to the national performance for each metric. Table 2.6 reveals that AEP's large surface mine coal suppliers had a Fatal IR at or below the national average for this mine-size category in 5 of 6 years. Once the number of fatalities for a subsector reaches a very low level, one or two fatalities make a significant difference in the Fatal IR, which was the case in 2013 for the large-surface coal mine category.

On the other hand, when a much larger number of injuries occur, as reflected by the NFDL IR, the metric is less sensitive to a low increase in count. Table 2.6 reveals that, during the period being considered, the suppliers' NFDL IR was better than the national average in all 6 years, reflecting a consistent superior performance.

The Severity Measure reflects the intensity of combined injuries and disabilities, and it can fluctuate significantly from year to year, as revealed in Table 2.6. If several full disabilities occur in a year (6000-day statutory charge each), then the Severity Measure will rise dramatically. During this period, the suppliers' Severity Measure

Table 2.6 Suppliers' accident measures vs. national rates for large surface mines

Year	Fatal IR	% of difference	NFDL IR	% of difference	SM	% of difference
2008	0.00	−100	0.70	−23.9	70.1	−56.8
2009	0.01	0.0	0.63	−27.6	228.0	28.5
2010	0.00	0.0	0.49	−30.0	46.4	−23.7
2011	0.03	0.0	1.03	−14.2	326.1	1.3
2012	0.00	−100	0.86	−6.5	138.4	−4.2
2013	0.02	45.5	0.80	−21.6	242.7	4.0

was better than the national average in 3 of 6 years, very near the national average in two other years, and exceeded the national average by 28.5% in the remaining year (2009). Overall, however, the suppliers' Severity Measure was consistently near or below the national average.

Table 2.7 gives similar results for large underground coal mines. For the Fatal IR, the supplier mines' performances were better than the national average in 4 of 6 years, and in 2011 two fatalities were sufficient to cause the 69.2% increase over the national average. In 2008, a single fatality caused the 23.5% increase.

The NFDL IR and Severity Measure metrics were superior to the national averages in 5 and 3 years, respectively, of the 6-year period. The NFDL IR was only slightly elevated in 2013. The severity measure, reflective of fatalities and likely some significant disabilities in 2011, was 33.3% higher than the national average. In 2008, 2012, and 2013, the suppliers' severity measure was near the national average.

2.2.5.2 Enforcement agency citation metrics

MSHA, the national mine safety enforcement agency in the United States, polices all mines for compliance with mine safety regulations. When noncompliance with provisions is found during an inspection, a citation is written giving details on the

Table 2.7 Suppliers' accident measures vs. national rates for large underground mines

Year	Fatal IR	% of difference	NFDL IR	% of difference	SM	% of difference
2008	0.03	23.5	3.44	−24.0	445.1	6.1
2009	0.00	−100	3.55	−15.2	334.2	−26.1
2010	0.01	−55.9	2.93	−17.5	305.3	−32.8
2011	0.01	69.2	2.79	−11.6	445.9	33.3
2012	0.01	−4.0	3.03	−11.8	346.0	−4.6
2013	0.03	−2.9	3.82	4.4	605.2	4.4

provision violated, the potential impact of it on workers, and the negligence level of the operator in not complying. Each provision violated is given a separate citation. In this chapter, a normalized measure (C/100IH), representing the number of citations per 100 inspector hours, is used to track the level of noncompliance. Once the negligence and impact aspects of a citation are judged by the inspector, and supported by the administration, as elevated (by a codified, legal definition), the citation is designated as Significant and Substantial (S&S). S&S fundamentally means the noncompliance was serious (some level of negligence) and could have injured or killed one or more miners working in or near the location where it occurred. In some situations, where the negligence and impact on workers was judged extreme and too dangerous for miners to continue to work in the affected area, the citation is designated as a withdrawal order, or unwarranted failure to comply with the specific provision. These latter two normalized metrics are indicated as S&S/100IH and O/100IH in the tables later. The total number of citations, including nonelevated ones, is not given. The raw data are available online [11].

Table 2.8 presents the elevated citation metrics for large surface mines. The coal suppliers' performance for S&S/100IH was better than the national average for this mine-size category in 4 of 6 years, while the metric for 2008 was near the national average and the metric for 2011 was reasonably close to the national average. For O/100IH, the suppliers' performance was better than the national average in 5 of 6 years, and close to the national average in 2009. Overall, those suppliers evaluated in the aforementioned AEP study performed very well on compliance with coal mine regulations.

For underground coal mines, Table 2.9 indicates that AEP coal suppliers' S&S/100IH performance was better than the national average in 5 of 6 years, and very near the national average in 2009. Further, the performance steadily improved during the period 2009–20, which is remarkable. For O/100IH, results were even better as suppliers' performance was superior to the national average in all 6 years while demonstrating a steady decline from 2009 through 2013, with the exception of 2012.

Together, Tables 2.8 and 2.9 show that concerted effort of coal suppliers to comply with regulations, and the superior citation results, even in a major-disaster year (2010), were significant. The added influence of pressure to maintain market share (desire to provide coal to a major power company embracing sustainability principles), coupled

Table 2.8 Suppliers' citation measures vs. national rates for large surface mines

Year	S&S/100IH	% of difference	O/100IH	% of difference
2008	4.30	3.6	0.22	−38.9
2009	3.28	−2.9	0.20	8.2
2010	2.87	−13.8	0.19	−19.7
2011	4.45	13.8	0.23	−28.8
2012	3.69	−1.6	0.22	−35.1
2013	3.47	−26.3	0.20	−13.0

Table 2.9 Suppliers' citation measures vs. national rates for large underground mines

Year	S&S/100IH	% of difference	O/100IH	% of difference
2008	4.08	−23.8	0.21	−30.6
2009	4.67	0.4	0.27	−1.4
2010	4.15	−9.7	0.15	−56.0
2011	3.30	−15.0	0.12	−56.1
2012	3.23	−10.5	0.14	−28.3
2013	2.64	−15.1	0.11	−35.3

with enforcement pressure, no doubt redoubled the suppliers' effort toward achieving superior citation performance. The suppliers also likely sought to shape a significant public impression of their efforts.

2.2.5.3 Combining injury and citation metrics into an overall safe performance index

A combined safety metric, the Safe Performance Index (SPI), was included in the AEP sustainability survey reports. The SPI methodology [12] combines three injury metrics and three citation metrics using weighting factors reflective of the seriousness of each metric. Fifty percent of the weighting is applied to injury metrics and 50% to citation metrics. Weighting factors within the injury performance category are 5% for no-days-lost accident incidence rate (NDL IR), 15% for NFDL IR, and 30% for SM. Weighting factors within the citation performance category are 5% for C/100IH, 15% for S&S/100IH, and 30% for O/100IH. In the SPI methodology, the average SPI nationally is always 66.7 for all mine-size categories. This SPI methodology was supported in US Congressional hearings on coal mine safety legislation [13], publicly supported by coal-mining organizations, and a separate bill on coal mine legislation explicitly adopted it; however, no final action was taken on new legislation. Rather, MSHA decided to adopt the use of the SM in a new enforcement action targeting "bad" operator safety performance and known as pattern of violations (POV) [14], which will be discussed later. Results of AEP suppliers' SPI performances over the 6-year period from 2008 to 2013 are shown separately for surface and underground mines.

Table 2.10 gives suppliers' average SPI performance for surface coal mines by mine-size category. Suppliers' SPI was higher (better) than the national average of 66.7 in all years for large, medium, and very small mines, and it was lower (worse) than the national average for small mines in 2012 only. No very small surface mines supplied coal to AEP in 2009 and 2012. Table 2.11 reveals that large, medium, and very small underground mine suppliers' SPI exceeded the national average of 66.7. Small underground mine suppliers' SPIs were below the national average in 2010 and 2011, but not dramatically.

Table 2.10 Suppliers' average safe performance index for surface mines

Mine size	2008	2009	2010	2011	2012	2013
Large	75.1	69.0	71.1	73.5	72.2	71.6
Medium	91.4	82.3	79.2	70.1	82.5	86.8
Small	98.6	79.8	85.0	97.2	59.7	91.6
Very small	96.8	N/A	84.8	96.4	N/A	95.0

Table 2.11 Suppliers' average safe performance index for underground mines

Mine size	2008	2009	2010	2011	2012	2013
Large	74.2	73.5	78.2	71.7	70.9	72.9
Medium	81.4	73.7	76.2	83.2	92.8	N/A
Small	82.1	89.0	56.6	55.0	N/A	81.9
Very small	N/A	N/A	93.9	N/A	N/A	N/A

2.3 Evolution of the role of the US federal safety and health enforcement agency

MSHA's website provides a comprehensive review of all legislative action addressing coal mine safety and health in the United States [5]. This section highlights some important milestones.

Mine safety legislation in 1891 is believed to have been the first, but it was administered only in territories of the country. In 1910, significant legislation (The Organic Act) established the Bureau of Mines as a federal research institute to address primarily the great death toll from mine explosions and fires in underground coal mines. No mine inspection capability was given until 1941 when federal inspectors were given inspection authority; however, there was no code of regulations to govern these inspections until 1947, and enforcement authority was weak.

By 1952 annual inspections in underground coal mines were authorized, and the Bureau of Mines was given limited enforcement power, including writing citations and issuing imminent danger withdrawal orders. It also authorized civil penalty assessments on withdrawal orders and against operators who did not allow inspectors access to their mines. No authorization was given to issue citations for noncompliance with the code of safety regulations. Finally, in 1966, inspectors were authorized to enter all underground coal mines.

In 1969, a very comprehensive and tough piece of legislation was passed to improve coal mine safety and health. The 1969 Coal Mine Health and Safety Act included significant new provisions, such as:

- Embracing surface coal mines as well as underground coal mines.
- Requiring two complete inspections of surface coal mines each year.
- Requiring four complete inspections of underground coal mines each year.
- Increasing enforcement powers.
- Authorizing monetary penalties for all violations.
- Authorizing criminal penalties for knowing and/or willful violations.
- Establishing comprehensive safety standards for all coal mines.
- Adopting new health standards for all coal mines.
- Authorizing compensation for miners with total disability due to lung disease caused by respiratory dust.

Following another major disaster at the Scotia Mine in Kentucky in 1976 (26 fatalities in two explosions over 3 days), coupled with a parity of total fatalities per year between coal mines and noncoal mines, the Mine Safety and Health Act of 1977 was passed with the following major changes:

- Transferred the Mining Enforcement and Safety Administration (MESA) from the Department of Interior to the Department of Labor and renamed it the Mine Safety and Health Administration (MSHA).
- Promulgated comprehensive safety and health regulations for noncoal mines.
- Expanded the rights of miners.
- Provided better protection from retaliation for miners exercising their safety rights.
- Established the Mine Safety and Health Review Commission to allow independent review of MSHA enforcement actions, as necessary or on appeal.

As mentioned previously, the Mine Improvement and New Emergency Response Act was passed in 2006 following major fatality events in 2005, one of which resulted in 12 miners perishing while housed in a barricaded area in the Sago Mine in West Virginia. Aimed at providing regulations and new technology to enhance escapeability or safe housing of miners who cannot escape an underground coal mine following an explosion or fire, significant changes include the following:

- Required current emergency response plans, to be reviewed every 6 months by MSHA.
- Promoted use of equipment and technology commercially available in mines to enhance escape and/or survival.
- Required implementation of wireless 2-way communications and electronic tracking systems within 3 years.
- Required each mine to have two experienced rescue teams capable of responding within 1 h.
- Required operators to notify MSHA within 15 min of any accident that had reasonable risk of death.
- Raised the criminal penalty cap to $250,000 for a first offense and $500,000 for second offense, and established a maximum civil penalty of $220,000 for a flagrant violation.
- Gave MSHA authority to request an injunction to shut down a mine when the mine has refused to pay a final order penalty.
- Established other grant programs, a scholarship program, and an interagency work group to enhance mine safety.

In addition to enforcing new legislation and the resulting regulations, MSHA also pursues regulatory reform and uses special initiatives to enhance mine safety and health in the United States. The regulatory process requires full participation of stakeholders in providing input to a proposed regulation, and it can take a number of years to complete; however, most often it can be completed in 1 or 2 years. To give the nature of the types of issues that are addressed through rulemaking, new regulations pursued by MSHA over the past 5 years include the following [15]:

- Maintenance of incombustible content of rock dust in underground coal mines.
- Examinations of work areas in underground coal mines for violations of mandatory health and safety standards.
- Criteria and procedures for proposed assessment of civil penalties, including an inflation adjustment.
- Pattern of violations.
- Lowering miners' exposure to respirable coal mine dust, including continuous personal dust monitors.
- Proximity detection systems for continuous mining machines in underground coal mines.

MSHA also pursues special or strategic issues to improve mine safety and health performance through more scrutiny of mine operations that appear problematic based on certain criteria. Two examples include Impact Inspections and Rules to Live By.

MSHA initiated the Impact Inspections program in 2010 following the Upper Big Branch mine disaster (29 miners died). The agency scrutinizes the compliance records of coal and noncoal mines identifying those who demonstrate a poor compliance history or have particular compliance concerns. These mines are then subjected to an intense inspection involving multiple inspectors. From April 2010 to May 2016, MSHA had conducted 1156 impact inspections issuing 16,315 citations and 1313 orders [16]. On multiple occasions, the MSHA chief has noted that at mines having received impact inspections, with at least one follow-up inspection, the impact inspections "have made the miners safer" [17,18].

The Rules to Live By initiative was instituted by MSHA to target prevention of fatalities that happen one or two at a time, and to reinforce prevention of disasters. After investigating each of the 623 fatalities that occurred between 2000 and 2009 in all types of mines, MSHA focused on those safety standards that were most violated in these fatalities, their root causes as determined from citations and accident reports, and practices used to abate the conditions cited. Four subcategories were created [19] as follows, to guide emphases during inspections:

- Rules to Live By I (fatality prevention) focuses on 24 frequently cited standards (11 in coal mining) that cause or contribute to fatal accidents in nine accident categories.
- Rules to Live By II (preventing catastrophic accidents) focuses on standards cited during major disasters that contributed to five or more fatalities over the past 10 years.
- Rules to Live By III (preventing common mining deaths) focuses on 14 standards (8 in coal mining) cited as a result of at least five mining accidents and resulting in at least five deaths during the 10-year period from 2001 to 2010.
- Rules to Live By IV (preventing common mining deaths) focuses on 2 safety standards (1 in coal mining) cited as a result of at least five mining accidents resulting in at least five fatalities during the 10-year period from 2006 to 2015.

2.4 Evolution of safety practices at mines and plants since 1969

The coal industry faced an entirely new and significant challenge following passage of the 1969 Coal Mine Health and Safety Act. The challenge was shaped by many new, very specific safety and health provisions, including a comprehensive set of provisions related to mine equipment, mine systems, hazards, examinations, training, and work practices, to name a few. Underground coal mines, in particular, had the greatest challenge.

At the time of the 1969 Act, safety practices were many, but not driven hard by regulations or the possibility of penalties (i.e., elevated citations and fines). Many safety practices, particularly in mines owned and operated by larger corporations, were reasonably comprehensive and led by a safety director with assistants. The following list of safety practices prior to the 1969 Act are not all encompassing, but also are not reflective of the more extensive requirements following the 1969 Act:

- Initial training for new miners.
- Walk-through and/or drive-through orientation to the mine environment.
- On-the-job training for a newly assigned job.
- Training on hazards and their control in the mine.
- Training on first aid, ventilation principles, roof control principles, escape from an emergency, use of a CO-type self-rescuer, etc.
- Check-in, check-out system.
- Mining sequence plan.
- Transportation and mine hoist/cage signals.
- Safety department inspections of mines and reports with follow-up.
- Safety inspections of mines by a safety committee with reports for follow-up, especially if a union mine.
- In union mines, rights of miners.
- Accident reporting.
- Employment of a doctor on call for injuries.
- Often a nurse on day shift only.

Following the 1969 Act, most everything became prescriptive, and not following the regulatory provisions resulted in citations, after which abatement of the cited conditions had to be done quickly. Special legal books had to be kept on mine examinations, which were much more comprehensive and occurred multiple times on all work shifts. Examination records also had to be kept and made available for scrutiny on equipment, mine fans, sampling of dust levels in high-exposure areas, etc. Training was much more prescriptive with topics and time-requirements set. Task training, hazard training of visitors, and annual retraining became required.

New safety practices began to emerge to help maintain compliance with the comprehensive new regulatory provisions. Systematic planning and record keeping was necessary to ensure such things as equipment inspections, examinations of the mine on time schedules, general checks of mine systems, assurance of first-aid and emergency materials, etc. Tools such as Job Safety Analysis, plasticized cards for roof and ventilation control procedures, checklists for job procedures, formal on-the-job training handouts, etc. were all generated without the use of computers. Later, of course, computers helped reduce the amount of time for creating, changing, updating, and using such tools.

As time progressed, and when particular stress on compliance was generated by mine disasters or high fatality counts for a year, more computer-based tools and broader tools for accountability of compliance became useful. Risk assessment and management became popular, as did national and internal codes of practice and safety management systems and handbooks.

The keys, however, to excellent compliance with regulations, low injury rates, and minimization of risk are dedication of management to achieving progressive and realistic goals; a well-informed, trained, and dedicated workforce; observations and inspections to check on excellent performance; continuous feedback on achievement of goals (personally and organizationally); and revision of goals as achievements progress. The type of loop process applies to each person's job accomplishment, including management and supervision. The International Labour Organization's Code of Practice on Safety and Health in Underground Coalmines [20], the New South Wales Mine Safety's Safety Management Systems in Mines [21], and the US National Mining Association's CORESafety [22] provide guideline documents on pursuing such strategies for achieving safety excellence. Many other similar systems exist, and generally speaking, any one will suffice with good application. The "devil" is in the details and being able to gain total organizational support, at all levels, to pursue the system faithfully is imperative.

Achieving excellence in safety also hinges on the creation and adoption of new technology, which has occurred throughout coal-mining history. Much of the new technology in the 1920s through the 1950s addressed production (e.g., drilling machines, loading machines, mobile haulage conveyances, roof bolting machines, and continuous mining machines). However, significant developments were also achieved in making mines safer through scientific research by the Bureau of Mines. Some examples include making coal dust inert (from explosions) through rock dusting, methane control methods, making explosives permissible (unable to detonate methane in an explosive concentration), roof bolting hardware, etc. From the 1960s to 1980s, more robust, productive, and safe longwall mining equipment was developed, along with respiratory protection that removed 80% of the coal dust in the miner's breathing zone via a helmet with forced fresh air. More recently, the progeny agency of the Bureau of Mines, the Office of Mine Safety and Health Research in the National Institute for Occupational Safety and Health (NIOSH), supported and/or developed continuous respirable dust monitoring equipment, a rock dust meter that can estimate reasonably accurate inert content in mine dust, wireless communication and tracking technology, proximity detectors for mobile mining machinery, and more.

However, adoption of the technology, as it becomes available, is the critical factor in improving mine safety and health. Decisions by operators to adopt productivity improving machinery are easy to make, but decisions to adopt readily new technology to improve safety and health are often not made. Sometimes, legislation or rulemaking that drives the adoption of new safety and health technology forces it into the industry.

This occurred, for example, following the series of disasters during the period from 2001 to 2006 after which the MINER Act was passed.

One final point is made: The best operators link improvements in productivity and safety into the core of their decision making because they believe the adage that "a productive mine is a safe mine." The sanctioning of the Mine Safety Technology and Training Commission (MSTTC) report [23] as an independent tripartite study by the National Mining Association (NMA) demonstrated the commitment of its members to couple safety and productivity together as linked priorities. Boldly, NMA also ensured that the MSTTC was tripartite in composition (industry, miner representatives, and neutral parties, including government) and included a chief executive officer of a large corporation and a vice president of safety for another large corporation as part of their commitment. The aforementioned CORESafety, as one example, is a product of pursuing the commitment made by NMA members to the more than 71 recommendations of the MSTTC.

2.5 Making productivity and safety linked priorities

The primary goal of an equal-priority production and safety management system is to seek continuous improvement of performance in both aspects. This is accomplished by eliminating losses of life, disabilities, and injuries to workers; by building competent infrastructure and equipment; by prioritizing effectiveness and efficiency of the operation; and by making everyone aware of the financial performance of the enterprise. Pursuing this overall goal along with establishing the driving policies, effective planning of system and performance details, good execution of specified practices, monitoring of key performance measures, giving feedback on performance to the entire workforce, and correcting deviations from plans will achieve a safe, productive, efficient and cost-effective coal-mining enterprise. An embedded strong risk management subsystem is at the core of the detailed aspects of any production and safety management system.

Many countries, provinces within countries, and national as well as international governing bodies and organizations have developed such systems and/or codes of practices. For example, in the US, CORESafety was developed by the National Mining Association over a period of time following the report of the MSTTC in the aftermath of the Upper Big Branch mine disaster. Another excellent example was published by the New South Wales (Australia) Mine Safety organization. The management systems described in these documents contain essentially the same provisions, with the key aspects described in the previous paragraph fully developed.

Critical components essential to achievement of the goals of such a system, once created, are the following:

- Commitment of top management to it.
- Full commitment and indoctrination of management, at all levels, and the workforce.
- Creation of a culture of continuous improvement of the corporate-wide mining operations to a safe, productive, efficient, and cost-effective enterprise.

Table 2.12 **New South Wales coal fatal injury frequency rate (FIFR) and lost time injury frequency rate (LTIFR) 5-year averages [23a]**

End year	FIFR	LTIFR
2004–05	0.062	24.60
2005–06	0.041	21.04
2006–07	0.038	18.90
2007–08	0.038	15.95
2008–09	0.024	13.19
2009–10	0.024	11.01
2010–11	0.028	9.19
2011–12	0.021	6.97
2012–13	0.021	6.21
2013–14	0.021	5.42

An example of significant progress made by requiring a mine safety management plan (system) on a broad basis can be seen in New South Wales, Australia. In Table 2.12, significant continuous improvement on the fatal injury frequency rate (FIFR) for the coal sector is revealed over a 10-year period from 2004 to 2014 using 5-year averages. The FIFR is calculated as follows:

$$(\text{Number of fatalities} \times 1000\,\text{employees})/\text{number of employees}$$

Similarly, Table 2.12 also shows continuous improvement on the Lost Time Injury Frequency Rate (LTIFR) for the coal sector over the same period, again using 5-year averages. The LTIFR is calculated as follows:

$$(\text{Number of lost time injuries} \times 1,000,000)/\text{number of hours worked}$$

Reduction improvements for these metrics were 66.1% and 78.0%, respectively, for the FIFR and the LTIFR. Two other important features driving these improvements were part of the NSW 2004 Act, namely, the requirements of duty of care and risk management.

In the US, during October of 2010, MSHA was considering rulemaking on safety and health management programs for mines and sought comments from across the nation on the role of such programs for improving mine safety and health performance [24]. Seventeen written presentations were submitted on the topic. Presenters included large mining companies from coal and noncoal operations, organized labor, national associations, and a government agency. Although good evidence was given regarding the value of strong safety and health management systems, requiring all mining companies to do it, regardless of the size of the company and its mines, appeared problematic. There are tens of thousands of small mines in the US, and the burden of such programs on economic viability would have been tremendous for them. It is acknowledged, however, that the physical expanse of such mines and the limited number of workers at them are much smaller than for large companies. In the end, rulemaking on mine safety and health management programs was suspended, and MSHA undertook other administrative initiatives instead, as described previously.

2.6 Conclusion

There have been historical periods of sustained improvement of safety performance and productivity, but also periods of regression. The historical safety landscape has seen a steady decline of the fatality and serious injury experience, but relatively safe intervening periods were interrupted by reoccurrences of mine disasters with five or more fatalities, including as recently as 2010. Since 2010, however, there has been a dramatic improvement of all safety performance metrics at both underground and surface coal mines. Productivity continued to improve in surface mines during this period, but it has vacillated in underground mines.

Safety performance has been impacted greatly by research done at the US Bureau of Mines and NIOSH, mine equipment manufacturers adoption of better protective equipment and features on them, and improvement of ventilation and roof control technology. Enforcement of mining regulations, coupled with enforcement agency initiatives beyond regulations, has made an inestimable impact on injury metrics, including fatalities. Along with both technological change and regulatory action and enforcement, safety practices have improved over time, including the adoption of more systematic management methods among many operators that embrace both productivity and safety as joint goals.

Productivity has historically improved with adoption of more efficient mining methods and technology, but the quality and experience of the workforce plays a major role as well. For the industry as a whole, surface mining greatly improved productivity.

2.7 Future trends

For productivity and safety enhancement, new technology will continue to be developed. Sometimes in the past, technology development was driven by new regulations (e.g., continuous personal dust monitor, proximity detection system on continuous mining machines, and wireless communication and tracking systems). This trend will likely continue in the US as MSHA focuses on eliminating fatalities toward the goal of zero and seeks to address them through requirements of technology or by enforcing more systematic examinations with quick follow-up for identified hazards.

The challenge of attaining compliance with MSHA regulations and meeting obligations of other initiatives will likely force operators into using a much more systematic approach to improve their safety performance, regardless of company and mine size. As NMA member companies pursue CORESafety implementation according to its planned timeline, the use of mine safety and health management systems may grow among the larger companies, and possibly among medium-size companies. It is doubtful that such systems will be incorporated in very many small companies and mines; they simply do not have the resources to do it.

With that said, US fatality and injury rates will continue to be reduced significantly over time, particularly as it has over the recent past. It is quite feasible that soon there will be no coal-mining deaths in a particular year, especially since the size of the US industry is now diminishing, primarily because of market forces responding to US environmental initiatives, which target coal use for huge reductions.

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Zero Harm coal mining

3

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Safety (noun): "Freedom from danger and risks."
Concise Oxford Dictionary

"The greater danger for most of us lies not in setting our aim too high and falling short; but in setting our aim too low, and achieving our mark."
Michelangelo: Sculptor, painter, architect, engineer, poet. 1475–1564

"The most important thing to come out of a mine is the miner."
Frederic Le Play: French mining engineer and mine inspector. 1806–1882

3.1 Introduction

For as long as coal mining has been a human endeavor, safety, or the lack thereof, has been an overriding and inseparable challenge. In many instances, safety is the determining factor in the economic viability and survival of both coal mines and coal miners [1–4]. It is undeniably hazardous work when viewed from the perspective of optimizing production to the greatest extent possible, which requires accepting risk to a point that harm to miners and the mine cripples or destroys production capacity—as has been the case in many coal mine disasters. However, this outcome is clearly not inevitable. The story of coal mining and its development as an industry and as a catalyst for industrial and societal growth is the story of the double-edged struggle to simultaneously leverage and control risk. Over its long history, the painful reality of injury, illness, and death has forged a subtle fatalism across the coal industry (Fig. 3.1), an industry that desires and celebrates safety excellence, but which lacks clear consensus regarding how to achieve safety excellence. This is most apparent in countries like the United States (US) where significant differences of opinion exist about the structure of regulation in relation to industry best practice.

It is both appropriate and overdue that a text be devoted to the means and mechanisms of coal mining with "zero harm." This is not the first time a concerted effort has been invested in reviewing this topic, but the inherent stigma of danger and the industry's safety performance over hundreds of years has made serious efforts to communicate the messages contained in this book somewhat self-limiting. Few industry representatives, academics, government officials, or labor experts have acknowledged

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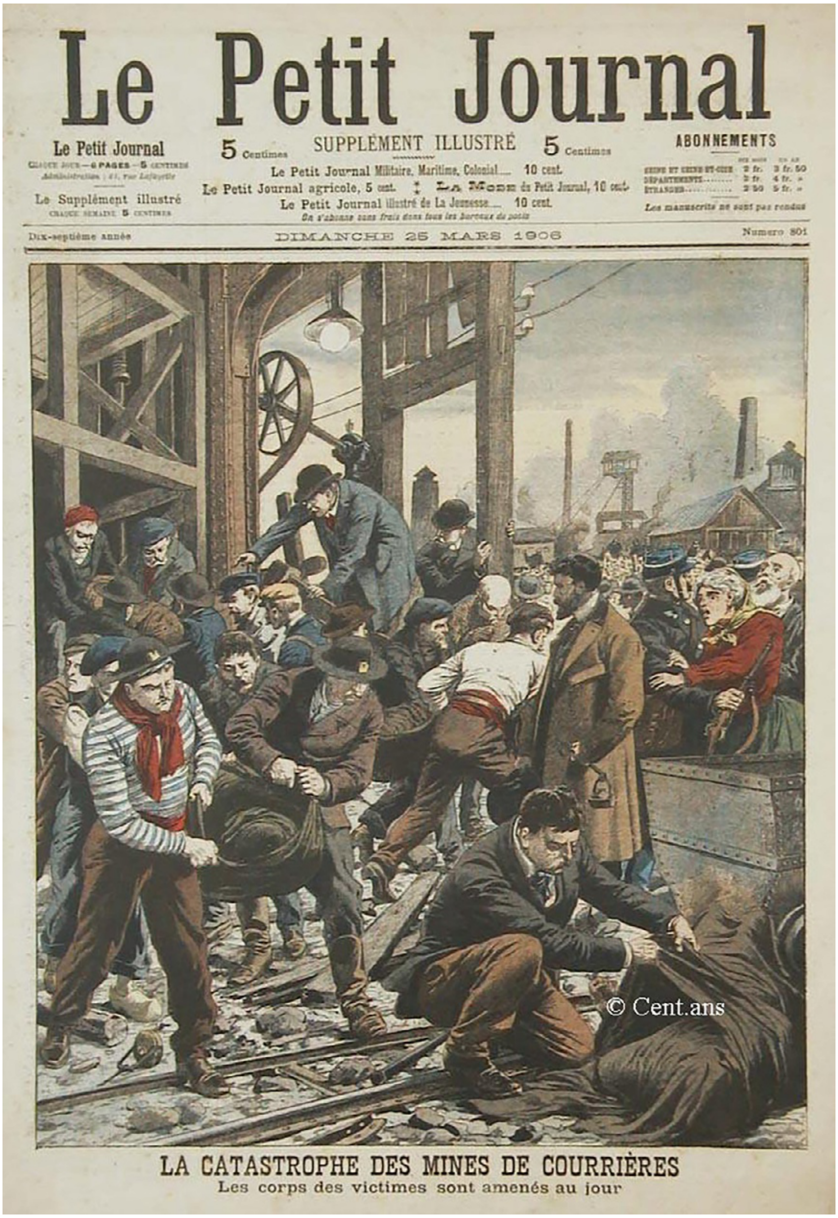

Fig. 3.1 Newspaper report of the Courrières Mine Disaster 1906.

the possibility as realistic. That is now changing. Not because the answer is suddenly obvious or some new technology has mitigated coal-mining risk to a level that makes it apparent, but because attitudes, beliefs, and coal company cultures are changing and experience in achieving profit expectations while maintaining operational risk at levels that approach Zero Harm are accumulating. The industry is learning and new possibilities are being integrated into the expectations of a new generation of coal-mining managers and leaders.

So, is Zero Harm the future of coal mining? Opinions vary and do not lack for conviction in their veracity. It is the author's contention that it is realistic for the coal industry to sustain safety excellence. However, this potential is unlikely based on the current status quo. It requires changes in thinking, strategy, and approach to sustainably advancing mine safety and health and it is important to understand the context in which improvement must occur to maximize the degree of improvement. This includes the history of the industry, mining methods and equipment, the application of risk management principles and methods, defining acceptable operational risk, the role of regulation, human error and behavior, and the opaque, but potent influence of leadership and organizational culture. But any review of Zero Harm in coal mining must begin with an understanding of the basic concepts of Zero Harm.

3.2 Zero Harm

The term "Zero Harm" is relatively new, first appearing about 15 years ago in the safety literature and used by a handful of progressive companies seeking to articulate the achievement of a very high standard of safety and health performance [5]. In the ensuing years, the term has motivated a remarkable volume of pronouncements, gesticulation, discussion, and debate about both its meaning and potential. Today, it is literally everywhere. There is no universally accepted definition of Zero Harm, but there is broad consensus that it implies operating a business, such as a mine or mining company, without any individual experiencing injury, or, causing no harm to anyone at any time at work.

The word "zero" serves as the catalyst for the growing controversy about the benefit of using the phrase in practice, whereas the word "harm" has been subjected to an unlimited number of definitions and interpretations. Zero is zero, but what does harm really mean? Many in the mining industry seeking to use it as a specific goal believe harm to be the regulatory-codified definition of a reportable or recordable injury or illness; i.e., an injury or illness that requires medical intervention. In most mining countries, this implies any injury or illness more severe than simple first aid. Other stakeholders suggest that even this interpretation is too lenient. They advocate from the perspective of the industry's moral obligation to absolutely protect people from any harm resulting from work. And herein lies the dilemma: while it is admirable to advocate for Zero Harm (of the strictest interpretation), it is a seemingly and statistically unproven ambition. There are clear safety performance improvement trends across coal mining, but no significant mining organization has achieved and sustained that level of performance. The result is a growing number of companies who publicly pronounce Zero Harm as being everything from their brand, vision, goal, culture, and core value to their most important priority. However, when companies fail to achieve their own version of Zero Harm, it frequently leads to concerns of unrealistic expectations, skepticism, demotivation, and even hypocrisy. There is also evidence that the inability to achieve Zero Harm can result in under-reporting of injuries, which not only undermines a company's intent and culture, but can increase risk by unintentional misapplication of risk management resources [6,7].

Another complication is the scope of Zero Harm beyond injury. Does it include occupational health, environmental impacts in and around a specific mine, impacts on the broader environment (e.g., regional air quality, watershed impacts, etc.), community harm, etc.? Again, there is no globally accepted governmental, nongovernmental, or trade association criteria to mandate decision making on scope. The broader the scope, the more challenging the achievement and accompanying positive and negative consequences are likely to be. While there are some government-mining agencies in countries like Australia that uses the term in a pseudo-official capacity that may affect the company-defined scope, this is an organization-specific decision. A growing influence on this decision is international sustainability reporting standards and criteria, which encourage some coal companies to treat multiple forms of risk as integral to Zero Harm; e.g., environmental emissions and impacts, biodiversity, employee development, economic opportunity for local communities, etc. [8].

Given the historical lack of parity between safety and occupational health in mining, it is important to include occupational health in any use of Zero Harm [9]. In doing so, organizations should recognize differences and similarities between occupational injuries and illnesses in defining the harm portion of Zero Harm as outlined in Table 3.1. Using clinical symptoms as a the measure of Zero Harm for occupational illnesses introduces a bias into the process as the harm in occupational illness begins when work exposures exceed the body's ability to absorb and recover from those exposures; i.e., harm is likely to occur before clinical symptoms are apparent.

A common question in this debate is: "Should Zero Harm be a vision or a specific performance goal?" It depends on intention and what is judged to be possible. Some mining companies select this audacious objective as a general aspiration or vision, while others intend that it serve as a specific objective and try to manage accordingly. Some see it as a means to help align employee decision making and behavior with policy. One thing should be clear—Zero Harm should not be used without providing an accompanying context for both internal and external stakeholders. This means senior management should only take their organization down the Zero Harm path with a full understanding of its meaning and the potential to achieve it given their current approach to safety and health management, which must be communicated to all others affected by these pronouncements. Using the phrase Zero Harm is not a prerequisite to achieving safety excellence that approaches or reaches zero harm; however, optimization of risk management and employment of tools to understand and minimize human error while enhancing organizational culture are essential [10].

If the leadership of every coal company were surveyed regarding their respective Zero Harm philosophy and approach, results would likely reflect a large majority who think it helpful to articulate Zero Harm, but who are uncertain about how to achieve it, or have an unrealistic view of its success. It is intuitive to those who work in high-risk industries, including mining, that absolute zero and Zero Harm are admirable and appropriate from an ethical perspective, but unrealistic. Since this text is intended to provide insight into achieving Zero Harm, the following guidelines highlight appropriate use of Zero Harm as either a vision or a goal:

Table 3.1 Characteristics of occupational injuries versus occupational illness

Injuries	**Illnesses**
Onset	
Primarily acute. Exception: nonsymptomatic internal trauma	Primarily chronic. Many occupational exposures require years before the development of clinical symptoms
Presentation	
Expression of injury is normally obvious to others or through pain to the victim	Expression of illness often subtle or unobvious, especially early in the illness development; e.g., lung dysfunction
Requirement for Medical Confirmation	
Yes, but most often to define the severity and treatment	Yes, occupational illnesses are easily missed or underestimated without medical surveillance
Consequences	
Functional: Loss of functionality directly related to severity. Potentially reversible *Economic*: Typically a loss of income for victim despite workers compensation. Can be substantial for victim, family, and company as severity increases	*Functional*: Loss of functionality directly related to stage of illness and impact to vital systems. Seldom reversible beyond a certain degree of dysfunction; e.g., noise-induced hearing loss *Economic*: Typically less than injuries. Can be substantial for victim and family, but disproportionately lower for company versus serious injury

- Ensure consensus regarding the meaning and purpose of Zero Harm. The process of coming to consensus will enhance the credibility of its intent.
- If the intention is to reflect a vision to protect employees at the highest achievable level, ensure that is clear to everyone. Don't assume. Acknowledge the challenge in making the vision reality.
- If the intention is a specific performance goal, define "harm" in a manner that is aspirational, yet realistic. Do not define the destination and promise a successful trip without having a very good map and compass and the knowledge to use them. Ensure your intent is clear. Don't assume.
- Consistently check to ensure the use of Zero Harm in any context is not promoting under-reporting. Confidential perception surveys can assist with this.
- Alternatives to the strict interpretation of Zero Harm focus on an appropriate definition of harm (i.e., damage, loss, injury, outcomes, etc.). For example, instead of Zero Harm, another more balanced option is: zero **h**azardous **a**ctivities (and/or actions) **r**esulting in **m**eaningful injury (exposure, illness, etc.); i.e., zero H.A.R.M.
- There should be no obligation to use Zero Harm or zero H.A.R.M. for relevance among peers or competitors, to make an impression, to appease a consultant, analyst, trade association, or other entity not directly responsible for the welfare of those at risk.

3.3 Coal-mining history

The history of underground coal mining suggests it is one of the most at-risk mining sectors and arguably one of the most dangerous industries in human history. It is among the most inherently hazardous of all commodities mined in the world. Many of the most significant disasters in modern mining history have occurred in coal mining, and especially in the underground sector (Table 3.2). This inventory of tragic outcomes highlights the number of lives lost, but excludes the number of miners injured, which often exceeded the number of fatalities.

Coal mining has existed in one form or another as far back as 3490 BCE based on supportive historical records, but likely occurred for thousands of years prior without archeological evidence. Early coal usage was essentially domestic with coal extracted from readily accessible surface sources [11]. First used as a source of heat for personal warmth and cooking, coal grew into a primary industrial fuel source driving work through direct heat, steam, and electrical power. The Romans utilized coal-fired furnaces to forge weapons and for building agricultural implements [12].

The industrial revolution in both Europe and North America between 1760 and 1840 was essentially underwritten by the advent of coal as a power source. This development fundamentally changed mechanized work and facilitated the expansion of economies of scale in the development of both consumer and commercial goods and tools. The demand for coal realized its first global expansion and was primarily satisfied through surface sources, which were more accessible to exploit. In Europe and specifically Great Britain, as large volume surface coal sources became rarer to secure, mining inevitably shifted underground. In both surface and underground environments, coal mining resulted in catastrophic loss of life and disability.

Coal mining has evolved through multiple phases of mining methods and the advent of mining equipment that serially improved production, but also contributed

Table 3.2 Catastrophic coal-mining disasters

Mine	Location	Date	Fatalities
Benxihu Colliery	China	1942	1549
Courrieres Colliery	France	1906	1099
Mitsubishi Hojyo Coal Mine	Japan	1914	687
Laobaidong Colliery	China	1960	684
Mitsui Miike Coal Mine	Japan	1963	458
Senghenydd Colliery	UK	1913	439
Coalbrook Mine	South Africa	1960	435
Wankie Colliery	Rhodesia (Zimbabwe)	1972	426
Oaks Colliery	UK	1866	388
Dhanbad Coal Mine	India	1965/1975	388/372
Monongah Coal Mine	US	1907	362
Dawson Stag Canon #1 Mine	US	1913	263

to mine safety through lessening the labor intensiveness of the work. Innovations have been consistent and impactful for their time. Beginning with hand mining, the industry has evolved through the use of beasts of burden to supplement human labor, the introduction of commercial explosives, and the development of mechanized systems, including steam-driven water pumps, diesel-powered mobile mining equipment, continuous miners, longwall technology, large-volume drag lines, and autonomous surface haulage units, etc. As economies of scale increased, the use of human labor to mine coal decreased, which benefited both productivity and safety. Today, longwall mines can produce more than 35,000 tons of steam or metallurgical coal in a 24-h period and tens of millions of tons per year, while surface coal mines in the Power River Basin of the Wyoming/Montana region of the US have achieved production exceeding 115 million tons per year.

As well as remaining a primary source for electricity generation, coal has been a foundational material for the development of society over the millennium and today remains at the center of debate regarding the economics of power generation and the environmental consequences of global warming. Some industry observers and critics have concluded that the risk of coal as a fuel sources is unacceptably high and warrants coal being phased out of use. Some of these observations originate from a reaction to poor safety outcomes, but have been supplemented with correlations between coal-fired power plant emissions and atmospheric warming—a broader environmental and societal risk [13]. Neither of these legitimate concerns negates the history of coal as a competitive fuel source.

In both the US and abroad, coal remains a central electric power fuel source due to its relatively low cost to extract and its stable market pricing relative to other sources including nuclear, oil, and solar. Its low relative cost is a driver for its use in developing economies; however, these economic benefits do not reflect human costs, which have created barriers to entry in developed countries with robust regulation requiring capital-intensive controls that are generally absent in developing countries. Coal is found is abundance in many countries and is relatively easy to burn and coal-fired power plants can be scaled up across a wide range of power outputs making it flexible and modular, especially using modern turbine technology.

A common historical theme in coal mining is that without taking risks, mining could not advance, both figuratively and operationally, at least not with economic justification. Given that coal is first and foremost a source of power and heat and secondarily a consumable in the production of steel, justification for coal mining as an essential human need is harder to rationalize. This struggle has been characterized by widespread and severe injury and illness of miners through both acute mechanisms and chronic exposures. Its value as an economically important fuel also reflects its risk as a material capable of generating explosive concentrations of gas and dust in addition to other hazards associated with underground and surface mining.

While the general public in most developing and developed counties is generally unaware of the source of their power supply [14], broad access to information through the internet and social media has resulted in broader and deeper advocacy, both passive and active, with regard to the social acceptance of coal mining. In countries like the US, Canada, and the European Union, an increasing percentage of the population is opposed to coal mining, especially in geographies in which there is proximity, but no direct economic benefit. Coal mines are no longer bound only by their access to viable coal deposits, their own financial resources, and the acquiring of government permits to develop and operate a coal mine. Today, they must also consider their social license to operate. While there is no one definition of social license to operate, aspects that are generally accepted include mutual economic benefits, organizational reputation as a responsible mining company, environmental impacts, and safety and health performance, among others [15].

3.4 Zero Harm framework

In light of the lack of consensus regarding the complete mining safety body of knowledge, but also the increasing acceptance of the concept and vision of zero harm, what do mining companies need to do in attempting to achieve and sustain this level of performance? Achieving and sustaining safety excellence is not just a matter of institutionalizing common sense or trying harder at doing the same things that are common practice today. Such a goal is among the most complex and multifactorial challenges facing the industry. It involves many variables that can influence safety and health outcomes, including, but not limited to: fitness-for-work, competency, process design, equipment optimization, rock mechanics and geophysics, the effectiveness of risk management and hazard controls, behavior and human error, organizational culture, and leadership. Bringing a consistent degree of control to these many variables affecting mining risk requires an integrated strategy. There is no simple solution. That is not to say that there are not mines and moments when safety is enhanced and preserved through seemingly simple activities and actions, random and otherwise. However, all indications suggest the "keep it simple" strategy has not resulted in zero harm performance on any meaningful scale to inform the rest of the industry and it is unlikely to sustain it should it be achieved. Zero Harm is a complex, difficult task.

Broadly speaking, the potential to achieve and sustain zero harm in coal mining depends on three key domains of mine management: (1) systems that are at the center of controlling mining risk: verifying worker qualifications, fit-for-duty status, training, and competency; mine engineering, including fixed and mobile equipment selection, use, and maintenance; energy distribution and application; information and communication systems; comprehensive risk assessment, including hazard control selection and verification; systems for change management, effective supervision, continuous improvement, error management, and regulatory compliance; (2) culture, as measured by organizational climate, and the organizational characteristics associated with strong, effective safety and health management such as open communication, employee consultation and empowerment, trust, accountability, vigilance relative to risk, among others; and (3) leadership. The most significant influence on organizational culture is the collective behaviors of leaders. Leadership development based on defined competencies that drive performance and the organization's culture helps to lay a foundation to optimize the systems that are designed to manage risk to a level that Zero Harm is realistic. Furthermore, it is not enough to manage the challenges associated with safety systems, culture, and leadership; but it is critical that this work be integrated [16]. Otherwise, the complexity can become unmanageable.

Safety improvement ideas regularly percolate and reverberate across the industry with two primary sources of origin: dissatisfaction with status quo performance and a lack of consensus regarding the optimal approach required to achieve safety excellence, including zero harm. Serial experimentation with safety initiatives, without the ability to make empirical conclusions regarding intervention effectiveness, together with a tendency to seek silver bullet solutions work to hamper the industry's ability to clarify the optimal strategy and subject it to continuous improvement analyses. This is ironic given the disciplined and structured approach applied to identifying operational enhancements through logical, incremental, process improvements and the application of statistical experimentation tools to help define what works and what does not. In the context of Zero Harm, safety would benefit from a greater dose of process improvement.

Coal production can be optimized while maintaining a sufficiently low enough risk profile to enable a mine and a whole company with proactive leadership and effective safety systems to achieve safety excellence. Above all, pursuing safety excellence in the form of Zero Harm or similar manifestations is an exercise in systematic risk management. In the zero harm framework, one of the most significant changes required to facilitate zero harm performance for those who are not already using it is an accurate understanding and control of operational and human risk.

3.5 Coal-mining risk

Coal-mining safety is a reflection of coal-mining risk and its unique chemical–physical properties as a carbonaceous mineral. Risk has several meanings that are important to coal safety. Risk is both a perception held by every person who works in the industry and a technical concept open to qualitative and quantitative assessment. Perceptions of what is acceptable and unacceptable risk can vary widely and for very different reasons. However, the lack of consensus regarding how coal-mining safety and health risks are viewed is a challenge for which companies motivated to achieve zero harm must address. A lack of normalization or consensus is a major challenge.

It may be obvious that the benefit of risk assessment is in understanding potential incidents that have negative impacts. However, it is worth mentioning that the coal industry could never have evolved into its globally important status today without accepting risk. Risk is a necessary element in successful coal mining because it is not possible to mine coal, by any method, in any location without accepting some degree of risk. That is, risk is both a negative and a positive concept. When a coal mine is developed and operated with high levels of productivity and quality and low levels of frequency and severity of safety events, it may be characterized as managing risk positively. Given the nature of mining in general and coal mining in particular, there is no risk-free coal mining. To truly eliminate risk would require cessation of mining operations. As such, it must be understood that zero harm performance cannot be achieved through comprehensive risk elimination. It is the management of risk at a level that results both in appropriate productivity and safety performance that must be the goal.

There are seven characteristics of coal-mining risk:

1. *Constant change*. The essential nature of a coal mine is to exploit the ore body as a means and indicator of the extraction of coal from the ground. Change as a dynamic variable increases the potential for risk to result in harm to both people and equipment. The more change that occurs, the greater the difficulty in understanding and controlling risk. Coal miners most commonly experience fatal injuries when nonstandard geological conditions, work tasks, and behaviors occur. Change is more often than not a promoter of increasing risk.
2. *The three-dimensional nature of geological risk, especially in underground mines*. Whether it is strata, gas, equipment, or tools, risk is present above, below, in front of, and behind miners while they work. In both surface and underground operations, the potential for miners to be exposed to rock falls, subsidence, uplift, bumps, falling equipment, proximity to mobile equipment, inundation, uncontrolled energy sources, etc. is omnipresent.
3. *Imperfect understanding of rock mechanics and rock behavior*. While the industry's accumulated knowledge of coal geology, geophysics, and rock mechanics continually increases, it remains incomplete in terms of the ability to exactly and consistently predict the behavior of coal and its host rock. There is a substantial body of knowledge regarding coal characteristics and behavior that is supplemented with new research findings and new information and perspectives derived from mining operations. However, the history of coal-mining incidents reflects retrospective recognition of unseen or inadequately characterized geological risks that contribute to increased risk.
4. *Chemical-physical properties of coal*. By its nature, coal has the potential to contribute to fires and explosions at surface mines, but especially in underground mines. Of course, contending with materials that are potentially combustible, flammable, and/or explosive is not unique to mining [17]. The petroleum industry is primarily defined by this risk from exploration to retail distribution and use of gasoline and other products. Offshore petroleum production and the refining process are at the center of that industry's risk profile, but they substantially minimize risk by isolating exposure to flammable gases and liquids through comprehensive controls. When operators or maintenance personnel are exposed to explosive hydrocarbons during instances when the otherwise closed process unties, risk assessment is required; yet, in underground coal mining where potentially flammable and explosive concentrations of gas and dust are present, operators and maintenance personnel actually work inside process equipment that is analogous to petroleum refining; i.e., coal mine drifts, crosscuts, and adits.
5. *The scale of mining equipment and tools*. As mining mechanization and economics of scale grow, the size of fixed and mobile mine equipment concurrently grows in relation to miners who assemble, operate, and maintain them. Surface mine haul trucks are the size of a three-story building, drag lines often eclipse 10 stories in height, and longwall installations can reach more than 1000 ft in length. There has also been a proportional increase in the size of tools used to assemble and maintain mining equipment. It is common to see wrenches in mine maintenance shops that are more than 36 in. in length and weigh in excess of 100 pounds. The greater the size differential between equipment and those who interact with it, the greater the risk. Larger hand tools, trucks, shovels, longwalls, conveyors, etc. are necessary to leverage economies of scale in the vast majority of operating mines today. The size and complexity of equipment increases while size and susceptibility of miners remain static and in some instances can be worse as in the case of aging workforces with declining reaction times, strength, and flexibility, among other measure of health and fitness.

6. *Proximity of miners to the working face and active equipment where coal is mined*. The advent of semiautonomous and fully autonomous mining equipment, particularly mobile equipment such as load-haul-dumps (LHDs), longwalls, roof bolters, continuous miners, and shuttle cars in underground mines; haul trucks in open pit mines; and drills in both surface and underground environments, lessens the risk by increasing the separation between operators and the equipment. However, this separation does not help those maintaining the equipment and is in itself dynamic and dependent for the most part on operator judgment and therefore human error. It is incomplete or absent for certain mining jobs such as in-shop and in-mine mobile and fixed equipment maintenance, surveying, and utilities crews. Sensors help to detect proximity zone incursions, but are susceptible to a variety of faults and failures.
7. *The human element*. Miners are at the center of mining safety and their decisions and behavior have a substantial impact on effectiveness of overall mine safety as well as their own individual safety and that of their coworkers [18]. A number of prominent incident causation theories identify human error and behavior as a very significant contributor to negative outcomes such as injury and property damage. It is easy to assume that thorough training, clear standard operating procedures, and effective supervision should minimize the potential for human error in the form of unsafe behavior. In the vast majority of instances, this is true; however, while important, these systems do not guarantee compliant behavior and correct decision making. Error in the form of poor decisions and/or at-risk behavior occurs daily in coal mines and, despite common beliefs that these outcomes are under the complete control of miners, there is evidence that human error has been a contributing factors in many coal mine disasters and it is not isolated to miners alone [19]. Managers are also susceptible to this form of risk. Many types of controls have been introduced in the coal industry from behavior-based safety (BBS), behavior modification, progressive discipline, human performance management, and values-based interventions, among others, with varying degrees of effectiveness. Research suggests that human error is not always preventable, but the best opportunity to do so involves an understanding of the sources of error and developing controls to mitigate its negative impacts on safety. Some error is intentional and person centered, other is person centered but outside the conscious control of the person (i.e., they are unaware that their actions are in error). There are also human failures for which the primary contributing factors are related to engineering, managerial, systems, and operational defects.

These seven characteristics are important as they help to refine risk assessment and management processes and to improve awareness both for miners and managers [20]. There is no such thing as zero risk in mining, including coal mining. As such, any serious approach to zero harm must not focus exclusively on *not having* incidents, but on managing operational and human risk to a level that will minimize the potential for incidents to occur.

3.6 Risk management and acceptable risk

The focus on risk management is a relatively new development even if the consequences of risk have been viscerally understood by people in their daily lives for millennia. Humans have been practicing risk management since the dawn of man as an inherent human tendency to recognize, understand, and avoid the consequences of risk.

While risk management is a universal human ability, people do not necessarily see a hazard and its related risk in the same way. One person may see a hazard as being more risky than another person. This perspective is reflected in theories that have been developed to understand individual differences in the response to similar risks. One such theory is risk homeostasis, which postulates that all people have a set point for acceptable risk and will function in their daily lives to optimize risk to their own benefit. When risk is perceived to be unacceptable, they will change their behavior or the circumstances of the exposure to the risk [21]. When they perceive the risk to be lower than is generally recognized; either by their peers, government regulators, company management, or others; they may seek to accept more risk as they may feel that they will not be negatively affected, or if they are, that the cost-benefit of the circumstance is justified. Inputs into this conscious and subconscious decision making may be economic, psychological, social, or cultural.

Mining risk can be understood through a technical lens that attempts to qualify and/or quantify mining hazards using a variety of assessment techniques. At the most basic level, understanding coal-mining risk requires identification of all relevant hazards and comprehension of: (1) the probability that a specific risk will be "expressed" as an incident or event; and (2) the consequence of such an incident. Mechanisms to inventory, assess, and communicate risk take the form of existing regulatory criteria, risk registries and inventories, as well as multimedia techniques such as images and video to minimize the lack of understanding due to language. There are a wide variety of techniques to facilitate this assessment from simple brainstorming to "what if" analysis, bow tie analysis (BTA), hazard and operability studies (HAZOP), failure mode, effects and criticality analysis (FMECA), and fault tree analysis (FTA), to list a short selection. These risk assessment techniques have the same core functionality: use of professional judgment and, if available, actual failure probability and consequence data, to express the product of the probability and consequence of any coal-mining hazard of interest.

As described previously, the application of risk-centered management systems is the primary tool to control risk. Consensus risk management standards, such as ISO 31000, have been developed for application to mining and other industries and many mining countries have integrated risk management into their mine safety legislation and regulations [22]. Most risk management processes are implemented as the most critical element of a safety and health management system. Other national or international consensus safety and health management systems include BS 8800 (UK), OHSAS 18001 (international), ANSI Z10 (US), C-1000 (Canada), and ISO 45001 (international). Examples of successful mining industry initiatives based on risk management principles include the US National Mining Association (NMA) CORESafety management systems [23]. The International Council on Mining and Metals (ICMM), an international trade association, has advocated refinements to the traditional generic risk management process by increasing focus on critical (aka, catastrophic) risk and using risk-specific management systems to define appropriate controls and verify their ongoing effectiveness [24].

Mining companies across the globe, including the vast majority of large mining companies, are currently using risk-centered management systems and many report anecdotal experience in reducing risk and safety incidents. However, despite reports of temporal benefits, there is an inadequate body of research that provides clear, empirical support to define the real intervention effectiveness of safety and health management systems. The reason for this lack of evidence is two-fold. First, this type of research is extremely complex given the large number of dependent variables that compose these systems and the varied independent variables that reflect outputs and system performance. Second, this research tends to be very invasive and can be disruptive resulting in difficulty gaining access to mines willing to cooperate with research needed to help answer what many observers see as an esoteric issue given the volume of anecdotal evidence that safety and health management systems are effective. Therefore, it is important that the global mining community collaborate to facilitate these important research outcomes.

Acceptability in the level of risk a mining company (and its stakeholders and shareholders, assuming it is publicly owned) is willing to tolerate in human and property loss is an underserved but important concept in any discussion of Zero Harm and related topics. Are coal-mining companies required to define a specific level of acceptable operational risk? Is Zero Harm such a standard? Zero Harm is an aspiration relating to safety and health outcomes. If an organization was aware of the specific degree of risk necessary from all its operations in order to achieve Zero Harm, it would by default develop a level of acceptable risk, or at least a range for acceptable risk. This is a difficult activity to accomplish given the inherent lack of reliable failure data for various mining equipment and activities to aid in defining appropriate risk levels; e.g., probability $\times$ consequence $=$ risk. As such, few companies are able to rely on objective data to determine risk calculations. The greater the subjectivity introduced in risk assessment, the greater the potential for risk error, risk blindness, and the likelihood of having the risk expressed in the form of an incident. Subjective error, whether an individual making a behavioral decision or a mining engineering team making a team-based decision, in the formal risk assessment process can introduce bias into effective risk management.

There are few statutorily defined levels of acceptable risk for safety in mining. By default, organizations make decisions regarding acceptable risk every day that they operate through decisions and actions taken. The exception is governments who utilize numerical risk criteria when promulgating occupational and environmental exposure standards. For example, the application of a one-in-one million (10^{-6}) lifetime risk criteria to the development of pulmonary fibrosis from chronic exposure to a specific airborne dust defines an acceptable level of risk. That is, setting the occupational exposure limit of an airborne dust such that the resulting exposure limits the potential of developing pulmonary fibrosis to no more than 10^{-6}.

In the absence of government mandates, each company must define acceptable risk for themselves. For many, it takes the form of geotechnical risk as safety factors or probability of failure in relation to ground control or slope stability [25]. For others, acceptable risk will be expressed as a variety of qualitative decisions using decision-making criteria such as the hierarchy of control so that when a critical risk control falls outside the defined level of control within the hierarchy, it requires approval from a higher level of authority in the organization.

A company's business model and resources can also have a direct and indirect impact on defining acceptable risk. For example, if a mining company has marginal capital resources and elects to open a mine that is capital intensive, it may absorb more risk in its mine design and operational practices because it cannot afford to do more. If the mine's profit margin in low or negative, it may not have the resources to make additional improvements. In this general scenario, the mine may not be out of compliance with governmental regulations, but it may carry substantial risk through operational practices that indirectly motivate human error by making it clear to the miners that the company lacks the resources to make additional improvements. The best intentions in relation to zero harm are less likely to result in adequate risk management when process and operational risk cannot be reduced by engineering, appropriate infrastructure development, and equipment selection and maintenance. Without saying anything directly about acceptable risk, companies define the prevailing sense of what is acceptable in decisions they make and the approach they take to engineering and operating any coal mine. If an underground coal mine has issues with gas intrusions that are outside the ability of the mine to control at the face, should the mine be operated, even if management is verbally committed to achieve zero harm?

If a company defines minimum criteria for developing a coal-mining project using internal rate of return (IRR) or return on investment (ROI) and the project team does everything it can in designing the mine to meet or exceed the IRR/ROI criteria, but in doing so, accepts greater-than-desirable risk, that risk is passed forward to the mining operation. It may be poor-quality ground that does not lend itself to consistently effective controls, but it is essentially left to the operations teams to address even though it could have been done more effectively with a more robust mine design or mining method.

When there are regulatory or legislative barriers to entry in permitting a mine, there will be an indirect application of acceptable risk by the government. For example, if a government requires mining companies to develop a safety case to justify their control of risk as a prerequisite for obtaining an operating permit, the government imposed barriers to entry that are in part a reflection of acceptable risk. In jurisdictions with fewer government requirements, risk that is not controlled in the permitting process may be transferred to the mine operations versus being "engineered out" by design. In some countries, this results in a limited number of large operators with better resources. In other countries, the barriers to entry are less substantial, and depending on market economics, industry can be populated with small companies whose miners are likely exposed to the same generic risk as all other coal mines, but who are less likely to have the knowledge and financial resources necessary to optimize operational risk management.

3.7 Regulation and legislation

Many of the daily activities intended to control unacceptable risk that occur in coal mines around the world are defined by national, provincial, and state legislation and regulation. In some jurisdictions, coal companies are expected to develop their own approach to complying with these governmental requirements whereas in other countries, compliance is highly prescribed and rigid. There are a wide variety of regulatory and legislative systems found throughout the industry; however, they can be divided into three primary categories: (1) hazard-specific prescriptive systems, (2) performance-based risk management schemes, and (3) some combination of the two.

The majority of regulatory schemes in developed mining countries began as hazard-specific systems in which the regulator defines specific hazards that if present in a coal mine required specific controls to be implemented. These controls were often in the form of programs with training, documentation, and specific parameters in relation to mine design, ventilation, ground control, fixed and mobile equipment operation, and emergency procedures. The advantage to such schemes is that they ensure a minimal level of risk control for key hazards and make it easier for the government to define compliance and noncompliance. The disadvantage of this type of scheme is that it does not mandate for more systematic assessment and risk-appropriate controls to be applied, as is characteristic of the performance-based risk-centered schemes. The advantages of the risk-centered approach are that it requires each mine to understand and control all relevant risk (not just those that are listed in the regulation under the hazard-specific approach), it allows for the introduction of new technology, mining methods, and new risks, without changing the regulations. The disadvantage of the risk-centered approach is that each mine and company will have its own system of risk management, which makes enforcement more complex—one size does not fit all. The US has used a hazard-specific prescriptive scheme for safety regulations since the 1970s whereas the commonwealth countries evolved to a performance-based risk-centered approach.

To appreciate the difference, one need only examine the present differences that exist in the US and Australia. While both countries had relatively similar beginnings with respect to the development of mine safety legislation and regulation, they have since taken different paths relative to the improvement of their respective mine safety schemes. On August 14, 1994 the Moura coal mine in Queensland, Australia exploded in a disaster that resulted in the death of nine miners. Australian government officials, in conjunction with academics, unions, and enlightened industry representatives, came to the conclusion that any incremental response to the specific circumstances that caused the Moura disaster would be inadequate to prevent future occurrences in Australia. Not less than a radical overhaul of the country's existing regulations was warranted. Not an update or modification, but a reassessment of first principles and restructuring of the entire system based on what was then beginning to be seen as the most progressive approach to mine safety: systematic risk management [26].

The resulting framework that eventually formed the structural elements of Australia's current mine safety scheme goes beyond just the manner in which risk is addressed and includes three interrelated provisions: risk management, duty of care, and legal accountability through industrial manslaughter legislation; i.e., civil and criminal liability for company officers from frontline managers to boards of director. These three elements have motivated a broader understanding of risk and reduced the willingness of coal companies to accept residual operational risk that will not protect miners and minimize liability for managers and officers of the company [27].

At the same time, and remaining essentially true through 2016, the US Mine Safety and Health Administration (MSHA) has remained steadfastly reliant on a more prescriptive approach to mine safety regulation. While some changes and amendments have been made to the Federal Mine Safety & Health Act of 1977 (Public Law 91-173), in relation to the general approach of identifying hazards and defining appropriate controls, it remains unchanged. What has changed has been the approach to enforcement, which has been defined more by political philosophy than science. The two dominant political parties in the US see regulatory enforcement with polarized perspectives: one sees unrelenting compliance enforcement using primarily monetary civil sanctions but with the option of elevating serious violations into a criminal enforcement activity. The second party generally believes the most effective approach to regulatory enforcement to be a mix of enforcement and compliance assistance, including education and consultation. Regrettably, neither perspective has been supported with unbiased data demonstrating the effectiveness of the respective approaches.

Today, it is increasingly rare for the industry to experience disasters on the scale known to the industry for hundreds of years as measured by outcomes such as fatalities, fires, or major explosions. From 1900 to 2006, the US coal industry experienced 104,398 fatalities, or an average of 984 per year for 106 years with the maximum count of 3242 in 1907 [28]. From 2006 to 2016, US coal mines experienced 227 fatalities, or approximately 23 per year [28]. Clearly, performance has improved exponentially. The question is, how and why did safety performance improve, and if the status quo is maintained, will the industry reach zero harm? Not surprisingly, MSHA views the improvement primarily as the result of consistent, strong enforcement of the regulations [29], while the industry sees the improvement more in relation to the discretionary risk management activities undertaken beyond MSHA's statutory obligations [30]. When national mine safety regulatory schemes are aligned on merit with the industry's understanding of best practice for achieving safety excellence, there is a clear mechanism for continuous improvement.

Notwithstanding the importance of regulation is driving performance improvement in the coal industry worldwide, it is important to recognize that the closer a mining company comes to achieving and sustaining zero harm performance, the less likely mine safety and health legislation and regulation will play a role in the company's approach to gaining additional improvements. Regulations are essential for mining companies who have not developed their own risk management systems, leadership, and culture to forge a continuous improvement path to zero harm. However, the test for the independence and maturity of a company's culture and approach to zero harm is how much difference the enforcement of mine safety regulation makes in driving performance improvement. What do coal-mining companies do when they are already in full and consistent compliance with government safety regulations, but have experience fatal injuries and illnesses?

Ultimately, for the global coal industry to have a reliable probability to achieve zero harm safety performance, there needs to be a greater understanding of the most effective role of government safety legislation and regulation. Many stakeholders believe that that will require a more objective analysis of different regulatory schemes in relation to simultaneous objective assessment of effective risk management strategies and systems—currently the most widely adopted approach to achieving safety excellence among coal companies with industry-leading performance. However, even with that complex outcome, no regulation can govern critical qualifiers and characteristics of zero harm performance: human error, organizational culture, and leadership.

3.8 Culture

Culture, including organizational culture, mine culture, and safety culture, has been a topic of discussion in mining for many decades, but has only come into focus as both a strategic and tactical element of safety and health management in the coal industry in the last decade or so [31,32]. Organizational culture is generally accepted to imply the collective beliefs, attitudes, priorities, values, behaviors, traditions, approaches to work, means of communication, etc. and often simplified to: "the way things work around here." Subcategories of organizational culture are believed to be variants of the overall organization culture. While it is a complex concept when viewed from a strictly academic perspective, it has become more accessible and relatable since the introduction of the concept of "climate." Organizational climate is based on the perception that employees have regarding aspects of an organization's culture [33]. Climate enables mining companies to gain insight into aspects of their culture using confidential perceptions surveys. There is strong evidence that positive cultures contribute to organizational effectiveness and productivity [34].

Culture in and of itself is not enough to significantly affect safety performance in the absence of effective risk management and mechanisms to control human error. However, a sociological and psychological construct culture can affect decision-making behavior, and as such the importance given to the tools used to manage safety and health in the coal mine environment. Risk-centered management systems are only as effective as the culture in which they operate. Regardless of how effective the design of a management system may be, if management through words and deeds shows disinterest or is dismissive of the importance of line management accountability for using the system, more likely than not, the system will be ineffective. Alternatively, if the management system is viewed as "the way we do work around here," it will likely be an effective means to control risk.

With increasing industry experience and structure research, there is growing clarity regarding the organizational characteristics that are correlated with strong safety performance in high-risk work environments. These include, but are not limited to: trust, accountability, effective communication, safety as a collective value, wariness and vigilance (regarding risk), and integration of safety into organizational decision making, among others. As coal companies seek to optimize their strategy to achieve safety excellence, culture is revealing itself to be necessary, indirectly measurable through organizational climate [35], and manageable once its strengths and weaknesses are understood. Among the keys to understanding means to improve organizational safety is the recognition that while organizational culture is affected by attitudes and behaviors of all employees, it is most acutely responsive, for better or worse, to the collective behavior of management from the front-line senior leaders.

3.9 Leadership

Leadership is the mechanism by which one or more persons influence a group of individuals to achieve a common goal [36]. The importance of leadership both to operational excellence and safety is not in dispute [37]. This observation has been demonstrated across national cultures, industries, militaries, genders, and a variety of organization types and business models. In the context of safety, leadership plays a crucial role in optimizing culture and influencing attitudes, behavior, effort, and safe decision making. Leadership is the primary influence on organizational culture [38].

While there is no one optimal approach to leverage leadership behavior to enhance organizational culture, structured leadership development is an obvious option. Whether using a competency model to define the behaviors most likely to drive the culture and optimize safety systems, or another approach, the only poor option is not attempting to improve the leadership of all coal mine personnel who control the means and mechanisms of production, and therefore safety. Competencies that have been associated with enhanced safety include, but are not limited to: trust and integrity, effective communication, having a relevant vision, accountability, personal example, conscientious decision making, etc.

Zero harm is an aggressive vision and/or goal involving advanced technical and sociotechnical systems and effort. Without unambiguous and effective leadership promoting a credible zero harm vision, and the accompanying management decisions consistent with risk management that makes zero harm a realistic possibility, Zero Harm may be doomed to be another fad visited upon the industry, but not resulting in the change all coal mine stakeholders seek—to optimally protect the industry's most valuable asset: its miners.

3.10 Conclusion and future trends

The coal-mining industry is expanding geographically and must pursue coal deposits with greater geotechnical risk due to the exhaustion of more accessible, lower-risk deposits. During this change, a host of voices are heard advocating for higher standards of mining safety. Not just standards that are higher than the historical trend, but the highest achievable. These voices come from multiple stakeholders, both in time and place, but they share a common theme: despite the inherent risk associated with mining coal, the coal community can and must do all that is feasible to protect its most important asset—the miners. Zero Harm, zero harm, and zero H.A.R.M. are sharpening the debate about safety and health excellence. Expectations are increasing as is the acceptance of new thinking outside the refrain of relying solely on regulatory compliance to optimize performance. Today, it is better understood that coal-mining companies, regardless of size, must focus on systematic risk management, including critical risk controls, human error management, and fit-for-purpose management systems, all based on a foundation of culture, driven by effective leadership development that expects safety excellence and makes the necessary decisions in terms of mine design and operation to achieve the desired results.

These challenges will not be realized solely through the concerted efforts of mining companies and their partners, the regulators. It will require the full cooperation and collaboration of the coal-mining community sharing a common vision, the willingness to recognize an optimal overall strategy, and the transparent sharing of achievement and lessons learned from the implementation of that strategy.

Certain future trends will foreshadow the wholesale shift in the thinking and potential to achieve zero harm in all its variations.

- There is need of more leadership in setting industry expectations regarding safety management and performance. Company will achieve the level of performance they come to expect (and for which they are willing to manage accordingly). Safety should not control coal-mining companies, the companies should control safety.
- Regulatory compliance is not sufficient to assure zero harm. Mine safety regulations cannot prescribe culture and leadership, nor minimize human error.
- The industry needs a tripartite debate and decisions made regarding policy that defines the most effective structure for coal mine safety regulation in relation to current knowledge about how to achieve safety excellence and not just what is most expedient in terms of enforcement.
- The closer a mining company comes to zero harm, the more important the sociotechnical aspects of mining risk management become. There is a need to better understand how to achieve this type of organizational development; i.e., culture enhancement and leadership development.
- There needs to be greater collaboration and transparency within the mining community to ensure lessons learned within any one company are learned collectively.
- There is need of a common, global taxonomy of root causes to facilitate the sharing of information regarding how to improve the industry's safety and health systems.
- There is need for a global consensus regarding the appropriate body of knowledge for effective safety and health management that, based on the perspective found herein, is more complex and multifaceted than the usual content of safety regulations. The body of knowledge will need to be known by both safety and health professionals and line management alike.
- There is need of consensus regarding acceptable risk that guides coal companies toward safety excellence, including zero harm, across the entire life cycle of mines.
- There is need for better funding for and industry access to research to enhance understanding of safety and health management system intervention effectiveness.

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Productive, safe, and responsible operations are not possible without visible safety leadership

4

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4.1 Who is a safety leader?

Traditionally, working personnel in most organizations are structured in a "chain of command," in which managers and supervisors oversee and delegate to their assigned subordinates, often arranged in various groups or departments that serve different functions. These leader-subordinate hierarchies are common not only within business organizations, but also reflected in the culture dynamics in all parts of society, like politics and government, the military, and families and households.

In most cases, the roles and responsibilities of leaders and subordinates are often thought of as definite and distinct: people either give direction or receive it; they are either in charge of others or not. However, in safety-sensitive industries like mining, leadership pertaining to workplace safety is not limited to only those in supervisory or management positions.

When speaking to groups of employees in consulting and training work, the authors often ask individuals to identify themselves if their positions are directly related to safety. Typically, only a few raise their hands, like the "Safety Manager," "Safety Supervisor," "Safety Technician," or others whose job title has the word "safety" in it. On occasion, employees quickly detect that they are being asked a trick question, because in fact, *everyone's* position is directly related to safety.

Safety leadership, simply put, involves engaging in and maintaining behaviors that help members of a group achieve safety goals. In this sense, it is possible for anyone at a mine site to be considered a safety leader. No matter one's title, position, or level of experience, *in mining, safety is everyone's job*. And in turn, everyone has the potential to actively engage in behaviors that help to promote safety. Safety starts with each individual at a mine site, as well as with the external stakeholders who also play a role in ensuring a productive, safe, and responsible workplace.

4.2 Culture is "the way we do things around here"

Culture involves customs and social norms among groups of people—how they greet and speak to each other, their conversational themes, methods and norms of their interactions, their routines. These practices are what separates one group from another, and

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also what holds them together and allows them to be sustained. Cultures can be examined at any scope, like that of an entire nation or people (e.g., *Mayan* culture; *Japanese* culture), or subcultures residing within larger cultures, whether generalized or specific (e.g., *New York* culture; *trucker* culture; *nerd* culture; *classroom* culture; *club* culture). Cultures are present wherever social interaction takes place with regularity, even among small groups. For instance, at an industrial operation, the subculture among front-line workers in a particular department or area, who spend much of their on-the-job time together, is likely different in many ways from cultural standards and norms exhibited among upper management. It is likely also different in some ways from that of front-line workers in different departments of the same company, as long as there is little interaction between these groups.

Regardless of the scope or societal stratum of a particular culture, its norms were likely established intrinsically and organically, albeit shaped by cultural forebears or the larger culture in which it resides. For example, the subculture among a team or department in an organization would be influenced by official operational procedures, and the language used would incorporate the jargon relevant to the industry and task. The interaction between group members would also be dictated by norms of the greater culture of their geographic location as well as their ethnicities. But unique social markers would define the subculture, too, like inside jokes, nicknames, common themes of conversation, slang, and even light-hearted ritual behaviors that arise from familiarity and a sense of fraternity among the work group. Such specific and esoteric things would not likely translate easily to management, or sometimes even across different work groups in the same organization.

The collective attitudes and behaviors among all personnel toward workplace safety is an inherent aspect of cultures and subcultures in a safety-sensitive workplace like a mine site. The concept of "safety culture" has been gaining momentum and become more commonplace in safety-sensitive industries as the impact that behaviors and attitudes among all personnel have on safety has become more visible and understood. The benefits of a positive safety culture—improved safety performance, production, reputation, and so on—are also becoming more widely known. Workplace safety expert E. Scott Dunlap puts it plainly: "Efforts in safety and productivity are not mutually exclusive. Development of a safety culture through an emphasis on protecting employees has positive consequences on business growth" [1].

Because noncompliance to safety regulations, whether in the aftermath of an accident or discovered during an inspection, is met with citations and fines, there are many examples of safety systems that handle risky or unsafe behavior with serious reprimand, which mirrors the punitive nature of citations and fines. Furthermore, it is not uncommon for safety training refreshers to be thought of as punishment—by both employees and management—following an incident.

Rosa Antonio Carrillo's article on communication and safety culture opens with an example:

> *"Bob…barely escaped with his life when an electrical arc burned the hair off his right arm and temporarily stunned him after he plugged in a piece of electrical equipment. No one came forward to admit they had failed to tag the electrical outlet as faulty because they knew it would mean days off without pay at best and dismissal at worst" [2].*

This scenario exemplifies symptoms of a safety culture that is more negative than positive, or as explained by Carrillo, is infected with "Safety Culture Toxins" [3].

One such toxin, the fear of reprimand, is a double-edged sword. As in the example here, it can prevent accountability and the acceptance of responsibility, both of which are necessary leadership qualities. Secondly, fear of reprimand can be felt when employees believe production is prioritized over safety. Carrillo further states: "Often employees [assumed] that it was more acceptable…to take a safety shortcut than to miss a deadline" [4]. When employees fear reprimand will result from being accountable for safety lapses, but at the same time are compelled or pressured to take shortcuts for fear of underperforming, leadership has failed to be consistent in promoting safety as a cornerstone of the organization's culture.

4.3 Promoting a safety culture

Because safety leadership is not limited to personnel with "safety" in their job titles and is a requisite of good safety culture, the concept refers more to a shared state of mind than a particular job duty. The collective responsibility for safe work makes everyone a safety leader. Reason suggests that if all personnel shared the responsibility to *ensure* safe work, fear of reprimand among the workforce, plus the perceived need to reprimand the workforce, would diminish because of greater self-accountability. At the same time, because of the fundamental top-down social dynamic, management and supervisors play a crucial role in ensuring that the shared responsibility for safety leadership has momentum and positive direction to promote positive culture change.

Traditionally, front-line workforces tend to be culturally isolated from their management principally because, as described earlier, they may not interact frequently. But research shows that front-line employees would prefer to see management visit their worksites with regularity, because management's visible presence demonstrates attention to the work environment and its workers, and by extension, their safety. Carillo reported that employees "said they cannot trust decisions made by managers who have never been to the job site, haven't demonstrated visible concern, competence or interest in learning about the real challenges workers face. They saw visibility as a symbol of the importance managers placed on safety" [5].

Although managers may claim time is too tight to spend any of it visiting the worksite, simply being visible to the workforce pays dividends that could counterbalance the number of issues that arise from miscommunications between groups that are often isolated from each other. Visible formal leadership helps diminish the cultural isolation between management and workers, and it engenders better communication. When communication is strong, trust follows.

Another common characteristic of a culture or subculture is its collection of values, and advocating shared values is an important responsibility of leaders and managers. A common toxin of safety cultures is a perception that production may sometimes be more important than safety, which, as described earlier, is a problem of communication. Dunlap explains that this occurs because safety is often viewed as a priority (the first step in undertaking a task, so to speak), thanks to empty expressions like "Safety

First" [6]. However, because job tasks can evolve and priorities can change, putting safety first may not always occur in a dynamic workplace. Instead, Dunlap's collective research [7] demonstrates that safety should not be thought of as a first priority; rather, it should be viewed as an immutable and core value throughout an organization, a value championed by management and espoused by employees. When safety is a core value, it is an intrinsic part of all tasks and in the forebrain of all personnel.

Effective communication is a large part of promoting a strong safety culture, but a good leader understands that communication is a two-way street—a good leader is even likely to *listen* more than speak. Jim Jensen's article from *Leadership Essentials* explains:

> *"In today's successful organizations who have transcended the old style management paradigm, there is an almost inverted pyramid where enlightened leaders understand the **value** and importance of 'working **for**' those who 'report to' **them**. They are constantly receiving input and feedback from their employees, so they can better understand the problems and issues one might be challenged with to be more productive in his work. The leader sees his/her role as helping to eliminate barriers and obstacles to heightened performance" [8].*

Active participation in the safety system also fosters accountability because employees are in control of their environment, the worksite. "This control then evolves into self-accountability. Employees take ongoing responsibility for their actions without the need for management to hold them accountable" [9].

Visible management, shared values, open communication, the input of employees, and a widespread sense of self-accountability can help clear a safety system of toxins and inspire positive and proactive safety management. Within skilled blue-collar industry, a positive safety culture involves positive workplace attitudes across all levels. Leaders have the greatest chance to affect the attitudes of employees who look to them for direction and clarity regarding expectations for their behavior. When employees are clear about their expectations, their attitudes about the organization and related safety culture are naturally more positive. Visible safety leaders provide training across all levels of the organization, and accountability exists at all levels as well.

Another characteristic of a positive safety culture involves mutual, meaningful, and measurable safety and health improvement goals. This relates again to employees' desire to have a clear understanding of the company's vision and expectations for their behavior. Additionally, when policies and procedures serve as reference tools, rather than obscure rules, leaders gain increased respect and credibility related to their attempt to enforce the policy. When employees feel as though policies exist to assist them in achieving safety and production goals, rather than as tools that management uses to punish behavior, the potential to achieve a positive safety culture is realized.

It is important to note that the goal of building and sustaining a positive safety culture does not happen overnight. Visible safety leadership, effective communication, and shared values are characteristics of a positive safety culture that is developed and requires leaders to be conscious of how employees perceive their actions.

4.4 Applying the push-pull concept

When all members of an organization play a role in management of the safety system, the fuel of its operation is the reciprocal pressure to meet safety goals. This pressure is applied both from employees to management and from management to employees.

When communication and cultural barriers are diminished as a result of more meaningful on-site interaction between employees and management, it can only benefit the circulation of and buy-in to safety as a core value. The alignment of values across an organization is crucial to the installation and upkeep of its safety culture, and visible safety leadership is a key component of this alignment. Carillo's work demonstrated that managers who "employees felt could be trusted even if the company could not" were those managers who took time to converse with employees and learn from their experiences in the worksite [10].

When employees feel as though company policies or procedures, whether related to safety or operations, are "pushed" on them, it is a natural human tendency to resist the pressure. Likewise, when employees feel as though supervisors or managers are "pulling" them in a particular direction, they experience the same natural tendency to resist. Instead, leaders are most influential and earn the trust and respect of their crews when they provide a clear path for employees to decide to follow on their own. Humans, just like animals, will respond to positive stimuli that provide a clear direction and reason for their involvement.

Visible leaders can respond to this natural human response to resisting the feelings of being "pushed" or "pulled" to buy-in to safety or any corporate attempt to affect change in the organizational culture. While it is no easy feat, visible leaders must work to provide the positive direction and reason for employees to buy-in to safety and a positive safety culture. This involves ensuring that the values of the organization are clear and responsibilities across all levels of the organization are defined. Of equal importance to visible leaders' behavior is ensuring that employees have the necessary resources to perform their jobs. Whether necessary resources involve money, time, manpower, supplies, or equipment, a primary function of visible leaders is to work *with* employees to identify what they need and then work to provide the necessary means for workers to meet expectations.

When employees perceive themselves to be empowered and responsible, rather than pushed or pulled, they will choose to work *with* leaders and management toward achieving organizational goals for safety and production. Visible safety leadership is a state of being, a developmental process that employees, supervisors, and management alike can commit to and promote across the organization.

4.5 Health and safety maturity model

Visible safety leadership is best recognized within organizations with a high level of health and safety maturity. One model of health and safety maturity that can be used as a benchmarking tool or roadmap is based on seven dimensions. In working toward safety and health maturity, it is important that organizations:

- Include leaders who demonstrate their *strong commitment*.
- Consider safety and health to be a *value* that is strategically important.
- Ensure components of a *safety management system* are visible in all processes.
- Establish *buy-in* to safety within the culture.
- Include respected *safety staff* who are a visible part of the leadership.
- Commit to *continuous learning* and development.
- Proactively seek to learn from and *advance technology* within/from industry.

Ranking an organization on these seven dimensions can be done to determine a total maturity assessment score. From this, action plans can be created based on gaps identified among the various components. The output from the total maturity score assessment, which ranges from "needs immediate action" to "maturity fully actualized," can be used to help organizations realize their potential either to improve or to celebrate the organizational behavior strengths contributing to a high level of safety and health maturity.

Benchmarking an organization's culture based on a health and safety maturity model can allow for a deeper understanding about the "way we do things around here" and provide a roadmap for making safety leaders visible. The applied skills and behaviors of visible safety leaders have the greatest chance to affect change in workers' behavior on the front-line, which is why leadership development and visibility is so important at the front-line leadership level.

The applied skills needed by leaders to develop desired behaviors they would like to see in others require a significant level of training and opportunities to continue learning. This concept relates to all members of an organization, regardless of their age, background, education, and so on.

4.6 A roadmap to develop visible safety leaders

Considering both dimensions of health and safety maturity in organizations as well as insights from what constitutes a positive safety culture, leaders can commit to behaviors that help them be more "visible" and effective in their organizations. The roadmap of four key behaviors shown in Fig. 4.1, can be thought of as a self-awareness reflection tool or a daily reminder of the values leaders could demonstrate through their behavior. Following this roadmap will allow leaders to be more visible and effective in their influence on others and on organizations as a whole.

Stop 1: *Self-awareness*—acknowledge one's individual strengths and personal challenges.
Stop 2: *Team-building*—utilize strengths—one's own and others'—to build effective teams.
Stop 3: *Effective communications*—work to ensure effective communication in all interactions.
Stop 4: *Organizational commitment*—demonstrate commitment to organizational values.

The first stop, developing self-awareness, is crucial for leaders, so they can clarify the type of people or leaders they aspire to be. By examining personal strengths and potential challenges associated with personalities, generational differences, personal learning styles, and past experiences, leaders can better understand their

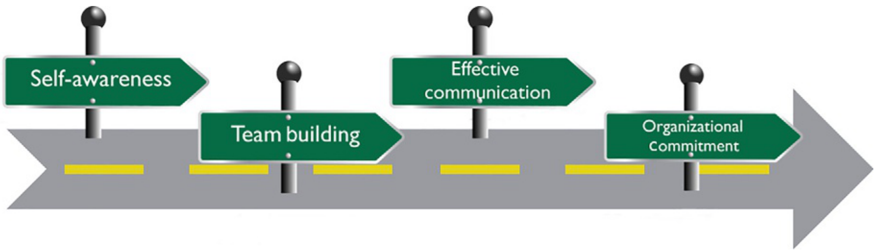

Fig. 4.1 Roadmap of four key behaviors that visible safety leaders demonstrate in influencing others.

potential as well as their possible limitations in influencing others. Self-awareness does not develop overnight and always requires thoughtful reflection. Such activities are not easy to engage in, and they take deliberate time and attention to prove useful. By committing the time necessary to learn and reflect on one's own dimensions, visible safety leadership can begin to develop.

A second stop along the roadmap to develop visible safety leadership involves transferring the same attention given to self-awareness in a deliberate attempt to understand others' personal strengths and possible challenges, then using this information to build effective teams. This is one of the applied skills on which leaders should focus in order to improve interpersonal interactions. While members of organizations often engage in work groups, these groups are not necessarily considered effective teams. Teams are recognized when the product or outcome of the teamwork is more significant than the sum of the inputs from individual members. One of the most important things that visible safety leaders can do to build effective teams is to understand the strengths and limitations of individual group members. With this knowledge, leaders can assign work more appropriately and ensure that team members understand their roles and feel valued and capable.

A third stop on the road map involves another applied leadership skill: effective communications. Visible safety leaders understand that the terms "information" and "communication," while often used interchangeably, are different. *Information* is the content of a message that is delivered through communicating, while *communication* is the process of getting information through to a recipient and verifying it has been received. A key distinction is that at various points in the communication process, a message (*information*) can be decoded (*received communication*) differently by the recipient than was intended by the source (*delivered communication*). Another distinction is that without proper feedback, the source or sender is unable to confirm if the receiver's decoding of the message aligned with how the sender encoded it. One way that leaders can ensure effective communications is to deliver information through appropriate channels, whether verbal or written, electronic or otherwise. Also, considering that much human communication is processed nonverbally, it is important that verbal messages and body language be clear and consistent. In addition, effective leaders should avoid messages that are too lengthy, disorganized, or contain errors. Oftentimes, leaders are served well by delivering information based on the insight that *less is more*.

A fourth stop on the leadership roadmap involves a commitment to organizational values and vision. Although leaders are responsible for enforcing policies and ensuring compliance to regulations, they are also most effective when they demonstrate the vision or values of the company. Front-line employees are constantly observing the behavior of leaders, no matter how "visible" those leaders are. In turn, employees are more likely to feel committed to the organization when they believe that their leaders are committed to the company. When employees feel committed to the organization, they perform better and take increased responsibility for their decisions and actions. When leaders visibly commit to the organization's values, they regularly communicate those values to employees, and they also live those values on a daily basis.

4.7 Summary

Although organizations are commonly structured in traditional top-down chains of command, the responsibility to ensure safe work is shared among everyone involved with a mining operation, from front-line workers and supervisors to upper management and even external stakeholders. When the responsibility to ensure safe work is shared, safety becomes more ingrained within an organization's culture. The resulting emergence of a strong safety culture pays dividends that directly influence an organization's bottom line—namely, its safety record, productivity, and opportunities for growth.

As part of a strong safety culture, visible safety leadership can transform safety-related issues and concerns into opportunities for communication and collaboration that can improve the safety system, rather than be occasions for mandated and immediate refresher training and reprimand, which do little, if anything, to improve the system. Safety leadership becomes more visible and therefore valuable when safe work is ensured through consistent safety-related communication, where safety is a core value of the organization and thus an intrinsic aspect of its culture.

A significant part of this leadership visibility is quite literal—front-line employees prefer to see their managers and supervisors regularly in the workplace, taking an interest in the worksite environment, and experiencing front-line job tasks first hand. The familiarity and rapport that results from these personal interactions and direct observations helps engender trust among the different levels of the organization's members. Further, showing interest in employees' first-hand experiences can help safety leadership improve the safety system in ways that realistically reflect what front-line employees see and do while performing their job tasks. When employees have an active role in fashioning and improving the safety system, adhering to safe work procedures becomes a matter of self-accountability rather than a matter of merely following the rules to avoid punishment.

Because it is human nature to resist being pushed or pulled toward any rule, policy, or idea aimed at influencing one's behavior, it is key that safety leaders empower front-line employees with the ability to affect the safety system within which they operate. This helps inspire reciprocal pressure between employees and their management to meet safety goals, not just a top-down pressure felt by employees to follow rules. Visible safety leadership often involves *listening* more than *speaking*. When the workforce provides valuable insight that can improve the safety system, management should feel pressured to incorporate that insight and improve the system, just as employees feel pressured to follow operating procedures and keep themselves and their coworkers safe from harm.

The level of efficacy at which a safety system operates is measured in terms of its *maturity*, so called because achieving a strong safety system with visible safety leadership at all levels is a slow and gradual process that involves the evolution of the organization's culture, and cultural change cannot happen quickly. The maturity of a safety system and culture can be benchmarked and assessed on seven dimensions: leadership's commitment, the view of safety and health as a core value, a visible and thorough safety management system, buy-in to safety within the culture, respected safety personnel, continuous learning, and advancing technology. The assessment of these seven essential aspects can uncover gaps between them that must be filled to further mature the model and thus improve the safety system.

As part of developing a mature health and safety model, leaders can enhance their visibility by keeping to a simple roadmap of values regarding their own self-awareness, effective team building, effective communications, and commitment to the values and vision of the organization. Enhanced visibility among the safety leadership helps transfer and aligns these values and the commitment to them among all personnel, thus encouraging active participation at all levels in creating safer, more productive, and more responsible mining operations.

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Optimization of coal recovery and production rate as a function of panel dimensions

5

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5.1 Introduction

It is estimated that at least 60% of coal production from underground mining comes from room-and-pillar (R&P) mines [1]. Despite many advantages that make it popular, one of the disadvantages of R&P mining is the low recovery that results from leaving pillars to support overlying material. Typical coal recovery in R&P mines ranges from 30% to 60%, with higher recovery possible only with pillar extraction. Geotechnical properties of coal and overlying material determine the size of pillars required to support the mine and, therefore, coal recovery. Design parameters such as panel width also affect recovery as they dictate how many barrier pillars will be required and their dimensions. Additionally, panel dimensions affect the production rate possible during mine operations.

Production panels are separated from each other by barrier pillars, which are designed to reduce the likelihood of progressive failures from adjacent workings. The width of barrier pillars separating two panels depends on the size of the panels and the integrity of pillars within the panels. However, barrier pillars, like pillars within a panel, reduce coal recovery. It is desirable then to reduce the amount of coal left in place in barrier pillars, if it is possible to do so without reducing safety. For panels containing the same size rooms and pillars, larger (wider) panels are likely to result in higher coal recovery because they result in fewer barrier pillars.

For most R&P mines, panel width is specified by the number of entries that are mined. The number of entries affects the production rate during mining operations in the panel. For example, mines with smaller panel widths (fewer entries) require fewer haulage units to minimize congestion and to ensure adequate ventilation. This can result in lower overall production rates despite faster advance rates associated with smaller panel widths. On the contrary, mines with larger panel widths (more entries) can use more haulage units because congestion and ventilation concerns are not as constraining. Also, more haulage units may be required to maintain production rates since haulage travel distances are longer. Otherwise, the utilization of the

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continuous miner is likely to be lower. However, such mines do not have to move production infrastructure as often since, once set up, production in a larger panel lasts longer. Research has shown that the relationship between panel width and production rate is not linear and engineers have to carefully consider the effect of panel width on the production rate [2]. The number of entries (panel width) must, therefore, be optimized to maximize production rates [1].

Consequently, while engineers may be motivated to use larger panel widths to increase coal recovery, there is a possibility that larger panels may lead to lower production rates. This leads to a dual-objective optimization problem where the objective is to maximize recovery and production rate. However, to model this optimization problem, the relationship between panel width and production rate must be established for each particular mine.

There is a push to increase production rates in R&P mines with newer technology and improved production practices. For example, the introduction of high-voltage continuous miners and automated haulers has increased loading rates, tram speeds, and payloads [3, 4]. Also, researchers have proposed approaches to improve production practices like optimizing cut sequences, haulage routes, equipment utilization, and equipment availability and minimizing roof bolt installation, change-out delays, and the width of main and submain panels [5–7].

Although the geometry of R&P mines would suggest that it is possible to optimize recovery by optimizing the layout of the series of panels and barrier pillars, the authors have not found any work in the literature that has addressed this issue. Improving coal recovery, even slightly, can have a significant effect on project economics. For instance, given a thermal coal price of US$41.00/ton, a 1% increase in recovery for a 10 million ton coal resource increases revenue by US$4.1 million over the mine life. These gains could be even higher if the objective function can also maximize production rate so that production rate is not unduly sacrificed in the quest for higher coal recovery.

In this chapter, an approach to optimizing coal recovery and production rate as a function of panel dimensions is presented. First, discrete-event simulation (DES) is used to model coal cutting and hauling operations in R&P mines in order to estimate production rates. Experiments are conducted using the model to determine the relationship between production rate and panel width. Second, a dual-objective optimization model is formulated that maximizes coal recovery and production rate. The model uses the relationship between panel width and coal recovery, which can be established for different mining conditions using the same model. The model is formulated as a cutting stock problem [8] and the optimization problem is solved using the integer programming solver in IBM's CPLEX, which is based on the branch-and-cut algorithm. A real-life case study is presented to illustrate how the relationship between production rate and panel width can be established using the DES model. A simple instance of the coal recovery optimization problem, which is formulated by incorporating the relationship established with the DES model, is then solved.

Section 5.2 of this chapter presents the DES model and the case study used to illustrate its usefulness. Section 5.3 presents the optimization model, the proposed solution formulation, and the case study. The final section presents conclusions and recommendations for future work.

5.2 Panel width and production rate

In this section, a discrete-event simulation approach to determine the relationship between panel width and production rate is described. A case study is used to illustrate the proposed approach, which is based on previous research [1].

5.2.1 DES of production rate vs. panel width

Discrete-event simulation is useful for evaluating what-if scenarios without expensive field experiments. It is also useful because it can estimate the performance of nonlinear and implicit systems and account for the stochastic nature of mining activities (such as hauling, loading, and dumping). In this section, DES is used to estimate the production rate of an R&P coal mining operation given the currently used panel width, equipment fleet, and cut sequence. The approach taken to study the coal cutting and hauling system is to

1. Build a valid DES model of coal loading, hauling, and dumping operations;
2. Determine the feasible range of input variables (panel widths, fleet size, and cut sequences);
3. Estimate production rates for all feasible values of input variables using the model.

5.2.1.1 Build a valid DES model

The general discrete-event modeling framework, which has been successfully used by many authors [1, 9, 10], includes the following:

1. Formulating the problem to be solved
2. Defining system and simulation specifications
3. Formulating and constructing the model
4. Verifying and validating the model

To study the relationship between production rate and panel width, this framework must be adapted for R&P mining. The analyst must understand system constraints, stakeholders' expectations, and the performance matrix needed to validate the model in order to clearly formulate model objectives. In this case, the simulation problem is to build a valid model capable of predicting the production rate for different panel widths using user-specified cut sequences and fleet sizes.

In order to construct such a model, it is essential to understand how the system operates and define its specifications. Thus, the loading, hauling, and dumping logic for the R&P mine must be defined to construct a model that predicts a production rate for given input variables. DES modeling requires the modeler to specify entities, resources, and processes of the system. To initiate the model, entities go through defined processes in a logical manner waiting for needed resources to become available at each process (i.e., resources are "busy" if they are being used by other entities) before they go through the process. Resources are static entities that provide services to other entities and processes. Typical resources include loading and dumping equipment; for example, the continuous miner (CM) and the feeder breaker (FB) in an R&P coal operation. Hauling equipment is usually defined as entities or transporters.

Once the model is constructed, the next important step is to ensure that it behaves as intended and accurately predicts the defined output. Model verification is usually done using animation to evaluate the behavior of the system. To validate the model, the simulation output (production rate) should be compared with real mine data to ensure that model predictions are within acceptable limits. The model can be used for further experimental analysis once validated.

5.2.1.2 Determine feasible range of input variables

The analyst determines the set of feasible scenarios by determining possible values of input variables given the existing mining constraints. A full factorial experimental design approach is then used to evaluate each combination of input variables in the feasible set. To understand the relationship between panel width and production rate, the primary input variable is panel width. However, the analyst must also specify the cut sequence and equipment fleet for each panel width since these input variables also affect the production rate.

The total number of experiments (scenarios) depends on system specifications, constraints, and stakeholders' expectations. For example, given the fleet size for an existing mine, if the analyst was to include relatively small panel widths, these will lead to long queues at the CM, which are likely to lead to exceedingly large cycle times. This makes it impractical to include relatively small panel widths in the feasible set as production rates are too low to be considered practical.

5.2.1.3 Estimate production rates

In this step, the analyst conducts simulation experiments for each of the scenarios identified in the previous step. The number of replications required for each scenario depends on the uncertainty associated with predictions of production rate in the particular instance. Often, the number of replications is determined based on the half-width (an estimate of the confidence interval assuming a normal distribution) [10]. Production rates can then be used in the optimization of recovery and production rates that is discussed in Section 5.3. First, a case study is described to illustrate the approach described in this section.

5.2.2 Case study

The case study used in this chapter is an underground R&P coal mine located in southern Illinois, the United States. The mine produces approximately 7 million tons of coal at a 54% panel recovery rate. The mine has experimented with multiple panel widths ranging from 11 to 21 entries. Each panel is a supersection mined with two CMs. Joy Model 14CM27 continuous miners are used in each section along with four 20 ton Joy Model BH20 battery-powered haulage units that transport coal from the cutting face to the FB. The CM cuts and loads coal at up to 40 ton/min with a maximum cutting height of 11.2 ft. Full panel width is mined in six-crosscut increments with the FB moved in three-crosscut increments. The FB is located at the center of each production panel to transfer mined coal from haulage units to the panel conveyor belt. The mine operates on three 8 h shifts—two for production and one for maintenance. The width of the panel is mined using two distinct cut sequences. First, the central 11 or 13 entries are advanced six crosscuts in the direction of mining. Second, the remaining entries on the flanks perpendicular to the directions of mining are mined as "rooms." Figs. 5.1 and 5.2 show examples of an 11-entry cut sequence for the central entries and a 15-entry cut sequence for two rooms on either flank of the 11 central entries.

5.2.2.1 Build a valid DES model

The objective was to build a model capable of predicting the production rate for different panel widths using user-specified cut sequences and fleet sizes. In this case study, researchers working with mine engineers decided that the model will be

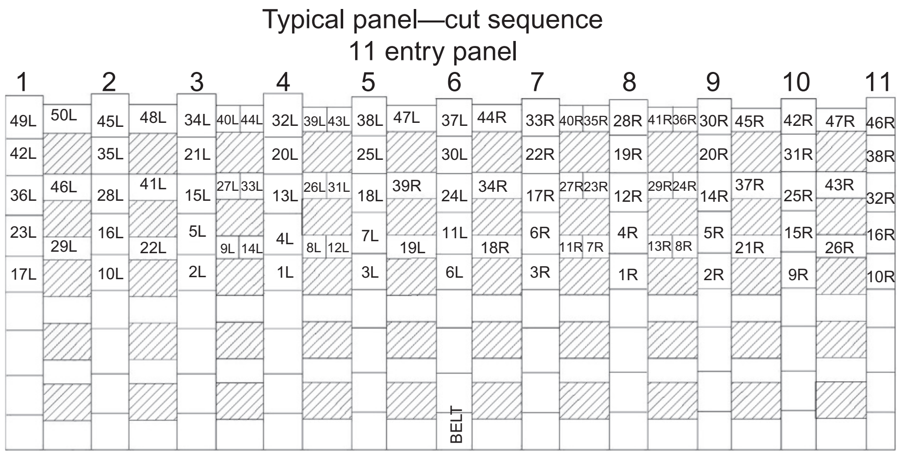


Fig. 5.1 Cut sequence for 11-entry initial advance.

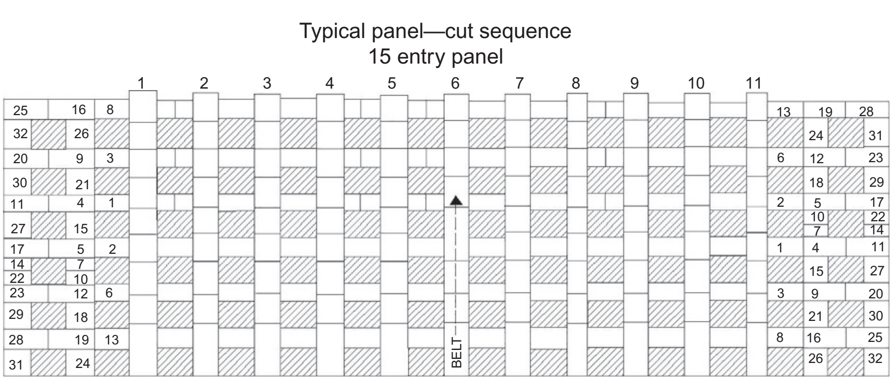


Fig. 5.2 Room cut sequence for two rooms on either side of an 11-entry initial advance.

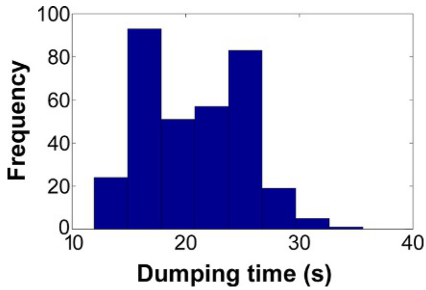


Fig. 5.3 Haulage unit dumping time.

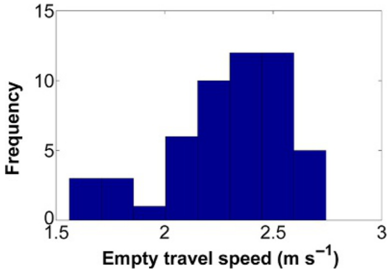


Fig. 5.4 Empty haulage unit travel speed.

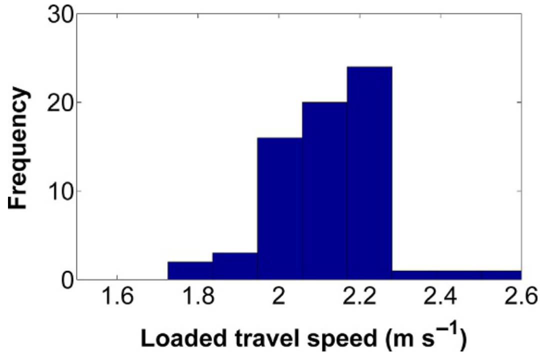


Fig. 5.5 Loaded haulage unit travel speed.

deemed valid if the predicted output is within 15% of actual levels achieved by the mine. Input data used to validate the model was obtained from time studies conducted at the mine (Figs. 5.3–5.8). These raw data were analyzed to fit statistical distributions using the chi-squared goodness-of-fit test, as shown in Table 5.1. Input data included loading and dumping times, payloads, and battery change data, which are sampled from these distributions. Input data was selected to reflect loading, hauling, and dumping logic. Besides production rate, other model outputs include production

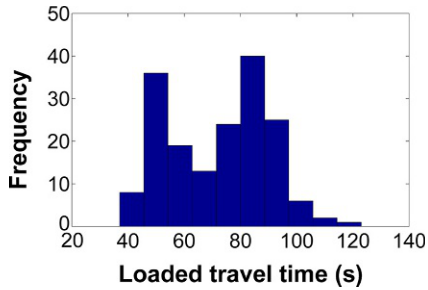


Fig. 5.6 Loaded haulage unit travel time.

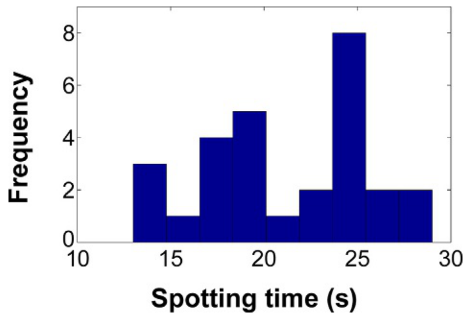


Fig. 5.7 Haulage unit spotting time.

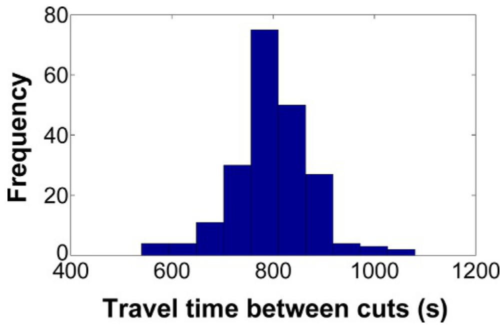


Fig. 5.8 CM travel time between cuts.

per shift and total operating costs. The model was built using Arena simulation software based on the SIMAN language.

The model logic conceptualizes loads of coal as entities with specific attributes (entity number, payload, and cut sequence—the cut sequence is assigned to each entity to ensure the information is available to "route" loads to the active cut). Battery-powered haulage units were modeled as guided transporters (an Arena-specific modeling construct) used for hauling loads (entities). Transporters use entries and crosscuts as haulage routes, which are modeled to restrict traffic flow such that

Table 5.1 DES input data

Data(s)	Distribution(s)	*P*-value
Payload (kg)	10,886	N/A
Empty speed (ms^{-1})	1.86 + GAMM(0.0987, 5.05)	<0.005
Loading time (s)	28 + ERLA(3.63, 3)	<0.005
Dumping time (s)	6 + GAMM(2.79, 5.36)	<0.005
Battery change (s)	TRIA(5,7,10)	<0.005
Loaded speed (ms^{-1})	1.77 + GAMM(0.0546, 7.26)	<0.005
Time between cuts (s)	NORM(797, 87.7)	<0.005
Spotting time (s)	12.5 + GAMM(4.22, 2.11)	<0.005

any point on a haulage route can only accommodate one haulage unit at a time since mine openings are not wide enough for them to pass each other. The CM is modeled as a resource used for the loading process and can only load one haulage unit at a time. The FB is also modeled as a stationary resource used for dumping loads (entities). The FB and each cutting face are modeled as stations, which are points in the model where transporters transfer entities. Haulage routes between stations are modeled as network links to capture varying haulage distances. Distances for each network link are inputs to the model. Fig. 5.9 shows the logic used to model the system.

An animation of the system that includes transporters, stations, network links, resources, and entities was used to ensure that the model behaves as intended. For example, the animation was used to verify whether the model follows the defined cut sequence. The model was validated using shift production data from the mine. The validation is achieved by comparing the simulated output (coal production rate and shift duration) with data from time-and-motion studies conducted at the mine. During the experiment, 150 replications were conducted to obtain estimates of load count, total coal production, and mining duration. That number of replications was selected because it ensures that the half-width of the mining duration (the most uncertain output) is less than 1% of the estimated duration.

The mining scenario used for validation was a 13-entry panel. Time study data collected for two 8 h shifts showed average payload per haulage unit was 12 tons. The mine reported production of 2448 tons of coal produced from 11 cuts with a total of 204 coal loads per shift. During one 8 h shift, only 6.33 h were spent mining, with the remaining time spent on CM and conveyor belt repairs. Table 5.2 shows results of the validation. The simulated output was within 15% of actual values. The model was, therefore, deemed valid and used for further experimental analysis.

5.2.2.2 Determine feasible range of input variables

In this case study, the main input variable is panel width since the objective is to determine the relationship between panel width and production rate. As stated before, the mine has experimented with 11 and 13 entries of initial advance before expanding into

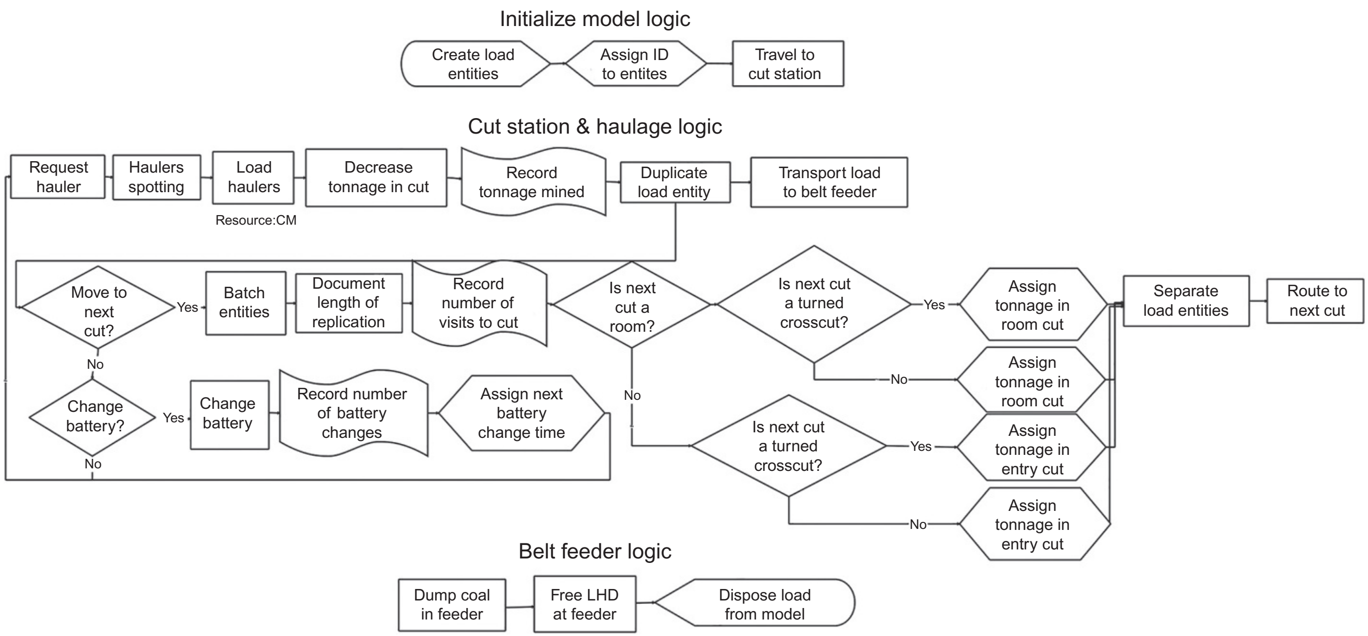


Fig. 5.9 DES model logic.

Table 5.2 **Results of DES validation experiment**

Parameter	Actual	Simulated	Difference
Duration of mining (h)	6.33	6.83	8%
Production (tons)	2448	2748	12%
Number of haulage units loads	204	226	11%
Half-width of duration (h)	N/A	0.012	N/A

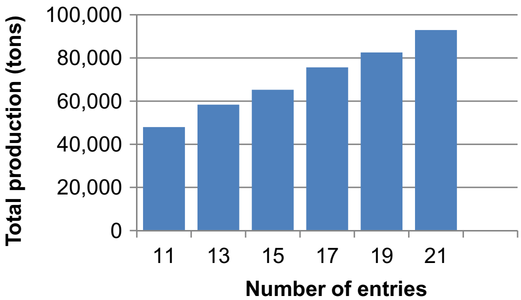
rooms if necessary. This means that for panel widths greater than 13 entries, there are two cut sequences: one where the central 11 or 13 entries are mined first before rooms are mined. Once the initial advance is mined, the mine has mined anywhere from zero to five additional rooms on each side depending on the designed width of the panel. For this case study, however, only cut sequences where the central 11 entries are mined first before rooms are mined were considered to generate the relationship between production rate and panel width. This is because previous research at the same mine indicated that cut sequences with 11 central entries lead to higher production rates than those with 13 central entries [1].

In this previous research, it was determined that assigning four haulage units to the CM is optimal for panel widths being considered and for operating conditions at the mine [1]. Hence, four haulage units are assigned to the CM in presenting the case study in this chapter.

5.2.2.3 *Estimate production rates*

Figs. 5.10–5.16 show simulation results describing the effect of panel width (number of entries) on production rate. Figs. 5.10 and 5.11 show that total production and duration of mining increase with increasing number of entries. This is to be expected if the model is performing well. Figs. 5.12 and 5.13 show that the percentage of production time the CM spends loading haulage units initially increases with increasing panel width until an *optimal* (with respect to production rate) panel width is reached at 17 entries. This indicates that there is excess haulage unit capacity in the system with less than 17 entries. CM operations are inefficient due to the excessive

Fig. 5.10 Total production.



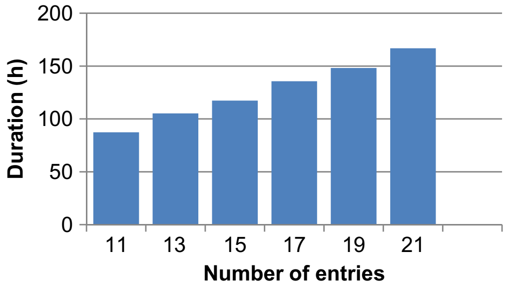


Fig. 5.11 Duration of mining.

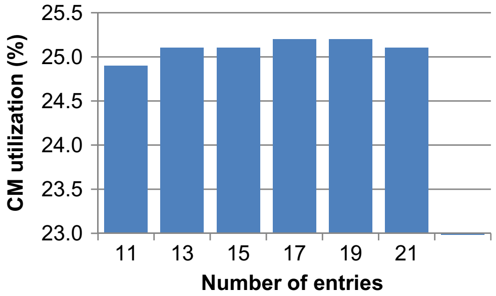


Fig. 5.12 CM time spent loading (LHS).

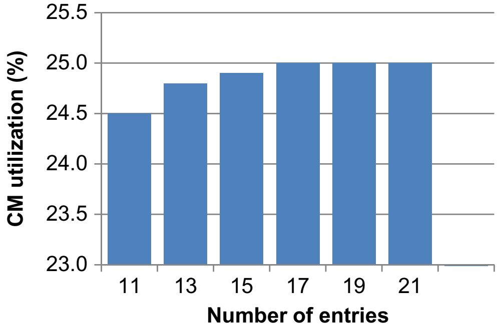


Fig. 5.13 CM time spent loading (RHS).

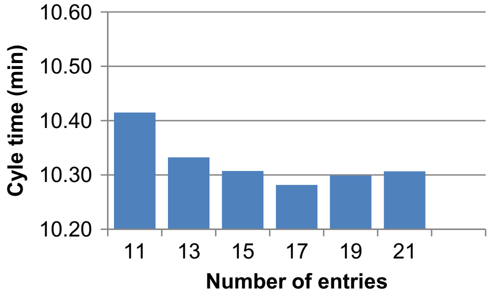


Fig. 5.14 Average cycle times (LHS).

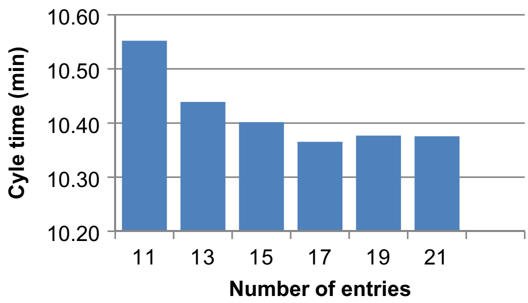


Fig. 5.15 Average cycle times (RHS).

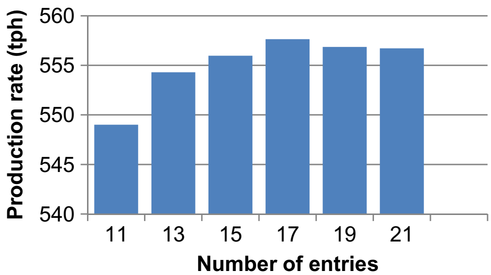


Fig. 5.16 Production rate.

spotting time resulting in long wait times and bunching; however, expanding panel width beyond 17 entries results in inadequate haulage unit capacity and underutilization of the CM. This is confirmed by Figs. 5.14 and 5.16, which show that the haulage unit cycle time increases when the panel width exceeds 17 entries. Initial expansion of the panel reduces the haulage unit cycle time (minimizes waiting time). However, further expansion of the panel increases haulage unit cycle times because haul distances become longer, leading to an operation constrained by haulage unit capacity. Adding more haulage units will increase production and CM utilization but will also increase the unit cost of operation. These trends (cycle time and CM loading times) directly result in the observed trend in production rate (Fig. 5.16), with a panel width of 17 entries generating the maximum production rate.

Fig. 5.16 provides the information needed to generate production rate indexes, which can be used in the optimization model to account for the relationship between production rate and panel width.

5.3 Maximizing recovery as a cutting stock problem

5.3.1 Modeling coal recovery as a cutting stock problem

The cutting stock problem (CSP) is the problem faced by someone who seeks to cut smaller pieces of material, given the customer demand, from a larger piece of stock material in such a way as to minimize waste [8, 11]. The problem is one-dimensional if cuts differ in only one dimension (width or length). Higher dimensional problems exist depending on the nature of the problem. There are many variations of the cutting stock problem with different objectives.

The one-dimensional problem is the most common and exists in multiple industries including fiber, paper, steel, timber, and aluminum industries. In this problem, a roll of material needs to be cut into pieces of different lengths (but the same width, which is the same as the width of the roll) that minimize the trim loss (what is left over after cutting).

Assuming that the width of paper rolls is $W > 0$ and the customer i ($i = 1, 2, \ldots, m$) wants b_i cuts of width $w_i \leq W$ (orders for cuts of width greater than W cannot be fulfilled), the maximum number of rolls needed, K, can be estimated by considering that in the worst case, each cut is from one roll (i.e., $K = \sum_{i=1}^{m} b_i$). The one-dimensional cutting stock problem can then be formulated as Eq. (5.1):

$$
\begin{aligned}
& \min \sum_{k=1}^{K} x_0^k \\
& \sum_{k=1}^{K} x_i^k \geq b_i \quad \forall i \\
& \sum_{i=1}^{m} w_i x_i^k \leq W x_0^k \quad \forall k \\
& x_0^k \in \{0, 1\} \quad \forall k \\
& x_i^k \text{ is a positive integer}
\end{aligned}
\tag{5.1}
$$

where $x_0^k = \begin{cases} 0, & \text{if roll } k \text{ is not used} \\ 1, & \text{if roll } k \text{ is used} \end{cases}$ and x_i^k is the number of cuts of width w_i from roll k.

For large numbers of cuts and stock rolls, the size of the linear programming problem becomes prohibitively large. Consequently, solutions to the cutting stock problem are challenging if all basic solutions are examined. Hence, most research on cutting stock problems has focused on how to solve problems without examining all possible solutions.

The authors posit that the optimization of coal recovery based on panel widths is a one-dimensional cutting stock problem. Each strip of the underground coal mine, which is made up of a series of panels separated by barrier pillars, can be considered a stock roll. Hence, the whole mine is made up of K stock rolls of differing widths, W_k ($k = 1, 2, \ldots, K$). Each panel can be w_i wide, where $i = 1, 2, \ldots, m$ for the m panel widths under consideration. Unlike the conventional cutting stock problem, the coal recovery problem does not have specific demands for how many times a particular panel width is used in the design. Also, the coal recovery problem requires that all strips be mined. This is analogous to having all available rolls used in the cutting stock problem. Hence, there is no need for the variable x_0^k, which defines whether a roll is used or not.

Therefore, the coal recovery problem can be modeled with Eq. (5.2) where η is a ratio used to specify the relative importance of recovery and production rate, w_i is the width of the panel from barrier pillar center to center, ρ_i is the width of the barrier pillar designed for panel width i, and π_i is a production rate index for panel width w_i, which is proportional to the estimated production rate for the particular panel width. It is also useful to ensure a production rate index of similar magnitude to panel width. The decision variable x_i^k is the number of panels of width w_i used in strip k. The objective is to maximize recovery by maximizing the overall width of coal that is mined ($\sum_{k=1}^{K}\sum_{i=1}^{m}(w_i-\rho_i)x_i^k$) and overall production rate ($\sum_{k=1}^{K}\sum_{i=1}^{m}\pi_i x_i^k$):

$$\begin{aligned}&\max \sum_{k=1}^{K}\sum_{i=1}^{m}\{\eta(w_i-\rho_i)+\pi_i\}x_i^k\\&\sum_{i=1}^{m} w_i x_i^k \le W_k \quad \forall k\\&x_i^k \text{ is a nonnegative integer}\end{aligned} \tag{5.2}$$

For this formulation to work, the user must carefully select values of η and π_i. The user can specify π_i as the simulated production rate from the DES model or an index that is proportional to the production rate. It is important that values of π_i are distinct enough that the solution discriminates between various panel widths under consideration. For example, a linear scale from 0 to 100 can be used to define values based on production rates (Eq. 5.3). Variables r_i, $r_{\min}$, and $r_{\max}$ are the production rate for panel width i, the minimum production rate, and the maximum production rate in the given set of panel widths, respectively:

$$\pi_i = \frac{r_i - r_{\min}}{r_{\max} - r_{\min}} \times 100 \tag{5.3}$$

It is also important to ensure that η is chosen to reflect the relative importance of recovery and production rate to the decision. If management desires for production rate and recovery to be equally important in the decision, then η should be chosen to ensure that ηw_i is of the same order of magnitude as π_i. Otherwise, the user should specify η such that $\eta w_i \gg \pi_i$ or vice versa.

5.3.2 Solution formulation

Eq. (5.2) is an integer optimization problem that can be solved with a variety of solution algorithms including branch and cut and branch and bound [12]. In the case study described in the next section, an illustrative example of the problem is solved using CPLEX together with its MATLAB application program interface (API). This allows preparation of the problem in MATLAB, which is then passed to CPLEX. CPLEX solves the problem and returns the solution to the MATLAB environment where results can be postprocessed into useful input for mine design. The CPLEX integer optimization solver is used, which is based on a branch-and-cut algorithm.

The CPLEX integer optimization solver solves problems like Eq. (5.4). Thus, to solve a problem using the solver, one needs to convert the problem input into vectors and matrices that correspond to Eq. (5.4). In this case, MATLAB functions were written to take input (e.g., number of strips, number of panel widths under consideration and their widths, and production rates for each panel width from DES) and convert the model (Eq. 5.2) into equivalent matrices and vectors in Eq. (5.4). This approach is adequate for small problems where the number of strips is few and the number of possible cutting patterns is not prohibitively high:

$$\begin{aligned}
&\min \mathbf{c}^T\mathbf{x}\\
&\quad \mathbf{A}\mathbf{x} \le \mathbf{b}\\
&\quad \mathbf{A}_{eq}\mathbf{x} = \mathbf{b}_{eq}\\
&\quad \mathbf{l} \le \mathbf{x} \le \mathbf{u}\\
&\quad \mathbf{x} \text{ is an integer}
\end{aligned} \tag{5.4}$$

In the case study problem where there are many strips and panel widths are small relative to the width of strips (i.e., there are many possible patterns to consider), the literature shows that the solution approach used will be computationally expensive [11, 13]. In such instances, the branch-and-price algorithm, which only incorporates a few patterns at a time and generates additional columns as the solution progresses, has been shown to be more promising [14]. Future work should focus on developing branch-and-price algorithms to solve the coal recovery problem. This will require finding means to perform the Dantzig-Wolfe decomposition of the linear programming relaxation of the problem. If successful, this line of research will ensure that realistic instances of the problem (Eq. 5.2) with many strips can be solved in reasonable time making this approach much more useful for mine planning and design.

5.3.3 Case study

To illustrate the coal recovery optimization problem, a case study with 10 strips is used. Table 5.3 shows strip widths, panel widths, and production indexes. The situation where all six panel widths in the simulation experiments are considered to be feasible is taken into account. This may not necessarily be the case in a real application. Engineers and managers may want to limit the number of panel widths used in the mine so that production practices do not vary significantly. This will actually make the problem easier to solve as there will be fewer feasible patterns. In this case study, it was assumed that all barrier pillars are 80 ft. wide regardless of panel width. The problem is solved for $\eta = 0.05$, 0.10, 0.15, 0.30, 1.00.

Table 5.4 shows optimization results for the case study. These solutions show that the model works as intended and can be useful for mine engineers and managers during R&P mine planning.

Based on the sensitivity of the result to η, it can be concluded that the model truly behaves as a dual-objective optimization model. The reader can observe that for $\eta \le 0.1$, the 17-entry wide panel, which is the panel width with the highest production rate, is the preferred panel width. It dominates the solution and is to be used on every strip and is used exclusively to mine strips 9 and 10. These solutions obviously prioritize production rate and only use other panel widths when one with a higher

Table 5.3 **Optimization input data (A) panel width parameters and (B) strip lengths**

Panel width (no. of entries)	**Panel width (ft)**	**Production rate index**
(A)		
11	620	0.00
13	740	61.30
15	860	80.65
17	980	100.00
19	1100	90.96
21	1220	89.22

Strip number	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
(B)										
Length (ft)	2990	3970	4950	5930	6910	7890	8870	9850	10,830	11,810

Table 5.4 Optimization results

η	Panel width (no. of entries)	Strip number 1	2	3	4	5	6	7	8	9	10
0.05	11	0	0	0	0	0	0	0	0	0	0
	13	1	0	0	2	0	0	0	0	0	0
	15	0	3	3	0	5	5	0	0	0	0
	17	2	1	2	4	2	3	7	8	10	11
	19	0	0	0	0	0	0	0	0	0	0
	21	0	0	0	0	0	0	1	1	0	0
0.10	11	0	0	0	0	0	0	0	0	0	0
	13	1	1	1	2	1	0	0	0	0	0
	15	0	1	1	0	3	5	0	0	0	0
	17	2	2	3	4	3	3	6	8	9	10
	19	0	0	0	0	0	0	1	0	1	1
	21	0	0	0	0	0	0	1	1	0	0
0.15	11	0	0	0	0	0	0	0	0	0	0
	13	1	1	1	2	0	1	0	0	0	0
	15	0	1	1	0	0	3	0	0	0	0
	17	2	2	3	4	4	4	6	8	9	10
	19	0	0	0	0	0	0	1	0	1	1
	21	0	0	0	0	2	0	1	1	0	0
0.30	11	0	0	0	0	0	0	0	0	0	0
	13	1	1	1	0	0	0	0	0	0	0
	15	0	1	1	0	0	5	0	0	0	0
	17	2	2	3	2	4	3	6	8	9	10
	19	0	0	0	1	0	0	1	0	1	1
	21	0	0	0	2	2	0	1	1	0	0
1.00	11	0	0	0	0	0	0	0	0	0	0
	13	1	0	1	0	0	0	0	0	0	0
	15	0	0	1	0	0	0	0	0	0	0
	17	2	0	3	2	4	0	1	2	4	5
	19	0	0	0	1	0	0	0	1	0	0
	21	0	3	0	2	2	6	6	5	5	5

production rate is not feasible. On the contrary, for $\eta > 0.1$, there is more diversity in panel widths selected, and solutions maximize coal recovery, but do not prioritize production rate. For example, consider strip 2, which is 3970 ft wide. When $\eta < 1$, the solution requires the strip to be mined with four panels as opposed to three panels when $\eta = 1$. Given that all panels have the same barrier pillar width, mining four panels leaves 240 ft of coal in barrier pillars compared with 180 ft of coal in barrier pillars when mining three panels. In addition, the solution when $\eta = 1$ leaves out 310 ft of coal at the end of the strip compared with 410 ft in all other solutions. Note that coal left at the end of a strip will likely be mined, making the last panel wider than the recommended width. However, even then, the practice will be suboptimal as it will lead to mining a wider panel than anticipated. Thus, solutions that leave very little coal are to be preferred, which is how the model behaves.

It can also be observed that solutions mine strips with more differing panel widths when recovery is the predominant factor (i.e., relatively higher values of η). These solutions deemphasize production rate twice. First, these solutions tend to use lower producing panels more, which will lower production rates during mining. Second, the production rate during mining will be even lower because the mine has to switch panel sizes frequently, which negates efficiency gains from repetition. The ability to recognize this tendency for lower production rate mine plans when production rate is completely removed from consideration is a benefit of the proposed dual optimization approach.

5.4 Conclusions and recommendations

This chapter presented an approach to optimize coal recovery and production rate as a function of panel dimensions. Discrete-event simulation (DES) is first used to establish the relationship between production rate and panel width. An optimization model is then formulated that maximizes coal recovery and production rate. The coal recovery problem was shown to be similar to the cutting stock problem and modeled by adapting the cutting stock problem. The model used production rate indexes for panel widths derived from DES results. The optimization problem is solved using CPLEX's integer programming solver, which is based on the branch-and-cut algorithm. Time study data from an underground room-and-pillar coal mine in Southern Illinois, the United States, are presented to illustrate how to determine the relationship between production rate and panel width using the DES model. Having used those production rates to determine production indexes, an instance of the optimization problem involving 10 mining strips is solved as a case study. Optimal panel widths used in such strips are examined when considering all six panel widths from the same room-and-pillar coal mine modeled in the DES case study. The case study results show that a dual optimization approach that maximizes recovery and production rate is beneficial because, in addition to accounting for management's dual objectives, it leads to solutions with more consistent mine plans (i.e., fewer panel widths are used in the mine plan). Also, results show that an analyst should carefully choose the ratio in the model that specifies the relative importance of recovery and production rate to obtain results that are most useful.

The authors recommend that future research explore how to solve the optimization problem using the branch-and-price algorithm. This will overcome the limitation of the current branch-and-cut algorithm, which is computationally expensive when there are many strips.

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Mine ventilation networks optimized for safety and productivity

6

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6.1 Introduction

The purpose of mine ventilation is to provide sufficient quantities of fresh air to all miners and to mining equipment and processes that require fresh air, including the operation of diesel or other equipment with internal combustion engines. Further, ventilation must dilute and render harmless any toxic, explosive, asphyxiating, radioactive, or otherwise harmful gases and dusts.

A mine ventilation network is similar to a network of pipelines or electrical connections that reaches every working area of the mine and provides a specific amount of air flow to each workplace. In order to distribute appropriate amounts of fresh air to each workplace and to exhaust used air back to the mine portal or air shaft, various ventilation controls are used, including:

- Mechanical mine fans that generate fresh air flow for the entire mine (main fan) or a large area of it (bleeder fan). In most cases, exhausting fans are used because they are usually simpler to install. It is common for large mines to have more than one main fan or installations of multiple fans in parallel.
- Regulators that restrict and control the amount of air fed to specific areas of the mine.
- Stoppings that prevent air flow through certain mine workings.
- Curtains—both line curtain and pull-through curtain.
- Seals that permanently block ventilation air from entering mined-out areas where it is no longer needed, thus stopping ventilation in those areas.
- Auxiliary fans that increase air flow in face areas, dead-end drifts, or other areas that require increased ventilation. In underground coal or gassy noncoal mines, the use of auxiliary fans is restricted to the immediate face areas of continuous miner development, while in nongassy, metal and nonmetal mines, auxiliary fans are widely used to ventilate dead-ended drifts.
- Underground booster fans may be used in nongassy, metal and nonmetal mines to improve air flow quantities in larger areas of the mine where the main mine fan no longer produces sufficient ventilation.

Some auxiliary fans and regulators may be controlled remotely or even automatically controlled. This is often done in nongassy, metal and nonmetal mines that employ control systems to provide the required local and face ventilation on-demand (VOD).

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Air flow through mine ventilation networks follows a quadratic mathematical relationship similar to Ohm's law in electricity:

$$H = R^* Q^2 \tag{6.1}$$

where

H = pressure difference between two points in the network. Units are Pa or inches water gauge (in-WG). It is analogous to voltage in electrical networks.
R = airflow resistance. It is a function of the number and size of the air ways. Units are $\mathrm{N\,s^2\,m^{-8}}$ or $\mathrm{in\text{-}WG\,min^2\,ft^{-6}}$. It is analogous to resistance in electrical networks.
Q = airflow quantity. Units are $\mathrm{m^3/s}$ or cubic feet per minute (CFM). It is analogous to current in electrical networks.

Similar to electric current, ventilation flow and pressure also follow the basic laws of all networks known as Kirchhoff's laws. Kirchhoff's first law provides for the conservation of mass. At any junction or node connecting two or more airway branches or paths within the network, the sum of mass flows of air going into the junction must be equal to that going out. Kirchhoff's second law requires that, in any closed loop through a ventilation network, the sum of all pressure differentials across each branch is zero.

A large variety of computer programs exist to calculate air flows and pressures in mine ventilation networks. These programs use iterative methods, which are necessary due to the quadratic relationship in Eq. (6.1). A well-known and frequently used iteration method is the Hardy Cross Method, but with modern computers, other mathematical iteration methods may be used as well. Some computer programs can simulate dust and contaminant flow in mine airways, while others can be used to calculate air temperatures and cooling or heating requirements. Some programs can also assess the influence of a mine fire on ventilation.

Airflow resistance, R, can be calculated using the following equations:

$$\text{For SI units, } R = k^* L^* O/A^3 \tag{6.2}$$

$$\text{For Imperial units, } R = k^* L^* O/\left(5.2A^3\right) \tag{6.3}$$

where

k = airway resistance coefficient or k-factor. Units are $\mathrm{Ns^2/m^4}$ ($\mathrm{kg/m^3}$) or $\mathrm{in\text{-}lb^*min^2/ft^4}$.
L = airway length, including the equivalent length added for shock losses. Units are m or ft.
O = perimeter of the airway cross section. Units are m or ft.
A = cross-sectional area of the airway. Units are $\mathrm{m^2}$ or $\mathrm{ft^2}$.

The k-factor conversion between unit systems is $1\,\mathrm{kg/m^3} = 1\ \mathrm{Ns^2/m^4} = 5.4 \times 10^{-7}\,\mathrm{lb.\,min^2/ft^4}$. The k-factor typically ranges between 3 and $40 \times 10^{-3}\,\mathrm{kg/m^3}$ (20 and $200 \times 10^{-10}\,\mathrm{lb.\,min^2/ft^4}$). Smooth, straight, circular airways tend to have k-factors at the lower end, while curvy, irregular, and obstructed airways have higher k-factors. For more information and a representative table of typical k-factors in Imperial units, see United States (US) Bureau of Mines Bulletin 589 [1].

6.2 Mine ventilation network design and planning

Whether for coal or metal and nonmetal mines, mine ventilation networks must be carefully planned and laid out in conjunction with mine design and production planning. Mine ventilation demands change significantly over the life of the mine. A mine ventilation system can be expanded over the life of the mine by adding air supply and exhaust capacity by means of additional shafts, drifts, and fans. Conversely, mined-out areas should be sealed as soon as they no longer require ventilation. Between major expansions or consolidations, the ventilation system must be designed to supply sufficient airflow capacities to all areas of the mine. The ventilation engineer must work in close cooperation with both long-term and short-term planners to understand exact ventilation demands at any time during the mining process. This includes knowing the individual ventilation demands of all pieces of mining equipment and sizing the mine openings to match these demands without generating undue airflow resistance. If Eqs. (6.2) or (6.3) are applied to circular cross-section airways, one can easily see that the resistance is proportional to the 5th power of the inverse diameter. Increasing the airway size by 15% reduces resistance by 50%, and increasing its size by 25% will cut resistance by 67%.

A mine planning engineer will lay out drift dimensions primarily based on the size of the equipment that will be used. Increasing airway size beyond this minimum often conflicts with ground control demands. If airways cannot be increased in cross section, ventilation engineers should consider adding one or more parallel airways. Two identical airways in parallel will cut resistance by 75%, while three airways will cut it by 87%.

When designing air shafts, the diameter is also an important consideration. The minimum diameter is sometimes determined by hoisting or equipment size requirements. The ventilation engineer should weigh carefully whether it is worth increasing the diameter in order to reduce resistance, thereby saving on ventilation power costs. This is usually a simple economic consideration, and many ventilation planning programs offer ways to calculate such power savings.

For planning purposes, mine airway resistances and fan power requirements can be calculated from initial assumptions of airway geometry and resistance. As the mine is developed, air quantity and pressure surveys should be conducted every 6 months to establish an accurate mine ventilation model using a computer network calculation program. A valid computer model can then be used to project ventilation requirements for future stages of mine development and to assist the mine planning engineer by establishing required airway geometries, new air shafts requirements, and future fan specifications.

6.3 Mine air quality and dust monitoring

US mine regulations are published in the Code of Federal Regulations (CFR) [2]. For underground coal mines, 30 CFR §75.321 requires that mine air contain a minimum of 19.5% oxygen (O_2) and less than 0.5% carbon dioxide (CO_2). Similar standards exist for metal and nonmetal mines and most other countries have similar requirements. Limits in the parts per million (ppm) range are established for most toxic gases, including carbon monoxide (CO), nitrous oxides (NO and NO_2, which are both commonly referred to as NO_x), hydrogen sulfide (H_2S), and sulfuric oxides (SO_2 and SO_3). In the US, limits for these gases are typically established by the National Institute for Occupational Safety and Health (NIOSH) and the American Council of Government Industrial Hygienists (ACGIH). Many countries have their own standards, while others accept and incorporate US or European limits.

For flammable and explosive gases, such as methane (CH_4), the limit is typically 1.0% or about 20% of the lower explosibility limit (LEL). Some countries permit a limit at 1.5% CH_4. In the US, exemptions may be granted up to 1.5% in certain return airways per 30 CFR §75.321(d), if additional precautions, including a minimum airflow quantity of 12.7 m^3/s (27,000 cfm) and atmospheric monitoring systems (AMS) with audible and visible alarms, are in place. In bleeder airways, up to 2.0% CH_4 is permitted per 30 CFR §75.321(e).

The average respirable mine dust concentration in US coal mines must not exceed 1.5 mg/m^3 (per 30 CFR §70.100, as of August 1, 2016). Dust concentrations beyond 60 m (200 ft) outby the working face are limited to 0.5 mg/m^3. Stricter requirements are provided if the mine dust contains quartz. Monitoring for dust, diesel particulate matter (DPM), and exposure to radioactive substances is conducted at regular intervals both by the mine operator and by government inspectors. For DPM, the personal exposure limit (PEL) for US metal and nonmetal mines is 160 $\mu g/m^3$ total carbon (TC) per 30 CFR §57.5060, while an equivalent rule does not exist for coal mines.

Quantity and quality of mine air are typically monitored by ventilation engineers and technicians, mine foremen, equipment operators, mechanics, and certified mine examiners. Unless preparing for special work, mine examiners test air quality and quantity at regular intervals ranging, in the US, between 20 min in face areas to daily or weekly in more remote airways. Unlike other countries, the US has no general requirement for installing AMS or other methods to continuously monitor air quality or quantity in key areas of mines. Technologies exist for such, from online air quality monitoring for single or multiple gases, air quantities, and oxygen deficiency; to simple, passive gas sampling tube bundle systems that continuously draw air quality samples from multiple locations throughout the mine to be analyzed with online apparatus at the surface.

6.4 Ventilation of continuous miner faces and sections in coal mines

In the US, underground coal mining is frequently carried out using the room-and-pillar mining method where coal is cut and loaded with a continuous miner. Longwall gate road development in US and Australian coal mines is also done with continuous miners, cutting two to four parallel roadways separated by pillars. European mines use road headers cutting single-entry gate roads with large, arched cross sections and standing support while continuous miner headings are primarily supported with roof bolts.

US coal mining regulations require a minimum air quantity of 1.4 m^3/s (3000 cfm) in each heading; 4.2 m^3/s (9000 cfm) in each heading while active mining is going on; and 14 m^3/s (30,000 cfm) at each longwall face. In most practical mining applications, these minimum quantities are exceeded by a factor of 2–4 in order to meet dilution specifications for methane and dust.

Many continuous miner sections use auxiliary exhaust fans to provide additional air quantity in the active face and to exhaust methane and dust directly to the return airway. Each continuous miner and longwall section must be ventilated on a separate split of fresh air, and return air from one section must not be used to ventilate another mining section. This is in stark contrast to European mining systems that use single-entry gate roads and may operate gate development ahead of the longwall face on the same split of air. Even in this case, these combined operations must still meet statutory requirements for methane, dust, and other contaminants.

Unlike for coal mines, US regulations do not provide for specific minimum ventilation quantities for underground metal and nonmetal mines. Dust and contaminant regulations are similar, though, and if mines release flammable gases, limitations similar to coal mines apply.

6.4.1 Methane control

Most underground coal operations cannot dilute the methane released from the coal seam by ventilation alone. These mines actively drain methane prior to, during, and after mining. Coalbed methane (CBM) drainage is accomplished through the following methods:

- Methane drainage boreholes drilled into the coal seam from the surface: Often, multiple vertical holes are drilled from a single location and are deflected 90° into a horizontal "daisy" pattern once they reach the coal seam. Within the coal seam, holes are often "fracked" to improve gas flow. Some mines establish pipeline networks at the surface to capture the extracted methane and sell it.
- In-seam gas drainage holes drilled from underground locations: Methane drainage from underground boreholes is also common in the industry. In seam drilling is usually easier, but underground gas drainage lines must be closely monitored for leakage.
- Roof and floor drainage holes: Often, coalbed methane that has penetrated roof and floor strata seeps out into mine openings, particularly if these strata are porous or contain cracks or fissures. Drainage can be accomplished by drilling from the underground workings or from the surface and, if necessary, fracking the strata.
- Gob ventilation boreholes: Longwall gob areas often fill with methane and must be drained, along with methane accumulations in the hanging wall strata. Often, rider coal seams above the mined horizon are broken up in the subsidence process as the longwall gob forms releasing additional methane into the gob area that requires drainage.

6.4.2 Continuous miner dust control

Respirable dust with particles sizes below 10 μm can lead to lung diseases, including coal miners' pneumoconiosis (CWP), also known as black lung disease, silicosis, and other debilitating respiratory illnesses. Recent reports [3] indicate a resurgence in

CWP and progressive, massive fibrosis (PMF) among coal miners over the past 15 years where previously, these diseases had been nearly eradicated. They state that the prevalence of PMF has reached levels not seen since the 1970s.

Dust control for continuous miner operations is chiefly accomplished by the face ventilation system in conjunction with dust control water sprays installed on the cutting drum as well as on the machine frame, and, in many cases, dust scrubber systems that scrub and filter the exhaust air behind the cutting drum. Face ventilation should preferably be exhausting so that fresh air enters the section and contaminated dust-laden air exits the section behind the ventilation curtain or through exhaust tubing and an auxiliary fan. It is important that the inlet end of the exhaust tube is maintained within 3–5 m (10 to 15 ft) of the face to prevent short circuiting of fresh air causing methane and dust accumulations in the face area.

Water sprays on the cutting drum as well as on the continuous miner frame capture and bind respirable dust with larger water droplets that either drop out of the exhaust air stream or can then be captured in a scrubber system. A scrubber system may be used on the continuous miner in conjunction with a blowing auxiliary fan and tubing. The blowing face fan has the advantage that facilitates fresh air reaching and sweeping the face more effectively, while the dusty exhaust air is then cleaned by the scrubber system.

6.5 Belt ventilation and dust control

Proper ventilation and dust control for conveyor belts and belt systems is equally important as face ventilation. Conveyor belts are a frequent source of mine fires and the belt fire at the Aracoma Alma mine in 2006 caused two fatalities. Following this fire, the US Mine Safety and Health Administration (MSHA) formed the "Technical Study Panel on the Utilization of Belt Air and the Composition and Fire Retardant Properties of Belt Materials in Underground Coal Mining" that recommended, among other findings, the following[4]:

- If belt air is used to ventilate a working section, an atmospheric monitoring system (AMS) must be utilized to monitor the air quality in the belt entry with CO and smoke sensors. This is codified in 30 CFR §75.350–351.
- Respirable dust concentrations in belt conveyor entries should be kept below 1 mg/m^3 determined in an 8-h, time-weighted average.

Belt air flow away from the face is preferred because it permits the mine operator to apply rock dust to belt entries while miners are working in the face area. Also, in the event of a belt fire, smoke and fire gases travel toward the return rather than to the face.

For belt dust control, belts should be equipped with water spray systems at the loading and discharge points. Scrapers should be installed and maintained to keep the belt clean and to reduce the formation of dust to a minimum. All belt runs must be frequently inspected for proper alignment to avoid rubbing on the structure.

6.6 Ventilation of longwall faces in coal mines

Longwall sections require considerable more fresh air on the face than continuous miner operations. Typical longwall panels are ventilated using between 30 and 50 m^3/s (70,000–110,000 cfm) of fresh air on the face. There are two fundamentally different ventilation systems in use, bleeder ventilation systems and progressively sealed gobs. Fig. 6.1 shows a schematic comparison of these ventilation systems. The bleeder system usually requires multiple, parallel gate roads that can be maintained open throughout the life of the longwall panel and is ventilated in an "H" configuration. The progressively sealed layout is known as "U"-type ventilation.

6.6.1 Bleeder ventilation systems

Bleeder systems are systems of exhaust entries surrounding a mined-out area. They are required in US longwall coal mines as per 30 CFR §75.334(b). The technique was first developed in conjunction with room-and-pillar mining and was effective in draining smaller, pillared areas of methane-air mixtures while keeping active faces supplied with fresh air. Research [5] shows that longwall gobs cave tightly under overburden stresses, which reduces the permeability of the gob and limits the ability of surrounding bleeder systems to effectively drain the methane, especially with today's longwall panel geometries that have widths of 300–450 m (1000 to 1500 ft) and may be over 6000 m (20,000 ft) long. Additional research [6–8] using computational fluid dynamics (CFD) modeling confirms that with bleeder systems, a fringe area with methane-air-mixtures in explosive concentration may form along the edges of the

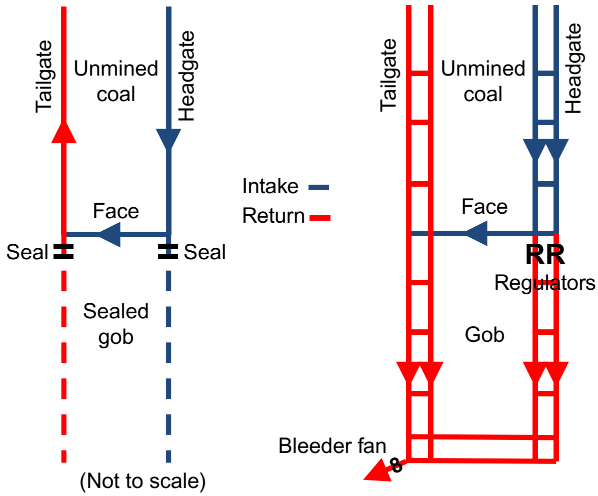

Fig. 6.1 Schematic comparison of longwall section ventilation systems: progressively sealed or U-type (*left*) and bleeder or H-type (*right*).

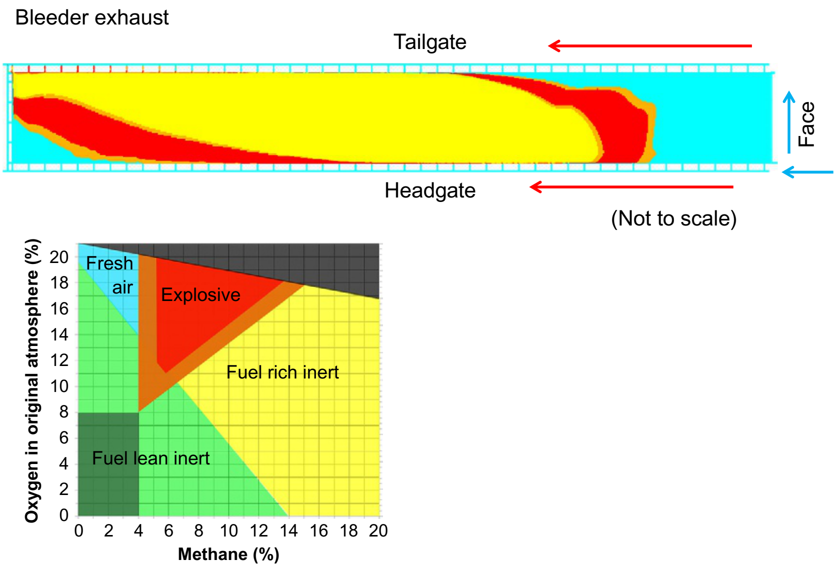

Fig. 6.2 Computational fluid dynamics model image *(top)* showing the formation of an explosive methane-air fringe *(red)* surrounding the methane-rich area in the center of the gob. Color coding is based on Coward's triangle of methane explosibility *(bottom)* with explosive mixtures in *red*, fuel-rich mixtures in *yellow*, fresh air in *blue*, and inert atmospheres in *green*.

longwall and pose a fire and explosion hazard for the mine. An example of the formation of such an explosive fringe is shown in Fig. 6.2.

A fuel-rich zone of methane-air forms in the center of the gob, surrounded by a fringe of explosive methane-air mixtures along bleeder entries in both gate roads. Since bleeder entries must be traveled regularly for inspections, mine examiners may be exposed to explosion and fire hazards. Also, depending on ventilation pressures, the location of explosive zones may vary within the gob and explosive mixtures may form in close proximity to the active longwall face. This will be discussed further in Section 6.7.2.

6.6.2 Progressively sealed gobs

Mine operators may petition MSHA to operate a progressively sealed gob instead of a bleeder system. In conjunction with progressive sealing, injections of nitrogen can be used to effectively control the explosion hazard resulting from methane accumulations in the gob. Progressive sealing is especially important if the coal tends to spontaneously combust, which is discussed further in Section 6.7.3. Progressive sealing is a ventilation technique successfully applied in European and Australian longwall coal mines.

For progressively sealed gobs with multiple gate road entries, Fig. 6.3 shows locations of seals built to close off crosscuts between gate road entries on the headgate side

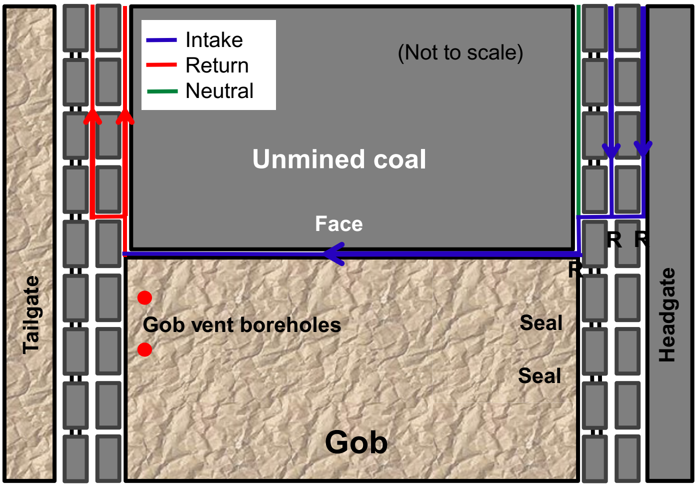

Fig. 6.3 Schematic longwall face ventilation detail for a progressively sealed panel.

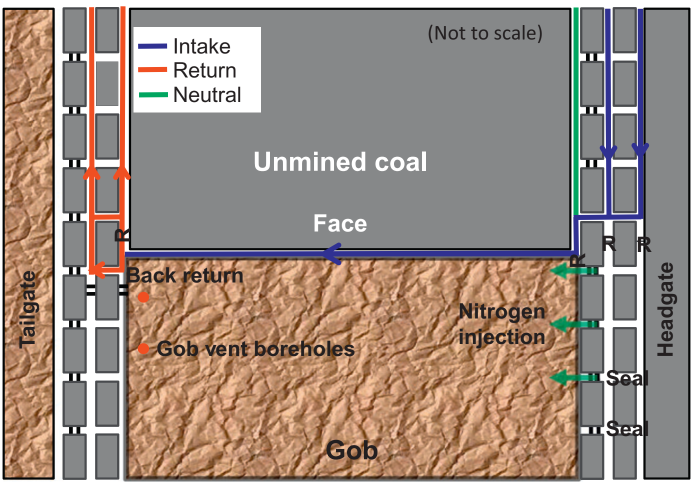

Fig. 6.4 Schematic longwall face ventilation detail for a progressively sealed panel with a back return arrangement.

as the face advances. Seals on the tailgate side were built when the previous panel was mined and the current tailgate served as the headgate. The fresh air, marked in *blue*, is fed to the longwall face from the headgate side while the exhaust air, marked in *red*, moves outby from the longwall face on the tailgate side.

Fig. 6.4 shows a variant of the U-type pattern referred to as a back return. By regulating the outby tailgate, part or all of the face air is directed inby and through the next open crosscut. This moves the low pressure point inby from the tailgate and prevents accumulations of methane near the tailgate corner. Green arrows show typical positions where nitrogen may be injected through headgate seals to inertize the gob behind the face. Similar back return arrangements are used in European longwalls. Europeans commonly use single-entry gate roads and build a continuous seal "dam" along the tailgate. To create a back return, a window is left in the dam to be sealed at a later time.

6.7 Mine fire and explosion prevention, refuge chambers

Fires and explosions have caused catastrophic accidents in coal as well as metal and nonmetal underground mines. In coal mines, methane gas and spontaneous combustion of coal are the most common causes for fires and explosions, with other ignition sources, including open flames, faulty electrical equipment, and hot metal smears caused by worn cutter bits. Blasting used to be a frequent cause of coal mine explosions, but the use of permissible explosives along with the mechanization of the cutting process has made such explosions rare. In metal and nonmetal mines, frequent sources of fires include diesel and electrical equipment, batteries, flame cutting and welding, and spontaneous combustion of timber and trash.

The mine ventilation system must be capable of exhausting smoke and fire gases away from escape routes for employees. Many modern mine ventilation modeling programs give the user the option of placing a fire at any point inside the mine and observing where the smoke travels and which escapeways may remain usable. Metal and nonmetal mines are frequently equipped with underground refuge chambers where miners whose escape is blocked can seek temporary shelter from fire and smoke. Such refuge chambers have been used during numerous mine fires and have allowed miners to wait safely until the fire hazards were cleared. In metal and nonmetal mines, fires typically burn out from lack of fuel after several hours.

In the wake of explosion disasters at the Sago Mine in the US State of West Virginia and the Darby Mine in the US State of Kentucky that occurred in 2006, the US Congress passed the Mine Improvement and New Emergency Response (MINER) Act requiring, among other safety improvements, the installation of underground refuge chambers in coal mines as well. Chambers must enable life support for all miners potentially unable to escape, and must be kept within close reach of face areas. Since faces in coal operations typically move 15–30 m (50–100 ft) per day, many chambers are mobile and moved up with the face regularly. Fires in underground coal mines may continue to burn for days, months and, in some cases, years due to the abundance of fuel the coal provides. Missions to rescue miners from refuge chambers are often difficult if the mine ventilation system is compromised allowing methane to accumulate near the fire area and creating explosion hazards. Planning and maintaining open, accessible escapeways provides the best chance for miners to survive and escape a coal mine fire or explosion.

If a coal mine is on fire, there is little chance of fighting the fire directly. Due to ventilation, mine fires grow rapidly, in some cases leaving only minutes before the fire grows out of control. Mitchell [9] states that fires that are not controlled within the first 2–4 h will likely lead to sealing the fire area or the entire mine. Fires have a significant effect on mine ventilation. Due to the heat and smoke generated, the air over the fire expands and makes the fire act as a regulator. The energy added to the air by the fire makes the fire act as a fan at the same time. This effect is stronger if the fire is on a ramp ventilated uphill. If the ramp is ventilated downhill, a fire can cause air reversal. There are several ventilation programs that allow examination of such air reversal effects caused by fires. Model calculations are also important to determine an appropriate sealing sequence for portals and shafts so that air reversals are prevented, especially in the fire area.

After sealing the mine, the atmosphere can be inertized by pumping in nitrogen, boiler gas, or, if the topography is suitable, flooding the mine with water. Boreholes should be drilled to access mine workings for air quality and temperature sampling to verify that the fire has been extinguished. Coal mine fires often require several months of wait time before the mine can be reopened.

6.7.1 Prevention of face ignitions

Face ignitions are frequently caused by longwall shearers or continuous miners cutting abrasive rocks such as quartzitic sandstone with worn cutter bits [10]. According to one study of statistics published by MSHA [11], US coal mines have experienced between 30 and 60 frictional ignitions annually. Since 2010, the number of reported ignitions has gone down to below 20 per year. Still, any frictional ignition has the potential to start a major mine explosion or fire. A frictional ignition is suspected as the primary cause of the 2010 explosion at the Upper Big Branch (UBB) mine in the US State of West Virginia [12]. This explosion fatally injured 29 miners.

Contrary to common belief, it is not the sparks that ignite methane but a smear of white-hot metal abraded from the cutter and left on the rock [13]. Such hot smears are often caused when the softer steel shank of a worn or missing cutter bit rubs against the rock. Therefore, prevention of frictional ignitions begins with regular inspections of cutter drums and replacing all worn or missing bits immediately. The mine operator should have a program in place to ensure that cutting is always done with sharp bits. This also reduces dust formation, as dull bits grind coal and rock and produce significantly more dust.

A well-designed, maintained, and functioning water spray system is equally important. Wet-head cutting drums on continuous miners and longwall shearers are designed to have sprays directly in front of or behind the cutting bit. Typically, water sprays are more effective if the water is finely dispersed. Smaller water droplets evaporate more easily and thereby take away heat from the cutting area. Water sprayed behind the bit cools off any hot smears. A simple way to ensure the effectiveness of water sprays is to continuously monitor both the water flow and the pressure of the bit spray system. European mining laws require an interlock that shuts down the mining equipment if either the flow rate or the water pressure deviate from design values by more than $\pm 5\%$. Such interlock systems are not required in US mines. In order to keep water nozzles from clogging, the supply water must be thoroughly filtered and filters and pumps maintained regularly. Again, mine operators should have in place a maintenance program that ensures that all water sprays are functioning properly and that clogged or damaged sprays are replaced immediately.

6.7.2 Prevention of gob gas explosions

Fig. 6.5 shows a frame from an illustration video published by MSHA [14] to visualize the gob gas accumulations that may have led to the UBB explosion. In MSHA's UBB investigation report [12], it is suggested that an explosive gas zone (EGZ), shown in *green* in the figure, migrated from the gob area into the active longwall face area where it was ignited, likely by the shearer cutting into sandstone roof in the tailgate area.

This explosion and numerous other incidents, including fires and gas ignitions at the Willow Creek (1998 and 2000) and the Buchanan (2005 and 2007) mines, as discussed in detail by the author [4], demonstrate that EGZs can be present in longwall gobs and that they can cause fatal mine explosions and fires. The safety hazards posed by EGZs can be characterized as follows:

- If the EGZ lies closely behind the longwall face, flames can penetrate shield supports and reach the active face area creating blast trauma and burn hazards.
- EGZs can be pushed around inside the gob by roof collapses and cave-ins. If they get pushed out into the face area, sudden methane inundations of the face area can result.
- Fresh air flowing into the gob can create an explosion hazard as the gas composition in the gob moves from fuel-rich inert to explosive, thereby creating an EGZ.
- Ignition sources inside the gob can be frictional ignitions or spontaneous combustion.

The best method to prevent the formation of EGZs in longwall gobs is progressive sealing of the gob inby the face, along with injecting nitrogen from the headgate side through headgate seals in combination with a back return ventilation arrangement on the tailgate side. Marts et al. [8] demonstrated from CFD modeling calculations that the injection of nitrogen forms an inert barrier between the methane-rich atmosphere deep inside the gob and the face area. This effect is shown in Fig. 6.6.

Color coding is again based on Coward's triangle. Both gobs are progressively sealed. Gob A shows the typical formation of an explosive fringe zone between the fresh air region along and behind the face and inside of the gob. Gob B shows the effect of nitrogen injection that forms a dynamic seal between the fresh air and the fuel-rich zone. Methane injection points on the headgate side are also indicated in Fig. 6.4.

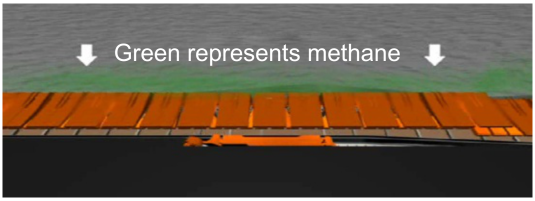

Fig. 6.5 An EGZ in the longwall gob at the upper big branch mine [14].

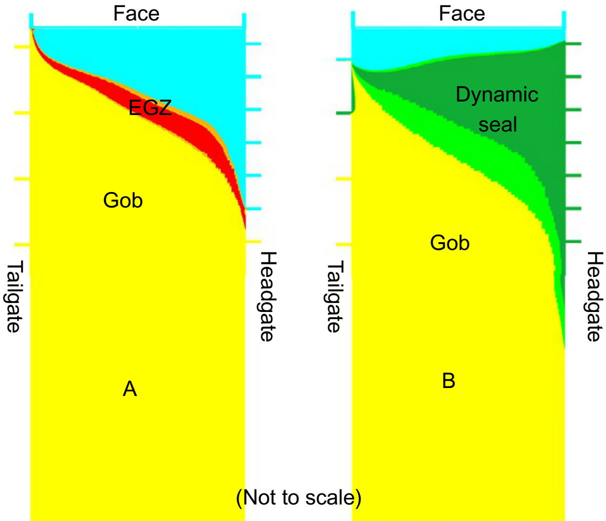

Fig. 6.6 Typical development of an EGZ in a progressively sealed gob *(left)* and formation of a continuous, dynamic seal *(green, right)* that separates the fresh air *(blue)* from the fuel-rich zone *(yellow)*, eliminating the EGZ.

6.7.3 Prevention of spontaneous combustion

Spontaneous combustion is the tendency of coal to react with available oxygen and burn. Spontaneous combustion is a combination of complex chemical reactions that are the subject of research worldwide and that are not yet fully understood. Not all coals have a propensity for spontaneous combustion. For example, most coals mined in the eastern US are usually not susceptible. European, Australian, and some Central and Western US coals must take precautions. Spontaneous combustion happens when the coal is exposed to sufficient amounts of oxygen to support the chemical reaction and when the air flow is insufficient to take away heat. Therefore, fractures and failing pillars, gob areas, and areas of geologic disturbance such as fault zones and intrusions frequently create conditions in which coal can spontaneously combust.

Combustion is noticed by an increase in CO measured in return airways, often combined with a characteristic odor of volatile hydrocarbons that are released as the coal heats. Unless controlled quickly, spontaneous fires can grow rapidly and lead to catastrophic loss of the mine. Spontaneous combustion can be controlled by isolating and progressively sealing all mined-out areas, injecting nitrogen or other inert gases; for example, exhaust gas from kerosene boilers. If spontaneous combustion is a known hazard, the mine must be set up to quickly flood-affected workings with inert gas. Continuous monitoring of all exhaust air streams and rapid response to any increase of CO are essential to managing the spontaneous combustion risk.

6.8 Ventilation network planning for metal and nonmetal mines

Metal and nonmetal mines often feature dead-ended drifts that are several hundred meters long. These dead-ended drifts require auxiliary ventilation using ventilation tubing to pump fresh air to the face or exhaust contaminated air from the face. In metal and nonmetal mines, ventilation of dead-ended drifts can be reduced to a minimum if there are no persons working in the face and no equipment is operating in the area. Often, auxiliary fans can be shut down until the next person or vehicle enters the drift. Ventilation planning must account for the intermittent operation of these auxiliary fans. Large mines may operate one hundred or more auxiliary fans, and accounting for all possible combinations of auxiliary fan operation is not feasible. The ventilation engineer must design for various peak demand and worst-case scenarios, which requires close coordination with both mine planning engineers and operators.

Some mines install remotely controlled auxiliary fans that can be turned on, off, or varied in speed based on current demand. Demand is either triggered by diesel vehicles passing certain control points prior to entering the drift, or by levels of diesel particulate, CO, and NO_x contamination determined by AMS stations. Some ventilation-on-demand (VOD) systems are controlled by a central computer system, while others operate with local control circuits. It should be noted that controls for auxiliary fans must be capable of turning each fan on in anticipation of the diesel equipment entering the stope. By the time the equipment enters, ventilation must be fully developed. Likewise, VOD fans must be kept running until the contaminant plume from diesel or blasting has been sufficiently diluted.

Proponents of VOD technology often claim savings in power cost, but investment costs for a VOD system may be substantial, especially when considering that auxiliary fans may need to be replaced with higher-capacity, variable-speed, controlled models. Also, the overall mine ventilation system must be designed for sufficient additional capacity that meets the highest potential demand from the control system, especially as operating times for VOD fans tend to overlap as contaminants are cleared.

Some metal and nonmetal mines practice controlled recirculation of air that contains contaminants at levels far below designated limits. Controlled recirculation can reduce the amount of fresh air needed to ventilate the mine, but must be carefully monitored to ensure that all contaminant and dust limits are being met at all times. Controlled recirculation is generally not permitted in coal mines due to the explosion hazard of accumulating methane.

6.9 Air cooling and refrigeration

In deep metal and nonmetal mines, ventilation air must often be cooled to maintain acceptable working conditions. A few deep European coal mines also require air cooling. The human body has evolved to maintain a core temperature below 38.5°C (101°F). This means that, as a rule of thumb, the dry bulb mine air temperature should be kept below 28°C (82°F). Mine air often has a high relative humidity, which makes working conditions worse as the human body cannot evaporate sweat to cool itself.

Heat in mine air comes primarily from three sources: autocompression, rock temperature, and diesel equipment. Autocompression occurs as barometric air pressure rises with decreasing elevation as fresh air enters a mine through a shaft or slope because gases heat up when they are compressed. Hartman et al. [15] provide the following estimates: Dry bulb temperature ΔT_d increases 9.7°C per 1000 m (5.3°F per 1000 ft) and wet bulb temperature ΔT_w increases 4.4°C per 1000 m (2.4°F per 1000 ft). For a 3000-m deep shaft, $\Delta T_d = 29$°C and $\Delta T_w = 13$°C.

The natural rock temperature also increases as mines are developed deeper underground. The rock temperature gradient with depth varies with location depending on geologic formation and proximity to volcanic areas. Gradients may range between 1 and 5°C per 100 m depth (0.6 to 3°F per 100 ft). Rock heat is transmitted to mine air via convectional heat transfer in a rather complex mechanism, as the rock is also cooled by the air. Water entering the mine workings after flowing through crevices in hot mine rock may also be a significant contributor to mine air heat and may increase humidity, increasing the perceived heat of the air.

Diesel engines reject heat of fuel combustion to the mine air. At an overall efficiency of 25%, a diesel engine running at 100 kW would create 400 kW of heat. On average, the engine puts out much less than its rated power, with common power factors between 25% and 50% depending on usage [16]. Therefore, a better estimate for total heat generated by a diesel engine is the combustion heat content of the fuel consumed.

To cool the ventilation air to acceptable temperatures for miners, three primary methods are used: bulk intake air cooling, spot cooling, and cab cooling. With bulk cooling, all intake air is cooled as it enters the shaft or as it flows through an underground cooling plant. With spot cooling, cooling fluid is chilled at the surface and pumped to local heat exchangers underground, often combined with auxiliary fans. The third method is to install local air coolers in each piece of diesel equipment so that the operator can enjoy cool temperatures inside the cab. Enclosed cabs also improve air quality as dust can be filtered out.

6.10 Summary and conclusions

Mine ventilation planning and close coordination between the ventilation, mine planning, and operations engineers are essential for all highly productive mining operations. Ventilation provides fresh air to all active mine workings and for internal combustion engines and dilutes harmful gases and dusts. In underground coal mines, ventilation air quality must be monitored primarily for methane, oxygen deficiency, diesel exhaust, dust, and any signs of combustion, indicated by the presence of carbon monoxide. In metal and nonmetal mines, ventilation quantities are typically lower compared to coal mines, as explosive methane is generally not present. Still, monitoring should be done for diesel particulate matter, respirable dust, and radon. Dust monitoring is especially critical if the dust contains quartz or other components causing lung disease.

In planning the ventilation system, engineers should establish mine ventilation network computer models and design airways and fans so that sufficient ventilation can be achieved in all mine workings at the farthest extent of the mine network. It is important to provide adequate airway cross sections to deliver the required airflow while maintaining acceptable fan pressures and power requirements. If larger cross sections cannot be mined due to ground control limitations, parallel airways should be considered to deliver sufficient amounts of air.

Continuous monitoring of air flows, pressures, and air quality is an essential element of mine ventilation management. Also, ventilation engineers should maintain an up-to-date ventilation model of their mines to make accurate ventilation projections as mine development progresses and new mining sections come on line.

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Developing effective proximity detection systems for underground coal mines

7

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7.1 Introduction

Accidents involving underground mobile mining equipment striking a worker or colliding with another piece of equipment can often be attributed to the operator's lack of visibility around the equipment, or, in the case of remote-controlled equipment, the operator positioning himself or herself in close proximity to the equipment during operation. To help address this problem, systems known as proximity detection or proximity-warning systems (PDS) are used to detect nearby objects or workers and provide this information to the appropriate equipment operators. PDS are also sometimes referred to as collision warning or collision avoidance systems. The use of these systems can augment safe operating procedures such as monitoring mirrors, checking blind areas before moving a piece of equipment, and selecting a safe operating location for remote control machinery.

The last several years have seen a rapid increase in available proximity detection technology for the mining industry. Early methods to assist operators in avoiding collisions consisted mainly of closed-circuit video camera systems. Now, detection and warning systems based on radar, radio frequency identification, magnetic-field detection, and computer vision are becoming available [1–3]. Major trials and deployments of these technologies for mobile machines are occurring worldwide in both surface and underground mines. This technology can also be applied to monitoring hazardous areas around stationary machines. Thus, intervention research and development has focused on the detection of personnel near machines.

After a brief illustration of why proximity detection systems are needed in underground coal mines, a discussion of the technology follows, including a summary of related National Institute for Occupational Safety and Health (NIOSH) research and future directions.

7.2 Fatal accidents at underground coal mines

Originally, continuous mining machines (CMMs) were operated from a seated position in an on-board cab that protected operators, but severely restricted their visibility. Unfortunately, this configuration also exposed operators to machine vibrations, dust,

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and noise. In the 1980s, CMMs were redesigned with radio remote controls. By removing the operator from the machine cab, several safety hazards associated with having the operator near the coal face were alleviated. With the implementation of remote controls, operators also became free to position themselves for best visibility to perform job tasks and were no longer subjected to machine vibrations during operation. However, this freedom of movement exposed the operator to the hazard of being accidently struck or pinned by the CMM and other nearby large moving machinery, since operators and their helpers often stand in close proximity to the machine in order to see visual cues needed to operate the machinery. Since 1984, there have been 42 fatalities involving striking and pinning of the operator and other workers by the CMM. In 2009, the Mine Safety and Health Administration (MSHA) conducted an analysis of 38 striking and pinning accidents involving CMMs. MSHA estimated that of these 38 fatal striking and pinning accidents between 1984 and 2009, the use of proximity detection could have been a preventative factor in at least 28 cases. Furthermore, MSHA estimates that proximity detection could prevent 20% of all deaths throughout the industry [2].

In August 2011, MSHA published a proposed regulation that would require proximity detection systems on all continuous mining machines except full-face machines [4]. These systems are designed to stop machine motion to protect miners from striking/pinning hazards. The final rule was published in 2015 and several MSHA-approved proximity detection systems are commercially available.

7.3 Technologies for PDS in underground applications

7.3.1 Radar

Radar systems transmit a radio signal from a directional antenna that is mounted—for the purposes of this discussion—on a vehicle. The radio signal is reflected off of objects that are within the transmitted beam and a portion of the reflected energy returns to the receive antenna. Depending on desired system performance characteristics, transmit and receive antennas can be combined or separate. From this reflection, the distance from the radar unit to the object can be determined. There are two primary measurement methods for a radar sensor—pulsing and continuous wave. Pulsed or ultra-wideband (UWB) radar detects obstacles by measuring the time of flight (ToF) of a pulsed signal that is transmitted and then reflected from an object within the radar's beam with distance between transmitter and reflector being proportional to this time. The continuous-wave method works by transmitting a radar signal of a known, stable frequency, and then measuring the Doppler shift of the reflected signal. The Doppler shift indicates the change in frequency of the reflected signal, which is proportional to the speed of the target.

Typically, these systems operate in the microwave (300 MHz–40 GHz) portion of the radio spectrum. Doppler radar detects the relative motion of an obstacle; i.e., detection requires either movement of the obstacle or the vehicle. Both types of radar are effective for detecting people, other vehicles, large rocks, and buildings. Some obstacles are not good radar reflectors, such as plastics, dry wood, or objects with large flat surfaces that can reflect signals away from the radar antenna. Possible obstacle detection ranges for a radar-based proximity warning system vary from less than a meter to 30 m or more. Radar works well in dirty and dusty environments and does not require any miner wearable component (MWC); however, it is only useful in line-of-sight applications and cannot differentiate between objects and humans.

7.3.2 Sonar

Sonar or ultrasonic sensors operate on the principle of transmitting a high-frequency sound wave at an object, and then measuring the reflected echo off of the target. The sensors used in these systems are capable of converting an alternating current into ultrasound and the reverse; converting ultrasound into an alternating current. Some systems use separate sensors to transmit and receive; others combine both functions into a single sensor. These systems determine distance based on ToF measurements along with the propagation speed of the sonic wave in the propagation medium. The technology cannot tell the difference between objects and humans. Also, no MWC is required for an ultrasonic PDS. The frequency of the sound is above that of human hearing (greater than 20 KHz).

Sonar systems for vehicles have very limited range—typically less than 3 m (10 ft). Thus, they will work well in near-range ranging applications but cannot detect through barriers. These sensors are directional, and provide a narrow beam detection range. Therefore, multiple sensors are needed to cover the width of a large vehicle. These sensors can be sensitive to particles in the air (dust, snow, and rain) and must be kept fairly clean to avoid any debris buildup on the face of the sensor. Improvements to these types of sensors are possible, and new systems may become available that would be suited for underground mining applications.

7.3.3 Radio frequency identification

Radio frequency identification (RFID) utilizes electromagnetic (EM) fields to identify and track objects using a reader and tags. Tag-based proximity warning systems use electronic tags that are worn by workers, attached to small vehicles, or attached to stationary objects. Tag detectors or readers are installed on mobile equipment. There are two types of RFID tags: active and passive. Active tags have their own power source and operate at further distances than passive tags, which collect energy from the RFID reader and must be near the reader. Active technology requires the tag to transmit a marker signal that is detected by the tag reader. If the tag is within a certain range, an alarm is generated in the cab of the equipment. Two-way communication between the reader and the tag allows alarms to be generated at the tag also. The passive methodology is similar, but the reader transmits the marker signal. If a tag detects this signal, an alarm condition is sent to the reader and an alarm is generated both in the cab and at the tag. Several technologies have been used to generate the marker signals that determine tag proximity: ultrasonic, magnetic, and radio frequency (RF). At the writing of this chapter, several of these systems were being developed for surface mining applications.

There are a variety of ranging calculations that could be used to determine distance between the reader and the tag. Commonly, ToF information, along with the propagation speed of the signal, is used to assess the distance from a tag to a reader. Or, similar to magnetic-field-based proximity detection systems, the use of multiple readers could allow for the tag's position to be determined through trilateration or other means.

RFID can detect objects outside of line of sight and its performance is not affected by dust in the air. This technology could provide localization that is similar in nature to magnetic-field PDS. As such, some problems will be common between the two, including electromagnetic interference (EMI), environmental effects, and general performance.

7.3.4 Magnetic-field systems

Currently available PDS approved for use in underground coal mines are based on EM technology. A typical system currently used in underground coal mines consists of four generators positioned around the perimeter of the machine. Each generator is a coil of insulated wire wrapped around a ferrite core. The magnetic-field strength is proportional to the current running through the coil. Miners working on the section have an MWC, which is a transceiver that measures the field strength emitted by the generators and transmits a data packet containing the field strength reading over a radio frequency (RF) link to the PDS controller mounted on the machine. The magnetic-field generator's pulses contain identification information so that the MWC can determine which generator's field strength it is reading. This type of system utilizes the principle of magnetic flux density: the closer the MWC is to the field generators, the higher the field strength reading. Thus, a miner's presence can be determined once he gets too close to the machine, because his MWC will measure a magnetic flux density beyond a certain threshold. These thresholds are utilized on a per-generator basis to "shape" the fields for both warning and stop zones around a machine, as illustrated in Fig. 7.1.

7.3.5 Infrared-based systems

Infrared proximity sensors transmit an invisible infrared light beam and detect reflections from nearby objects. Previously, infrared proximity sensors had limited detection range and there were concerns with reliable operation in the mining environment [5]. Improved systems with detection ranges of up to 9 m (30 ft) are now available, and these technologies are gaining popularity in some construction and industrial applications. It is not known how effective they would be in a mining environment. Infrared video cameras (thermal imagers) detect the thermal signature radiated from a person and provide an enhanced image, especially in low-light conditions. Applications of these devices for avoiding collisions between vehicles and people have been commercialized.

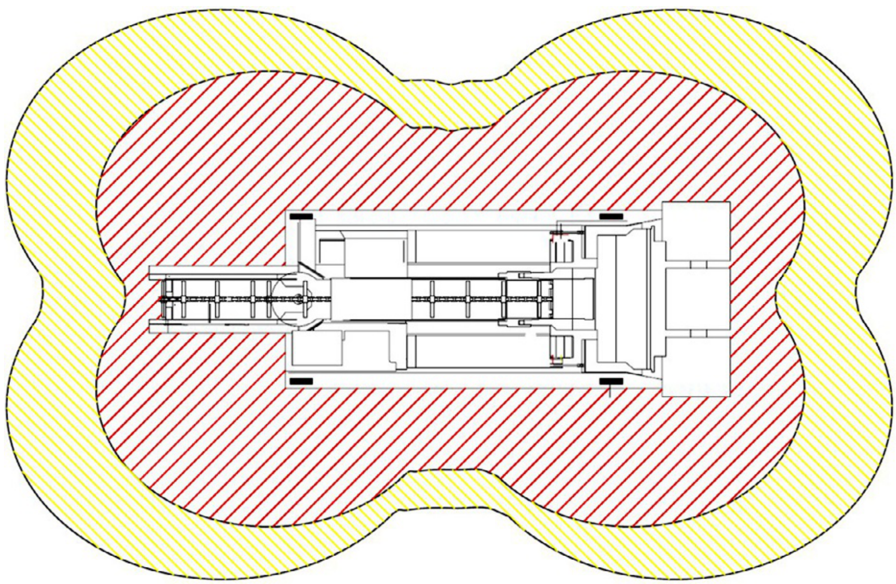

Fig. 7.1 Representation of a CMM equipped with an EM-based PDS, showing warning *(yellow)* and stop *(red)* zones.

7.3.6 Computer-vision-based systems

Computer vision is a broad and growing field of computer science and robotics that utilizes visual data captured from cameras to detect and analyze obstacles and environments. Monocular cameras are not capable of providing ranging without using a stereo camera system (use of multiple lenses and image sensors); however, cameras are capable of providing visual indicators to operators to enable better use of their equipment. Furthermore, computer algorithms can be developed such that mine workers can automatically be detected within the camera's field of view. This would apply to standard black and white images as well as thermal imaging cameras, as computer vision algorithms can be developed to detect edges and shapes, or to identify temperature thresholds that would indicate the presence of a miner.

7.3.7 LIDAR

Light detection and ranging (LIDAR) is a ranging method based on laser technology. These units typically involve a laser with an oscillating mirror that enables the unit to conduct ranging in 2D space. 3D laser scanning can be achieved with a multiaxis unit. The operating principle is based on ToF: the unit emits a pulse of laser energy and then measures the response using a photodetector. Once one position has been measured, the unit oscillates to the next position, and then performs another measurement. This process is repeated over the oscillating range of the unit allowing for a 2D measurement sweep to be recorded. LIDAR units can achieve up to 360 degree horizontal field of view with measurement rates up to 50 Hz. Like radar, it is only useful in line-of-sight applications and cannot differentiate between objects and humans; however, it does have high accuracy and does not require an MWC.

7.4 NIOSH research on magnetic-based proximity

7.4.1 Early NIOSH research (HASARD)

NIOSH first developed electromagnetic proximity detection technology in 2000 as the Hazardous Area Signaling and Ranging Device (HASARD) [6]. This system used an electromagnetic-field generator to create a magnetic-field measurable by an MWC thereby giving a rough indication of the distance between the generator and the MWC, as illustrated in Fig. 7.2. The HASARD system was based on the use of low-frequency (73 kHz) magnetic fields and the concept that the strength of the field decreases approximately proportional to the cube of the distance from the source.

It is important to emphasize that HASARD is a two-part active system composed of a single transmitter and one or more receivers, depending on the circumstances. The transmitter is installed on-board the CMM. The receiver is worn by the CMM operator, helper, and other miners working near the CMM. The HASARD receiver was designed to be an omnidirectional (direction-independent) magnetic-field strength meter. Multiple antennae detect the field, and the internal electronics amplifies and filters the signal and converts it to a DC voltage level. This signal level is compared

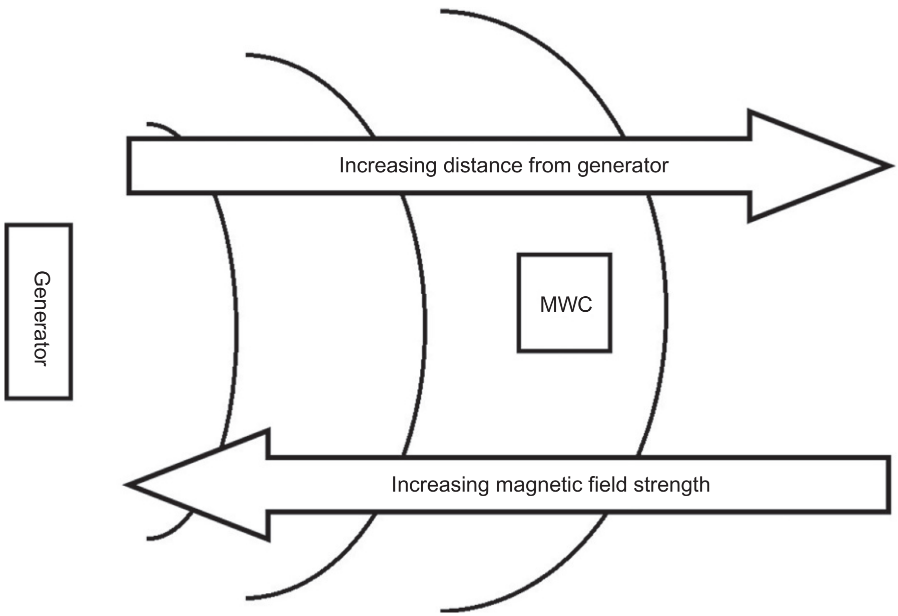

Fig. 7.2 Conceptual drawing of proximity detection operating principle.

to thresholds that define the field strengths representing zones of safety, caution, and danger. Each threshold can be adjusted and affects the size of the danger and caution zones for the entire machine. The size of the protection zone for a particular or specific location is changed by adjusting the current to the appropriate generator.

Several manufacturers adopted the concepts developed with HASARD and further refined the technology. Several systems are now commercially available to the mining industry; however, the complex and nonspherical shape of the magnetic fields still makes accurately determining distance difficult. Both HASARD and the systems currently available on the market simply trigger alarms or machine shutdown based on predetermined threshold values for the magnetic flux density. This results in a somewhat ambiguous protection zone around the machine, which is difficult to shape, and does not provide for situational or intelligent response to hazards. While the proximity detection manufacturers have spent a great deal of effort trying to shape the protection zones, the currently available systems sometimes interfere with the operator's freedom to efficiently perform his job.

Due to visibility and space limitations, miners must routinely work in very close proximity to the CMM, and it is common for an operator to be located within 1 m of the machine in order to see the visual cues needed to operate it [7]. To be acceptable to miners and to avoid false alarms, a PDS must provide the necessary protection while still allowing normal operation of the machine. This is difficult to achieve without an intelligent system that can make decisions based on situation-specific conditions. Accurate knowledge of worker position and posture enables the implementation of intelligent protection capable of issuing alarms that are more meaningful or disabling only specific machine functions, depending on the case at hand.

7.4.2 Magnetic-field modeling

A magnetic proximity detection system relies on magnetic flux density measurement (B) to determine the position of a worker relative to a mobile mining machine. It is desirable for the magnetic flux density distribution to be automatically adjustable to conform to the protection requirements for the different types of machines and working environments. In support of the development of an automatic field distribution adjustment process, NIOSH researchers developed a transferrable magnetic flux density distribution model [8], which can also be used to control and stabilize the field against field drift to enhance system performance.

Previous NIOSH research [9] showed the B field distribution from a ferrite-cored generator was described in terms of magnetic shells that are surfaces of revolution around the axis of the generator. Each shell (Fig. 7.3) represents a surface of constant B field magnitude. A shell function is an analytical expression for the magnetic surface. Shells vary in shape and size depending on the distance to the generator.

The general properties and parameters of the shell-based magnetic flux density distribution model for a generator are as follows. Eq. (7.1) shows the model covering the three-dimensional (3D) space around a magnetic generator. The model defines a magnetic shell with a given B value. The coordinate system and the symbols used in (1) are defined as shown in Fig. 7.4, in which a generator of length L lies along the x-axis and

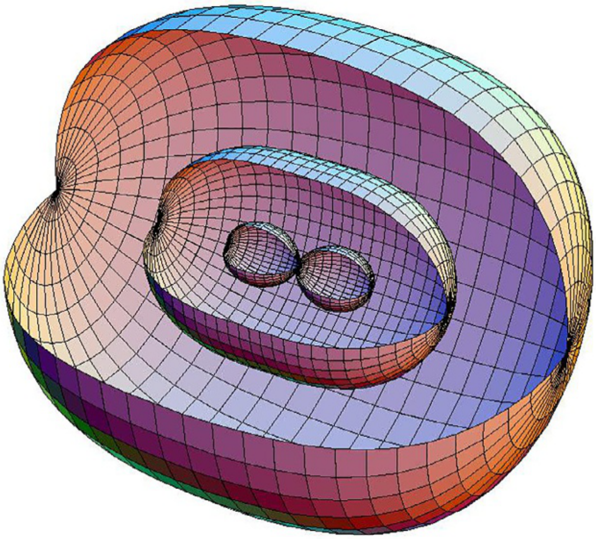

Fig. 7.3 3D shell model.

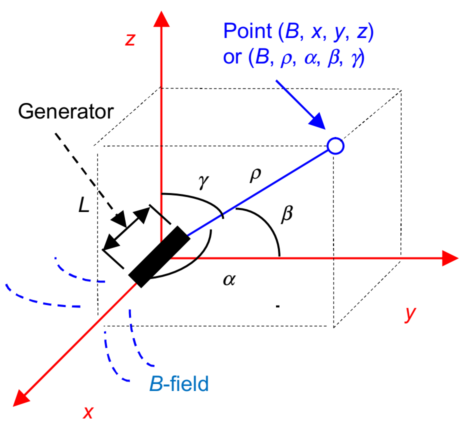

Fig. 7.4 Generator and magnetic-field coordinate system.

is centered at the origin. Eqs. (1a), (1b) are equivalent representations of the shell functions in the Cartesian and the direction cosine systems, respectively. In Eq. (7.1), ρ represents the distance from a point on the shell to the origin; α, β, and γ represent angles from the x-, y-, and z-axes, respectively, to the line on which ρ is measured. A shell can be generated from Eq. (7.1) with a given B reading, and each shell is described by a function in the form of either (1a) or (1b) that is uniquely and completely defined by two parameters, a and b, as defined in terms of the B reading by (1c) and (1d). The shell shape parameter, a, determines the variation of the shell from its basic shape of a sphere with radius, b, the shell size parameter.

$$\text{Shell }(x,y,z|B)\text{ or Shell }(\rho,\alpha,\beta,\gamma|B)=\begin{cases}(x^2+y^2+z^2)^{1/2}=a\left(\dfrac{x^2-y^2-z^2}{x^2+y^2+z^2}\right)+b & \text{(1a)}\\ \text{or } \rho=a(\cos^2\alpha-\cos^2\beta-\cos^2\gamma)+b & \text{(1b)}\\ \quad \text{for } a+b>L/2 & \\ a=c_aB^{-d_a} & \text{(1c)}\\ b=c_bB^{-d_b} & \text{(1d)}\\ \quad \text{for } B>0 & \end{cases} \tag{7.1}$$

The shell shape function (1c) has two positive constants, c_a, the shell base shape constant, and, d_a, the shell shape changing constant. Similarly, the shell size function (1d) has two positive constants, c_b, the shell base size constant, and, d_b, the shell size changing constant. These four constants have fixed values for a given steady magnetic field and completely describe the magnetic field in its defined space. These constants are defined by the physical distribution characteristics of a given magnetic field that is determined by many factors: primarily the length of the ferrite core, permeability of the core material and medium, the number of turns of the coil, the current flowing through the coil, and the impedance of the generator. These constants can also be determined empirically using data from magnetic-field measurements.

The ratio of a to b is called the shell shape changing ratio, and it is a good indicator of the variation of the shell shape from spherical. Its value is uniquely determined by the value of B, as shown in Eq. (7.2). No two shells have the same shell shape changing ratio, and, therefore, no two shells have the same shape. It has been observed that, typically $0<c_a<c_b$ and $0<d_a<d_b$, and that as the value of B increases, the ratio a/b will also increase. As B is larger close to the generator, the ratio a/b increases close to the generator, resulting in greater deviation from a spherical shape to a more deeply concaved shell. Conversely, further from the generator, the value of B is smaller yielding a ratio a/b that is smaller, and the resulting shell will be more spherical in shape.

$$\frac{a}{b}=\frac{c_aB^{-d_a}}{c_bB^{-d_b}}=\frac{c_a}{c_b}B^{d_b-d_a} \tag{7.2}$$

With a single B reading, it is only possible to determine the shell on which the measurement is made. The exact distance between the sensor and the generator cannot be directly determined since the shell is never a perfect sphere, and points at different locations on the shell will have different distances to the generator.

7.4.3 Effect of worker posture

The mining process requires workers to change posture and position based on several factors such as roof height, machinery location, and mine ventilation. Previous studies have addressed worker positioning around the CMM rather than posture [10,11]. Some investigations unrelated to mining have focused on wireless and embedded sensor technology to determine human posture [12–14]; however, these studies were ultimately concerned with human position in specific postures. NIOSH research identified underground worker postures and determined the transition between them. This research shows that by measuring and acquiring key reference joint angles, underground mine worker posture can be analyzed and determined.

Determining worker posture can be important because the height of the MWC can change dramatically with changes in posture. As described earlier, EM fields are a revolving shell about the generator axis. Fig. 7.5 depicts a hypothetical way that a difference in posture could cause a change in detection distance. In the case depicted, the shape is such that a kneeling miner could get closer to the machine than a standing miner.

In underground field tests, the stop zone distance with the MWC at a height of 16 in. (40.6 cm) was typically within 18 in. (46 cm) of the stop distance with the MWC at a height of 46 in. (116.8 cm). In some cases, however, a difference of between two and 3 ft (0.6–0.9 m) was observed. Similar observations were made relative to the warning zone. The 46- and 16-in. heights correspond to the 95th percentile waist height for an adult male in a standing position and the 5th percentile waist height for an adult female in a kneeling position, respectively.

7.4.4 Current magnetic system performance

NIOSH researchers conducted field tests on magnetic-field-based PDS installed on CMMs in active underground coal mines with seam heights ranging from 54 to 84 in. (137.2–213.4 cm). Nine field tests were conducted throughout the United States to

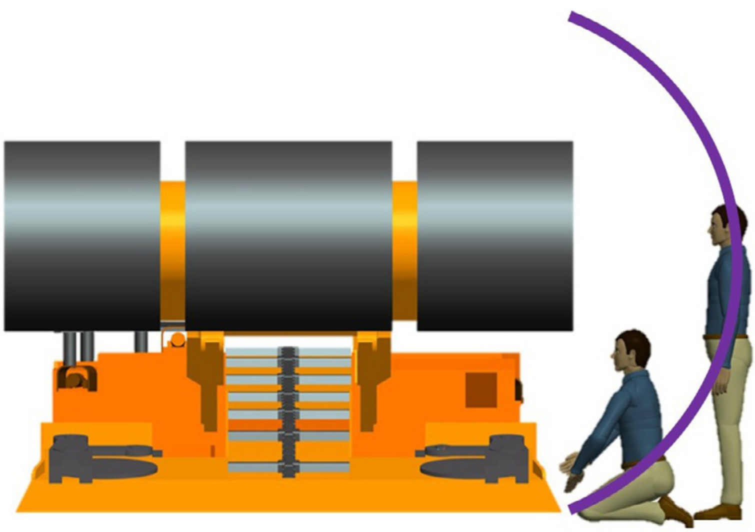

Fig. 7.5 Effect of posture on detection distance.

evaluate the performance of PDS. Performance was evaluated primarily by determining the distance from the MWC to the CMM when warning and stop zones were detected.

The goal of these tests was to assess the performance of the second generation of proximity detection systems in terms of functionality and repeatability. These tests followed a protocol developed with input from industry and the West Virginia Mine Safety Technology Task Force. NIOSH researchers used a custom measurement apparatus to characterize the warning zone and stop zone distances by moving an MWC toward the machine until an MWC alarm was activated. Researchers recorded the distance at which the alarm was activated for the warning zone, continued to move toward the machine until an alarm was activated for the stop zone, then recorded that distance. This was executed at 10 specific test points around the perimeter of the machine, as shown in Fig. 7.6. Each test was repeated and if the second measurement varied from the first by more than 6 in. (15.2 cm), a third reading was taken.

These field tests were designed to quantify the performance of PDS for four different conditions: baseline CMM operating (not in mining mode), CMM operating in mining mode (described later), influence of a trailing cable on the MWC, and influence of a shuttle car in the loading position. Each of these conditions is detailed here.

The baseline testing was performed in order to establish a performance baseline for which to compare the other test conditions. These tests were conducted with the following conditions: the conveyor boom centered and in its low position, the cutter head inactive and dropped to the floor along with the gathering pan, the trailing cable in the normal position, and no shuttle car. These tests established a performance baseline to compare against the other three tests (mining mode, influence of trailing cable, influence of shuttle car).

Mining mode is a feature on newer commercially available PDS that reduces the zones toward the rear of the machine along the conveyor boom and at the rear bumper, allowing operators to position themselves near the CMM while cutting coal. Mining mode is activated whenever the cutter head motors are energized. This allows operators to be in a position where they have sufficient visibility to perform their work while out of harm's way of the shuttle car when the CMM is cutting coal. Mining mode is not used in some mines because it was not implemented on older first-generation PDS.

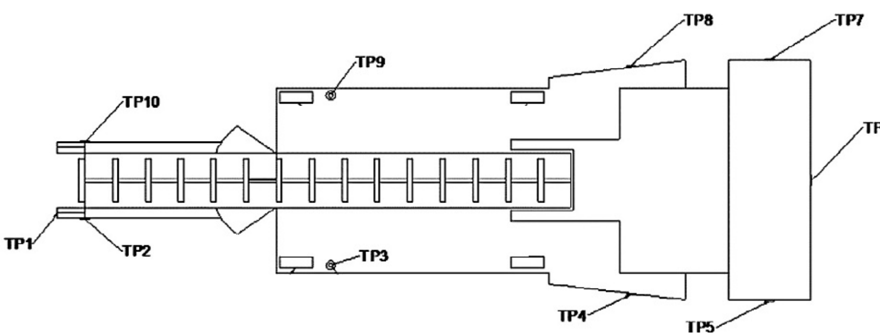

Fig. 7.6 Test point locations.

Also, it is not required in some applications that use continuous haulage. Whenever the cutter head on the CMM is not active, such as when the machine is tramming between locations, mining mode is not active, and thus the zones are not reduced. Field test results show that mining mode significantly reduces warning and stop zone sizes at the rear of the CMM, in some cases to zero. Although there are no recorded fatal striking/pinning accidents that have occurred while cutting coal, this reduction in warning and stop zones could allow contact with the CMM by an operator and could be a safety hazard.

A principle known as parasitic coupling has been reported [15] by a number of mine operators, where it has been suggested that MWCs located near a trailing cable are causing spurious machine shutdowns even though the operator is not near the machine. It is suggested that parasitic coupling may occur when the magnetic field generated on the CMM couples to the trailing cable, effectively extending the warning and stop zones farther than intended. Because of this possibility, the influence of a trailing cable on zone measurements was examined in field tests. The test protocol called for the MWC to be held 46 and 16 in. from the trailing cable in these tests at test point #3 or test point #9 (depending on which side the training cable exited the machine) while measuring warning and stop zone distances. These measurements were then compared to baseline test measurements for determining any effects from the training cable.

Data collected from these tests provide no indication that parasitic coupling has any effect on warning and stop zone distance measurements. Additional testing would be required using closer distances from the MWC to the trailing cable to further investigate reports of false alarms caused from parasitic coupling.

Also, when a shuttle car maneuvers into the loading position while the CMM is mining coal, a large mass of metal (the shuttle car) is introduced nearby the CMM. This could possibly reshape the field toward the rear of the machine. Specific field tests were designed to quantify the effect that the presence of a shuttle car has on the PDS system. This field test data show that the effect of the shuttle car being present was to enlarge warning and stop zones slightly.

Overall test results indicate that while system performance is generally good, there is still room for improvement due to large variations between field test sites. Since all approved PDS have adjustable zone distances, this variability is due to preferences set at installation. Site-specific conditions need to be considered when defining warning and stop zone distances. A number of measurements showed stop zones to be less than 3 ft (91.4 cm). This distance may be inadequate for a CMM operating under adverse traction conditions, such as a wet floor on a slope. These zone distance settings should be tailored to mine conditions, keeping safety in mind. Field test measurements for both warning and stop zones were repeatable, and the majority of readings only required two measurements. Detailing complete results is beyond the scope of this chapter, but interested readers are referred to a comprehensive NIOSH report [16]. Future development and installations of PDS should take this information into consideration and work to improve system precision. Additionally, mine operators and PDS installers may want to consider minimum distances when setting zones for production.

7.5 Development of intelligent proximity detection (iPD)

In 2009, NIOSH researchers began developing Intelligent Proximity Detection (iPD) technology designed to provide improved protection for miners working near CMMs. The iPD system continuously tracks the position of all miners near the CMM and compares these positions to known hazardous locations around the machine. When a miner is detected in a hazardous location, the machine functions that could cause an accident are automatically disabled. For example, if the operator is standing behind the machine, there is no safety concern in allowing the machine to move forward, regardless of how close the operator is to the back of the machine, but reverse movement is prohibited. In this way, safe mining practices are allowed to continue uninterrupted, but the safety of miners is protected at all times.

Because iPD disables only the hazardous machine functions dependent on the miner's position around the CMM, this is expected to improve operator acceptance by minimizing the impact on the operator's normal mining routine. In addition, since only the hazardous motions of the machine are blocked, miners are permitted more freedom with respect to where they position themselves. This enables them to better avoid other hazards such as other equipment in the area and unsupported roof or ribs.

7.5.1 Localization methods

NIOSH researchers developed a sophisticated mathematical model of the shape and size of EM fields as previously described in this chapter. At the core of this model is an equation for the shapes of three-dimensional magnetic "shells" formed around magnetic-field generators. Shells close to the generator have a more abnormal shape because as the distance between the generator and the MWC is increased, shells become larger and more uniform in shape. This nonlinear variation in size and shape is very well described by this model. For any measured field strength, an associated shell exists that can be approximated using this model. This means that if an MWC detects a given field strength, the associated shell can be determined by the proximity detection system, indicating that the MWC must be located somewhere on that shell. This does not, however, give an exact position.

In order to implement iPD, it is necessary to continuously track the position of MWCs. To accomplish this, the position of the MWC is found using multiple magnetic-field generators on the CMM. The magnetic-field strength for each of the generators is measured by the MWC, and a magnetic shell is determined for each generator based on the magnetic-field model. The position of the MWC is given by the intersection of two or more magnetic shells.

Although this concept for calculating miners' positions is fairly simple, calculating the intersection of the magnetic shells is not a trivial task. Shell shapes are irregular and vary nonlinearly with distance, making it difficult to find a direct mathematical solution for the intersection. Therefore, NIOSH researchers have developed a new search method using a series of geometric approximations to calculate shell intersections.

The iPD system uses this method to continuously track the position of multiple miners around the mining machine with a high degree of accuracy.

The achievable accuracy of the position triangulation is limited by the stability and repeatability of magnetic readings. A given magnetic-field strength reading should be associated with a distinct shell around the machine. However, normal variation in the electromagnetic proximity system causes the same reading to be observed over a range of distances at any given point on that shell. This variability is influenced by several operational and environmental variables. NIOSH researchers quantified iPD system accuracy in the laboratory by taking thousands of shell measurements while varying these conditions, with variability of up to 12 in. (30.5 cm) observed.

7.5.2 Development of zones and selective shutdown

Once the position of a miner has been calculated, the iPD system provides protection against striking and pinning accidents by disabling all machine motions that could cause a collision between the machine and a miner. The decision of which machine motions to disable is accomplished by comparing the calculated position of the miner to a preprogrammed set of zones around the mining machine. Each of these zones is associated with a set of potentially dangerous machine functions. When a person is detected in a zone, the functions associated with that zone are disabled. For example, the zone to the right of the conveyor boom would be associated with conveyor swing right and all tram functions that would move the conveyor boom to the right. If a miner were standing in this zone, these motions would be disabled, but the miner would still be able to tram forward and run the cutter drum.

To gain insight into the safety potentially afforded by different proximity detection zone configurations, NIOSH researchers conducted an analysis of 39 fatalities that occurred between 1984 and 2015 in which a miner was struck or pinned by a continuous mining machine in an underground US mine [17]. The objectives of this analysis were to estimate the number of cases for which a proximity detection system may have prevented the accident, and identify the potential safety benefits of iPD systems compared to conventional proximity detection systems. Fatality investigation reports from MSHA were reviewed and analyzed for each accident to determine whether a conventional proximity or iPD system could have prevented the fatality.

Although it is mandated that all machine tram and conveyor boom functionality is shut down when a miner enters any stop zone around a CMM, for the purposes of this analysis, it was assumed that all machine functions would be shut down on a commercial proximity detection system based on proximity detection manufacturer designs. Additionally, it was assumed that iPD systems would selectively disable machine functions as previously described. In considering conventional proximity detection, it is assumed that all machine motion would be blocked when a miner is detected in any stop zone, while iPD systems would selectively disable machine functions as previously described.

Two iPD zone configurations were examined. Neither configuration is intended to be a recommendation, but rather both are presented as examples for comparing factors associated with establishing zone definitions. The first is shown in Fig. 7.7, and will be

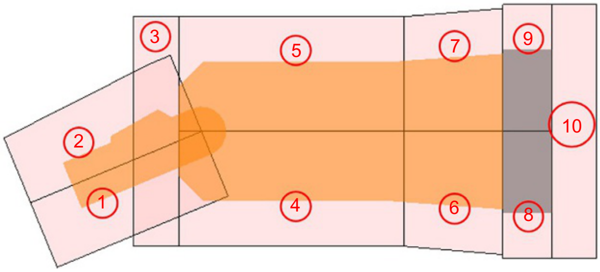

Fig. 7.7 iPD 1 zone configuration.

referred to as "iPD 1" throughout the remainder of this chapter. Zones 1 and 2 are dynamic, meaning that they follow the position of the conveyor boom as it pivots laterally to load coal onto haulage equipment. Zones 3 to 10 are based on the CMM chassis (frame) and are static.

Each zone is associated with a set of CMM functions that are disabled whenever a miner is detected within that zone. This zone layout was designed by NIOSH researchers as an example of one potential configuration for selective machine function shutdown [15,18–21]. These 10 zones were created to capture all possible machine motions that could affect a given location, and to provide zone logic to allow operators to perform actions that would not put them at risk.

The second zone configuration is shown in Fig. 7.8, and will be referred to as "iPD 2" throughout the remainder of this chapter. Zones 1, 2, 11, and 12 are dynamic and change based on the position of the conveyor boom. Zones 3 to 10 and 13 are based on the CMM chassis and are static. Some of the major differences between iPD 1 and iPD 2 are:

1. Functions blocked by Zones 1 and 2 in iPD 2 depend on the position of the conveyor swing. If the conveyor is centered (within some preselected tolerance), blocked functions are less restrictive than if the conveyor is swung either left or right.
2. Forward and reverse tram functions in iPD 2 are blocked for Zones 4 to 9 to prevent accidents when a miner is beside the CMM.
7. All tram functions in iPD 2 are blocked for Zones 6 and 7 to prevent any unsafe pivoting motions.
4. Zone 11 has been added to iPD 2 to account for the area that exists between Zones 1 and 2 around the conveyor boom. This prevents any unsafe conveyor boom motions when an operator is near the tail by providing a buffer between Zones 1 and 2.
5. Zones 12 and 13 have been added to iPD 2 to account for a miner being on top of the CMM. These zones were added to account for fatalities involving maintenance on top of the machine.

Results of this study indicate that 82% of fatalities could have been prevented by conventional proximity detection systems. The two different iPD zone configurations presented illustrate different factors that affect performance and both were analyzed over the same set of 39 fatal accidents. This analysis showed that iPD 2 could have prevented 82% of the fatality cases (the same as conventional proximity), while iPD 1

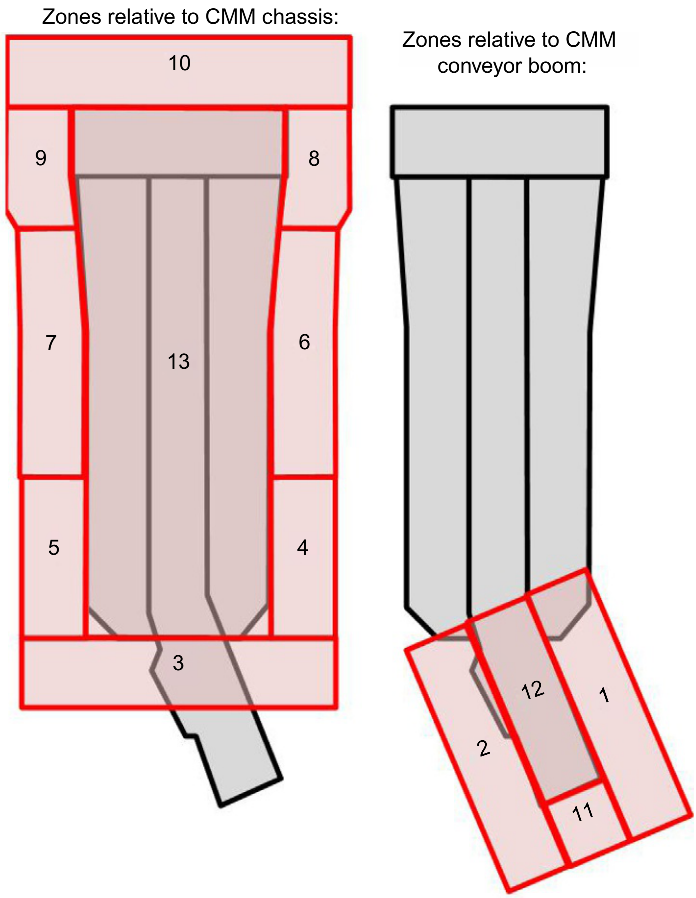

Fig. 7.8 iPD 2 zone configuration.

could have only been a preventative factor in 62% of the accidents. This indicates that by implementing iPD into commercially available proximity detection systems, miners may have the safety benefits of proximity detection systems while potentially having more freedom to move around the machine. They may also be able to work more efficiently, thus potentially enhancing acceptance of these systems. Results were based on MSHA investigation reports. While sufficient accident data are not available to yield statistically validated conclusions, the analyses performed provide some positive insight into the effectiveness of proximity detection systems.

The comparison importantly illustrates that zone configuration definitions are critical to proximity detection system effectiveness. Ultimately, neither iPD 1 nor iPD 2 should be considered recommended designs, but rather examples of how zones for an intelligent proximity detection system could be configured and how different parameters affect the ability to prevent worker injuries. It should also be noted that there are a number of other factors that can influence the performance of proximity detection systems, such as conveyor elevation, cutting drum elevation, tramming, and mining mode [19]. These factors should also be taken into consideration when designing zone configurations for intelligent proximity systems.

7.6 Future developments

7.6.1 Proximity considerations for underground mobile equipment

In 2015, MSHA issued a proposed rule that would require mobile machines on working sections (with the exception of longwall sections) to be equipped with a PDS. Currently available PDS for underground mining are all based on electromagnetic technology and were originally intended for use on CMMs. In an effort to identify the applicability and transferability of EM-based PDS onto mobile equipment, NIOSH conducted field evaluations of PDS-equipped mobile haulage machines. These field evaluations consisted of having an operator drive the mobile equipment toward an MWC positioned in an entry or crosscut. In most cases, the PDS did stop the machine before it struck the MWC. However, three primary concerns have arisen with respect to EM-based PDS for use on mobile machines in underground coal mines: environmental effects, electromagnetic interference (EMI), and general performance characteristics.

Recent field evaluations [22] observed some phenomena not measured in previous field studies, including diminished performance in the vicinity of metal mesh and powered cables on the working section. These occur due to mutual inductance, which is the linking or coupling of a magnetic field from the generator coil to metal objects or wires in close proximity. It should be noted that these observations were discovered during field evaluations, but were not tested under a peer-reviewed field test protocol.

NIOSH researchers have observed EMI from continuous personal dust monitors (cPDMs) that hinder the performance of PDS significantly. The cPDM is a belt-worn monitor that takes interval samples for respirable coal dust. It was discovered by mine sites and brought to the attention of NIOSH that when miners wear their cPDM and MWC close together, the MWC begins to behave erratically. In some cases, this would permit the miner to walk all the way up to the machine without incurring a warning or stop zone, which poses a significant safety hazard. This behavior has been observed with the cPDM, but could potentially occur from other electronic devices that emit spurious electromagnetic noise within the operating frequency of the PDS.

NIOSH conducted laboratory testing of one PDS manufacturer's MWC to measure its susceptibility to EMI, along with measurements to identify the EMI being produced by a cPDM, and found that it was best to keep the cPDM and MWC at least 6 in. apart in order to prevent EMI. This finding correlates with recommendations from PDS manufacturers [23,24] of maintaining a 6-in. (15.2 cm) separation distance.

A prototype shielded pouch for the cPDM was developed by a PDS manufacturer in an attempt to mitigate EMI effects. Lab tests showed that the pouch reduced the separation distance to only 2.0 in. (5.1 cm) needed to prevent EMI effects; however, underground testing at one mine showed an increased EMI effect from the use of the pouch. The mine had mesh on the roof and ribs in the mine section used for testing. Depending on the position around the machine, the shielded pouch could actually cause EMI to be experienced at separation distances of up to 13 in. (33.0 cm). This phenomenon is under investigation as of the writing of the chapter.

7.6.2 Hybrid proximity systems

NIOSH researchers have identified a number of alternative sensing technologies that could improve or complement existing PDS. Section 7.2 described these technologies. It is possible that an improved PDS will be a fusion of a number of sensing technologies, as opposed to a single technology system.

An example of an existing sensor fusion PDS system is the Becker Collision Awareness System (CAS), which is a PDS used internationally. The Becker CAS is based on EM to provide coverage close to the machine, high-frequency RFID to provide coverage beyond the close range, and an ultra-high-frequency (UHF) RFID to provide long-range coverage. This system utilizes the advantages of each technology to provide a robust system to prevent accidents from occurring.

Sensor fusion systems could consist of a combination of EM, cameras with computer vision, and radar such that multiple measurements are utilized for detection. These systems could be utilized to provide redundant detection and ranging with validation. Radar could mitigate some of the environmental and EMI effects that hinder the performance of EM systems, while cameras with computer vision could provide validation of obstacle detection, and the EM could provide coverage through brattice cloth and around corners.

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Communication and tracking system performance

8

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8.1 Introduction

Not long ago, most underground mines had limited or no tracking or wireless communications systems in place. Mine operators were only aware of the names of the individuals that were checked-in but had little knowledge of each individual's location and no real-time information. After several coal mine disasters, the United States (US) Congress passed the Mine Improvement and New Emergency Response (MINER) Act in 2006. Among other changes, the MINER Act amended existing laws and mandated that above-ground personnel be able to determine the location of all underground personnel at any given point in time. US coal mines had three years to comply and get wireless tracking systems in place.

A number of manufacturers developed and installed systems that allow for both tracking underground personnel and equipment as well as two-way communication between the surface and miners underground. Although tracking and communications may seem unrelated, current systems combine both, since the underground location of personnel needs to be communicated to the surface in real time. For tracking systems, accuracy is the primary design constraint. According to guidelines, the location of an individual should be known with an accuracy of ± 2000 ft (~ 660 m) when not in an active mining section or near strategic areas, whereas in active mining sections or strategic areas, the miners' locations should be known within 200 ft (66 m).

This chapter covers the method of taking a mine map with locations for a mesh-based tracking and communication system and predicting tracking system performance.

8.2 Measures of tracking system performance

A set of metric values describe the performance of a tracking system based on the accuracy of position calculations across a spacial area over time. The purpose of these metrics is to provide a basic set of values that can be calculated simply and are independent of the technology used in the tracking system. They treat the tracking system as a black-box calculator and are only concerned with the actual location of a tracked device and the tracking system's calculation of the location of the tracked device. The tracked device's actual location is expected to have very little error relative to the tracking system and is the definition of the ground truth position (GTP) or the ground truth position estimate (GTPE), used interchangeably. The tracking system calculates

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a location for the tracked device; this location is the tracking system position (TSP) or tracking system position estimate (TSPE), used interchangeably. These two positions can be thought of as where the device really is located (GTP), and where the device is considered to be located (TSP) at a point in time. The technology used by the tracking system and its reporting capabilities determine how one collects data from the tracking system. It is assumed that all tracking systems installed in underground coal mines are capable of reporting a tracked device's history over the last 14 days in subminute time frames.

The most basic metric is tracking coverage area (TCA), which refers to the area within the mine that the tracking system is working. Because mines are confined spaces and the tracking system is not required to have 100% coverage, the TCA is divided into active TCAs and Inferred TCA. Active TCA is where the tracking system is actively producing TSPs. Inferred TCA is where the TSP is not in communication, but the general position can be ascertained based on spatial limitations of the mine and information other than active measurements.

The next fundamental metric is the instantaneous accuracy (IA) of the TSP, which is the difference between an actual location (GTP) and the tracking system position estimate (TSP) actively made at that GTP at an instant in time. This is a simple straight-line distance from any single TSP to the GTP at the same point in time. The equation for IA is:

$$\mathrm{IA}_0 = \sqrt{(\mathrm{TSP}_{x0} - \mathrm{GTP}_{x0})^2 + (\mathrm{TSP}_{y0} - \mathrm{GTP}_{y0})^2}$$

If GTP and TSP of a miner's location are (100, 100) and (96, 103), respectively, at time T_0, then IA at time T_0 is: $\mathrm{IA}_0 = \sqrt{(96-100)^2 + (103-100)^2} = \sqrt{-4^2 + 3^2} = 5$.

At any location in the mine, over time there are several TSP values that are calculated by the tracking system. For each TSP value, there is an associated IA; and the general accuracy of the tracking system for that area is described by average accuracy (AA), the arithmetic mean of a set of IAs, and standard deviation of accuracy (SDA), the standard deviation of a set of IA measurements.

$$\mathrm{AA} = \frac{\sum_{i=1}^{\mathrm{N}} \mathrm{IA}_i}{N}$$

$$\mathrm{SDA} = \sqrt{\frac{1}{N-1}\sum_{i=1}^{N}(\mathrm{IA}_i - \mathrm{AA})^2}$$

90% confidence distance (90% CD) is the distance from a tracked device's actual location (i.e., GTP) that is greater than 90% of collected IA measurement magnitudes ("90th percentile"). 90% CD can be specified as a metric based on a standardized, fixed percentile of measurements, as this section is entitled, but a variant of the concept can also be specified with a percent of TSPs occurring within a standardized distance. This metric is only descriptive within the active TCA because it derives from a set of IAs.

The 90% CD describes the 90th percentile of IA values from a GTP, an IA measurement in which there is a very high (90%) confidence that a subsequent IA calculation will be within the 90% CD. The calculation is done by ranking IA measurements and the IA that is at the 90th percentile rank is the value. This metric value is the closest match to the regulatory guidelines of accuracy required in various parts of the mine. The important difference is that this is a confidence that a value will be within the limit; it is not a set measurement. For instance, an IA value at 105 ft, in an area where the 90% CD is 100 ft, is not an indication that the 90% CD of that area is in fact larger than 100 ft. A set of measurements that show a 90% CD greater than 100 ft is an indication that the 90% CD should be increased for that area.

Values for AA, SDA, and 90% CD as calculated for a particular system/section are one-dimensional representations of the accuracy of the tracking system. It may also be important to describe the relative skew in the TSP in two dimensions because tracking systems report coordinates for a tracked device. The analogy of IA in two-dimensional space is the error vector (EV), which is the simple difference vector between Cartesian X and Y components of GTP and TSP. Average error vector (AEV) is the average displacement of a set of TSPs from their corresponding GTP. Average cluster radius (ACR) is the average of distances a set of TSPs are from their center point, which is determined by the average error vector. These values are determined by moving each GTP to the origin and then average TSPs to their corresponding relative locations. The resulting graphs show bias and spread in the tracking system, which are described by AEV and ACR. For a set of N TSP measurements in the active TCA and corresponding GTPs, AEV is the vector representing average X and Y coordinates of EVs associated with the set. ACR is the average of the distance of a TSP from the AEV. It describes a circular area around the AEV in which the average TSP value would be located.

$$\langle \mathrm{EV}_{x0}, \mathrm{EV}_{y0} \rangle = \langle \mathrm{TSP}_{x0} - \mathrm{GTP}_{x0},\ \mathrm{TSP}_{y0} - \mathrm{GTP}_{y0} \rangle$$

$$\langle \mathrm{AEV}_x, \mathrm{AEV}_y \rangle = \left\langle \frac{\sum_{i=1}^{N} \mathrm{EV}_{xi}}{N}, \frac{\sum_{i=1}^{N} \mathrm{EV}_{yi}}{N} \right\rangle$$

$$\mathrm{ACR} = \left(\sum\nolimits_{i=1}^{N} \sqrt{(\mathrm{TSP}_{xi} - \mathrm{AEV}_x)^2 + \left(\mathrm{TSP}_{yi} - \mathrm{AEV}_y\right)^2} \right) / N$$

Other important metrics such as reliability, availability, and relative accuracy are discussed elsewhere.

8.3 Simulation of tracking system performance

Prior to installation and testing of the communications and tracking system, computer simulations were used to generate anticipated results. These predictions were then used as the baseline for the system being tested, a partial mesh system with several different components that provide network resources. The system uses fixed nodes that are powered by mine power with battery backups. This backbone has tracking beacons that are used to supplement tracking calculations. Tracked devices are the same radios or handsets that are used for communications in many mines. The system does have tracking only devices, i.e., ones that have no voice or text capabilities; however, these devices were not used in the data presented.

8.3.1 Example Mine layout

Testing was performed in a West Virginia mine (the Example Mine) that is typical of central Appalachian coal mines in both dimension and mine design. This mine has been actively producing for several decades. The area studied was at the mine's portal, a 10-entry system with four returns, which were excluded. Communications and tracking system hardware (the Test System) was installed using guidelines obtained from the manufacturer. Infrastructure placement was based on designs provided by the manufacturer and estimates generated by the author.

In this study, only the Mains section of the Example Mine was considered as shown in Fig. 8.1. The portal is indicated on the right side of the map along with some key ventilation controls such as stoppings, equipment doors, and regulators. Other key infrastructure often shown on mine maps (including some shown later in this chapter) includes power transformers *(red dots)* and existing tracking units *(yellow dots)*. Other symbols include ventilation controls *(blue lines)*, ventilation air splits (*red arrows* for return air), conveyor belts *(green line)*, and track haulage *(orange line)*.

Current regulations require that the mine track miner locations in primary and secondary escapeways (EWs) and in designated areas to which miners are trained to go in emergencies. Areas of the mine not normally occupied by workers and areas that are not places a worker would go to in an emergency are not required to be covered. In the Example Mine, the area in which coverage was tested is the *blue-shaded region* in Fig. 8.2.

The installed tracking system was a radio frequency identification (RFID) tag system with readers located at the portal and the turn. This provides a small active TCA that indicates the transient presence of a tracked entity, but leaves a majority of the Mains area as an inferred TCA.

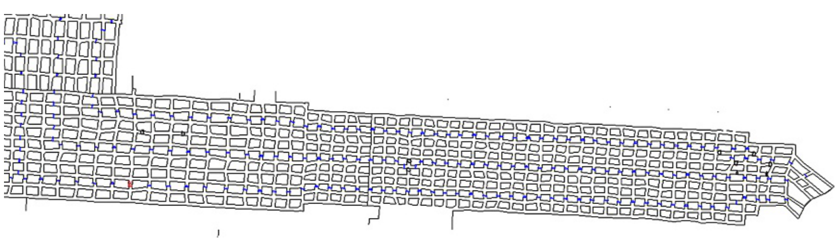


Fig. 8.1 Map of mains area.

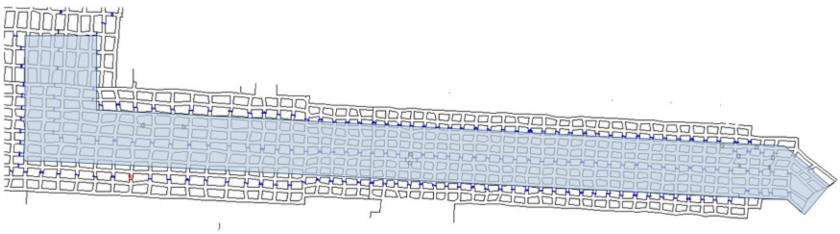


Fig. 8.2 Coverage area.

In the Test System, a computer running specialized software and located in the mine office receives frequent reports from mobile radios relaying signal strength they measure from in-range fixed radios located at fixed mesh nodes (FMNs), and from beacons (BCNs). FMN locations are known to the computer via the tracking database, which applies a proprietary algorithm to estimate the location of the mobile radio. The antenna and fixed node placement in the mesh network is determined by the system manufacturer based on the manufacturer's system design procedure. The signal-level modeling tool developed by the author, generates estimates of tracking system metrics at a large number of locations throughout the mine area under evaluation. The same calculations of tracking system metrics can be produced manually, though much greater efficiency is possible when using the simulation tool.

The mine geometry is extracted from the mine map and simulation locations emplaced at every intersection of entries and crosscuts, and also at locations halfway between the intersections. Mesh infrastructure device locations and configuration are likewise emplaced as CAD entities, being input parameters from which the signal paths to each simulation from each antenna are found. The CAD entity parameters determine signal loss of the path with the least nominal loss. The simulator then runs a number of statistical cases chosen by the analyst, for which the path loss is varied in a uniform distribution within the range of variation of loss along each path. The output from these predictions is used to calculate metrics for the tracking system. Estimated nominal and variation values used are listed in Table 8.1.

Table 8.1 **Nominal values and variation used in simulation**

Parameter	Nominal value (dB)	Max variation (dB)
Transmitter power	BCN: $-2\,\text{dBm/m}^2$ FMN: $16\,\text{dBm/m}^2$	± 1 ± 1
Forward propagation loss in entries and crosscuts	6 dB/100 ft	± 1
Loss through ventilation control stoppings	14 dB	± 5
Loss around 90 degrees corner	36 dB	± 10
Loss crossing conveyor belt	17 dB	± 10

After calculation of signal levels and variations at a location are completed, the manufacturer's tracking algorithm is applied to signal strengths appearing at half-pillar and intersection locations to generate calculated position estimates. These position estimates are then used to determine estimated values of tracking system metrics.

In the five illustrations that follow (Figs. 8.3–8.7), two examples showing plots of 250 TSP around the GTP for which they were simulated are provided. For the first example, Fig. 8.3 shows a close-up of the mine map where the selected GTP (916) is located. Fig. 8.4 is the scatter plot of tracking position data around the GTP 916 location for randomized signal attenuation factors. The maximum tracking error for this example is less than 40 ft. Plotting the scattered TSPEs shown in Fig. 8.4 and adding some of the nearby fixed radio nodes on the mine map renders an optional view like the one in Fig. 8.5.

The second example is of tracking results at GTPE 357 located about 800 ft inby the portal on the secondary EW. Fig. 8.6 shows the location enclosed in a ring on a portion of the mine map. Fig. 8.7 is a scatter plot with proportional mine map scale showing the 250 TSPEs produced from tracking measurements made at GTPE 357. The maximum tracking error for this location is a single outlying TSPE at a distance of 971 ft inby GTPE 357.

Simulations and calculations also can estimate how changing the tracking system component configurations affects metric values. Node and antenna position shifts and addition or removal of nodes are examples of configuration changes that can

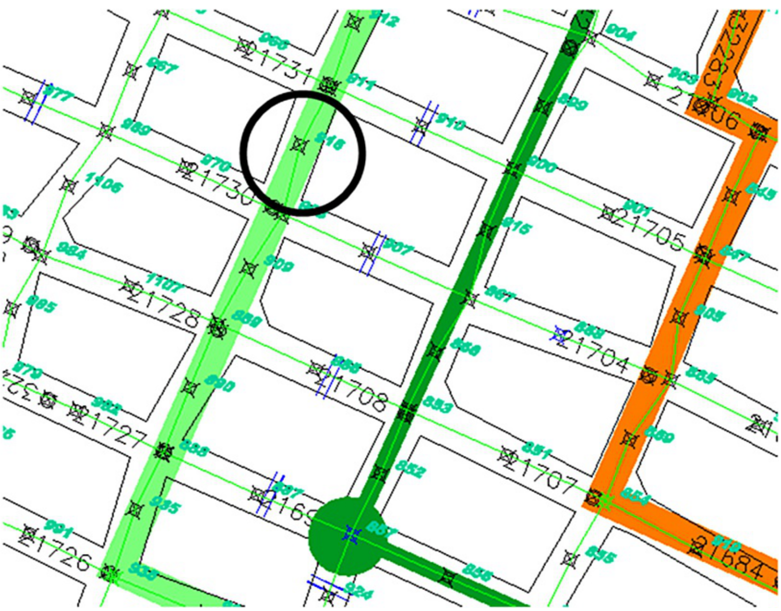


Fig. 8.3 Mine map showing the location of GTP 916 on the primary EW at about 4650 ft inby the portal.

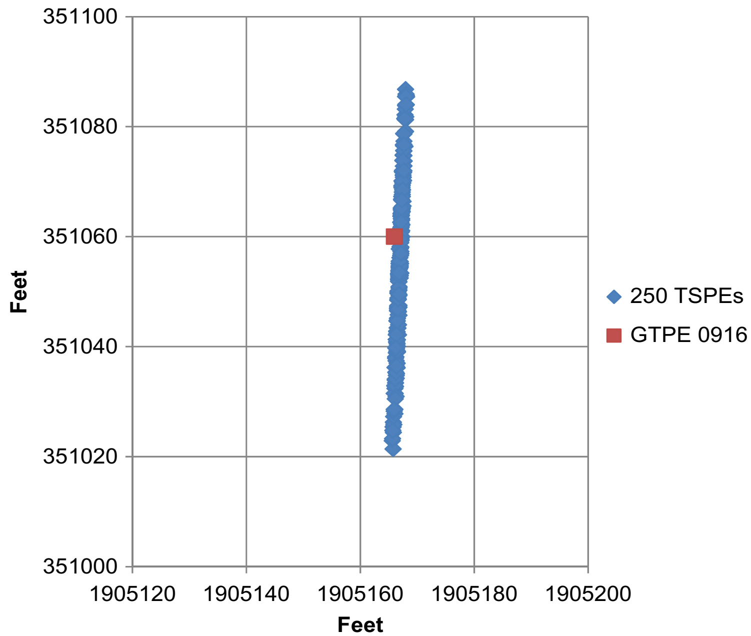


Fig. 8.4 Plot of 250 randomized TSPEs around the GTPE 916 location.

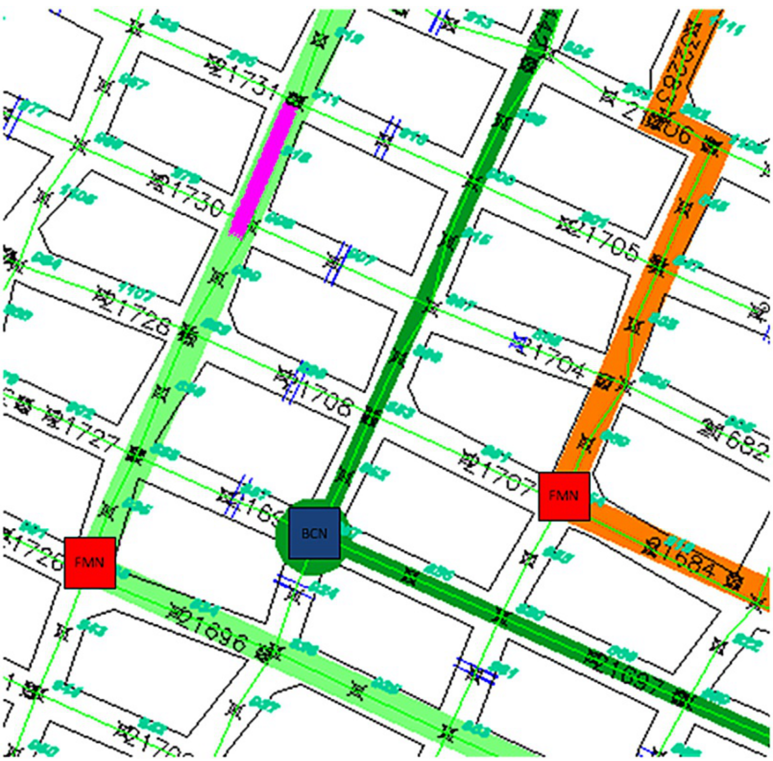


Fig. 8.5 Plot of simulated tracking TSPEs around GTPE 916 superimposed on the mine map.

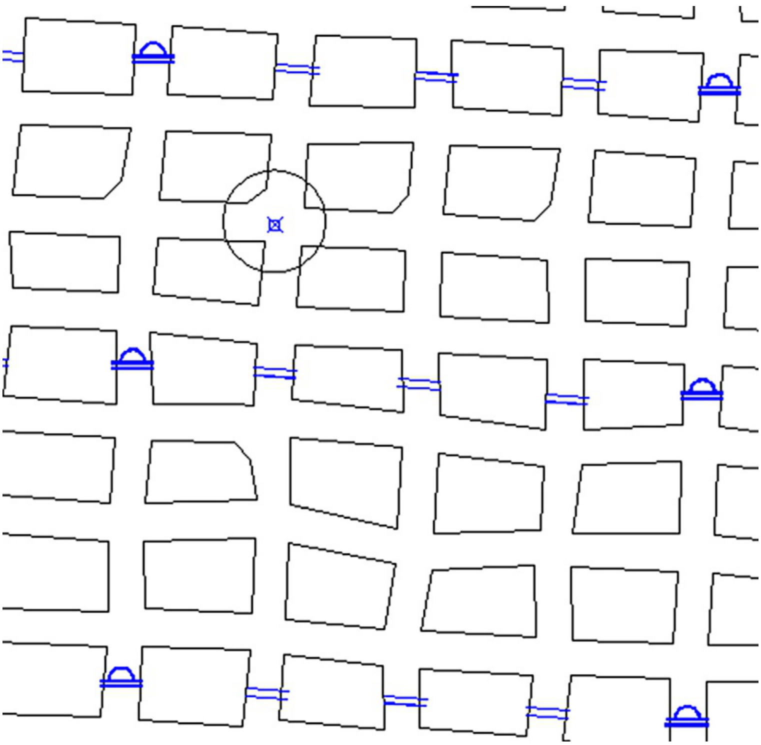


Fig. 8.6 Mine map showing GTPE 357 location in an intersection on the secondary EW.

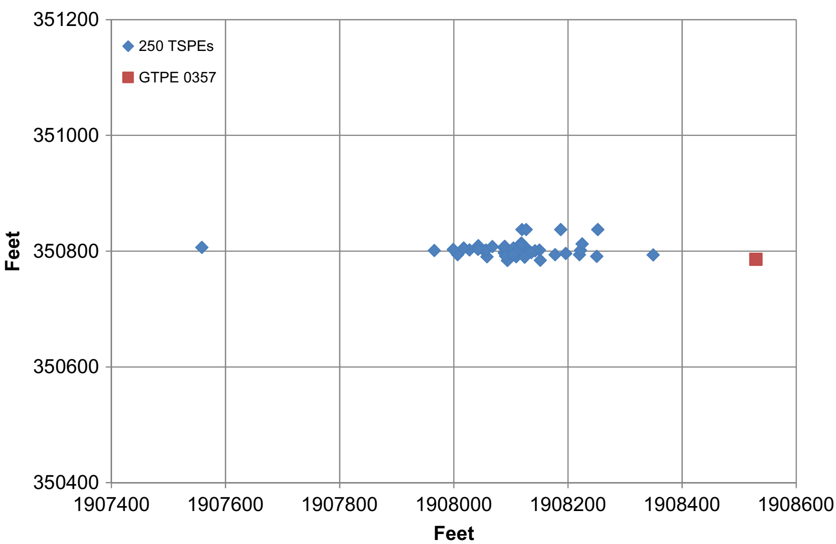


Fig. 8.7 Proportional plot of 250 simulated TSPEs for GTPE 357.

be evaluated. When accurate values of CAD entity parameters are determined, the simulation and calculations of metric values for different tracking system configurations may help optimize system design to meet tracking system requirements.

One of the main objectives of the author was to achieve tracking coverage in primary and secondary EWs and typical strategic areas in the study portion of the Test Mine, and generally, to get signal into the belt entry, which is not an escapeway in the Example Mine. The Test System layout for the Example Mine was designed by the vendor to provide radio coverage to meet communication and tracking standards set by MSHA.

In Figs. 8.8–8.16, FMNs are shown as *blue ellipses*; however, the placement and direction of antennas connected to FMNs is the most important factor for modeling signal levels throughout the mine. Antenna positions and orientations for each FMN are shown by *blue arrows*. There are 15 FMNs underground and two aboveground just outside the primary (Entry 5) and secondary (Entry 7) EW portals. The belt entry (Entry 6) has no FMNs and is expected to receive radio coverage through crosscuts from Entries 5 and 7. The two above-ground FMNs provide links to the gateway node (GWN) at the control shack. GWNs in this case are ignored in the simulation because they will have no underground signal.

BCNs are shown as *purple ellipses*. There is one about 210 ft inby the portal on the belt entry. There are two at SCSR caches (about 2500 and 4300 ft inby the portal) and one at the belt head at the corner (approximately 4300 ft inby the portal). Fig. 8.8 provides an overview of the test area and radio tracking node deployment used for baseline simulations. Figs. 8.9–8.16 show more detailed views of deployments.

Fig. 8.9 depicts the two outside nodes at the portal. They are linked to the first inby pair of FMNs on the escapeways. Outside FMNs have antennas pointing into the portals and also to the gateway at the control building about 300 ft away. Fig. 8.8 shows the short distance of underground coverage afforded by outside FMNs; however, the short distance assures robust links to FMNs at the dog-leg bends in entries (shown in the middle of Fig. 8.10). Also, note the airlock door (large "D" symbol) one crosscut inby the secondary EW *(orange color)*, which may attenuate radio signals.

Fig. 8.11 shows the location of FMNs and respective antenna directions about 700 ft inby the portal on the primary EW. The nearest FMN on the secondary EW is also shown. The two airlock doors shown on the secondary EW attenuate radio

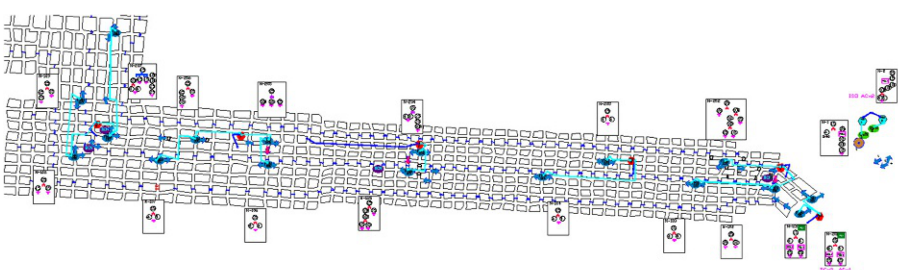
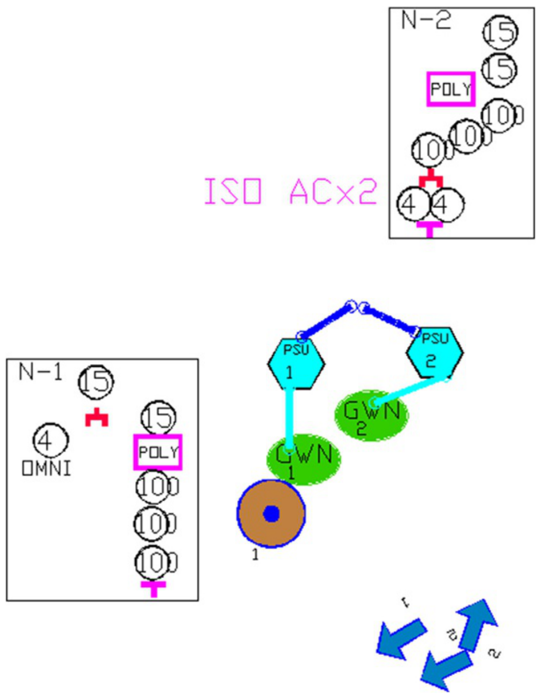


Fig. 8.8 Overview of Test System in Example Mine.

Fig. 8.9 Layout of gateway nodes and antennas at the mine office building.



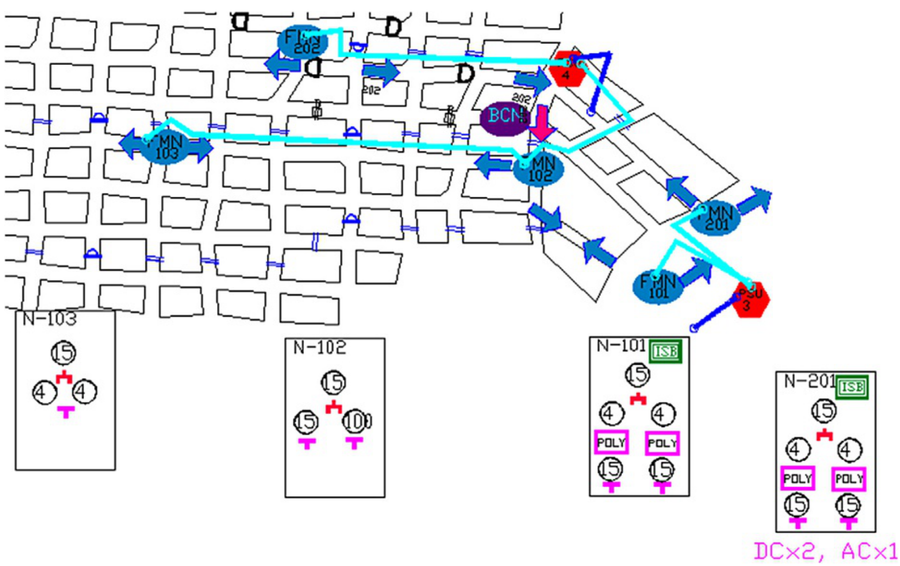


Fig. 8.10 Layout of FMNs at the portal.

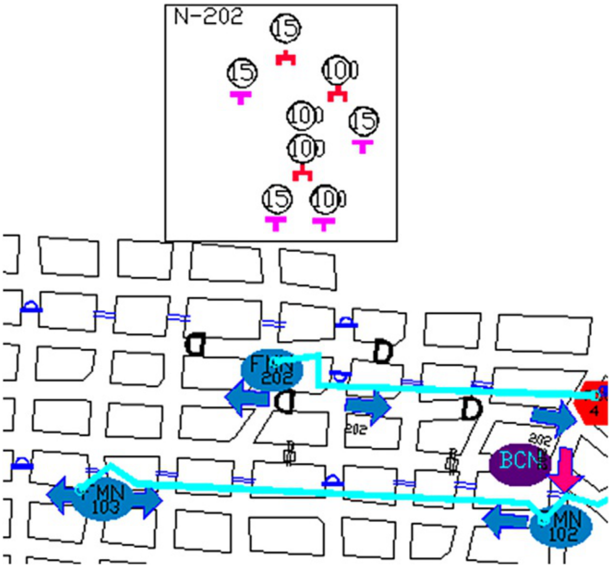


Fig. 8.11 Layout of primary EW node about 700 ft inby the portal and the nearest secondary EW node.

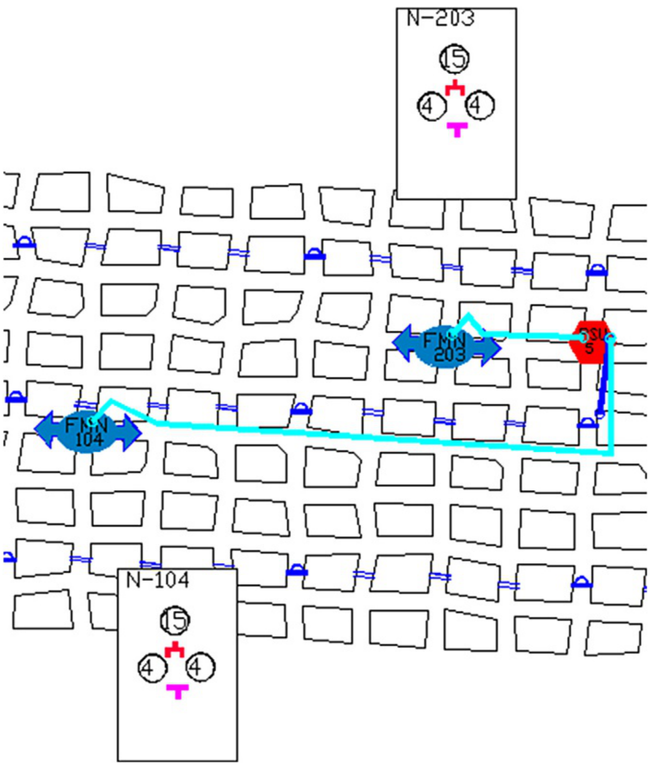


Fig. 8.12 Layout of primary EW node about 1600 ft inby the portal and the nearest secondary EW node.

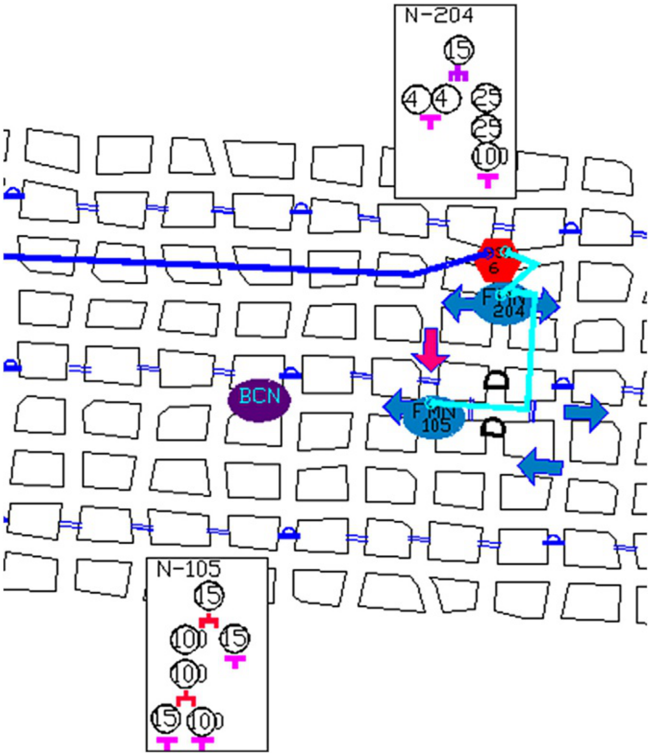


Fig. 8.13 Layout of primary EW node about 2400 ft inby the portal and the nearest secondary EW node.

signals passing through them. Antennas on the secondary EW are placed on either side of these air lock doors. These antennas are connected to FMN 202 by coaxial cables. This enables avoidance of signal attenuation due to the air lock doors.

Fig. 8.12 shows mine features in the area of FMNs deployed about 1600 ft inby the portal on the primary EW. The FMN on the secondary EW several crosscuts outby is also shown. These nodes provide network links in long straight sections of these entries. FMN 104 on the primary EW is 900 ft from the next FMN outby, FMN 103. FMN 203 is 700 ft from the next FMN outby, FMN 202, on the secondary EW.

Fig. 8.13 shows the FMN layout 2400 ft inby the portal on the primary EW. Of interest is the primary EW detour to Entry 4 for two crosscuts and then back to Entry 5. This detour goes around a pair of airlock doors providing vehicle access to the belt entry. Two related stoppings block Entry 5.

SCSR caches are designated as “strategic areas” by the operator. More accurate tracking is warranted in these areas, and accordingly a beacon is placed at the cache shown in Fig. 8.13. In the Test System, mobiles radios carried by mine personnel report receipt of signals from nearby tracking beacons via communications links afforded by FMNs.

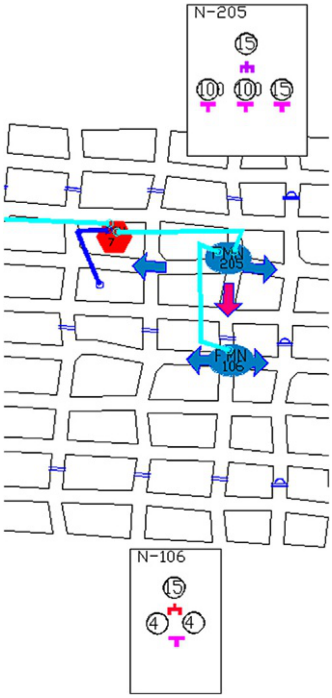


Fig. 8.14 Layout of primary EW node about 3200 ft inby the portal and the nearest secondary EW node.

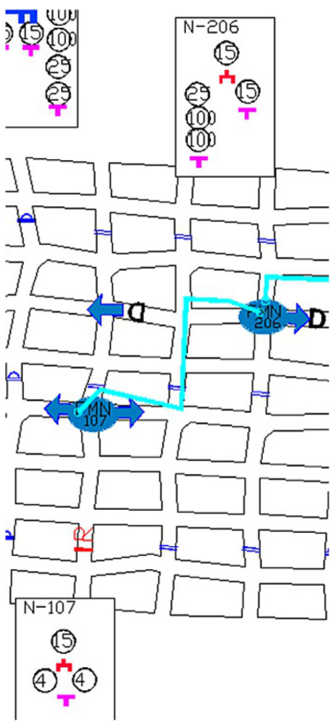


Fig. 8.15 Layout of primary EW node about 3850 ft inby the portal and the nearest secondary EW node.

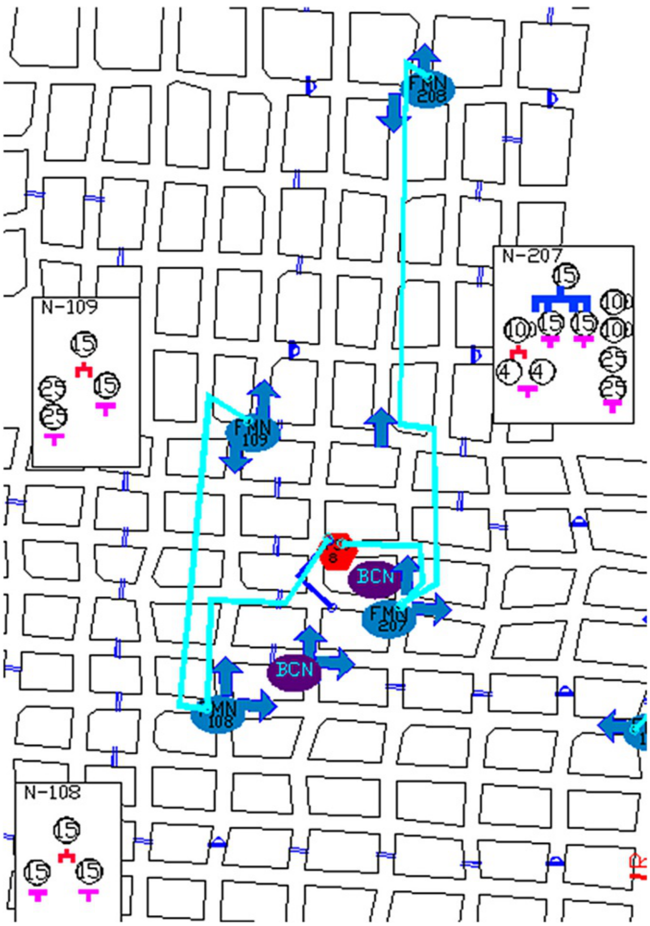


Fig. 8.16 EW node about 3850 ft inby the portal and the nearest secondary EW node.

Fig. 8.14 shows location details for a pair of FMNs in the area 3200 ft inby the portal on the primary EW. These nodes provide links in long straight tunnel segments with no major obstructions. Fig. 8.15 shows the FMN on the primary EW 3850 ft from the portal. The nearest FMN on the secondary EW, also shown, is located between a set of air lock doors. The inby pointing antenna associated with the node is positioned inby the airlock.

Fig. 8.16 shows the terminal portion of the EW test area. The 90 degrees corner of the primary EW is about 4400 ft inby the portal. There are two BCN locations in the area of the corner. The primary EW FMN is at the corner with two antennas directed at right angles, inby and outby. The secondary EW FMN at the corner has three antennas configured for 25%/25%/50% power split. The 50% portion is directed outby toward the airlock (backward "D" symbol at the lower right of the figure). One of the two 25%

portions is directed inby on the secondary EW, covering approaches to the SCSR cache and the associated BCN there; and the other 25% power portion antenna is placed in the belt entry enhancing coverage there, with the strategic area BCN located at the belt head.

The inby-most portion of the tracking test area has two FMNs at offsets in entry alignment resulting from change in pillar dimensions. These locations allow antenna placement that should assure signal past these offset corners. The inby-pointed antennas of these FMNs are estimated to extend coverage to well over 5000 ft inby the portal on both EWs.

Using the mine map as input, the network building utility was run to create a network model that can be used by COMMs to create field strength values that may be used by the tracking calculator. The tracking calculator values can then be used to show metric values in various zones. A majority of the network is automatically generated and a full version of the network is shown in Fig. 8.17 with a detailed view in Fig. 8.18. Area of the mine that consist of the ventilation return entries has been removed from the network, as shown in Fig. 8.19.

The reduced network is then used to place FMNs and BCNs as described in Fig. 8.9–8.16 layouts. An example of this is shown in Fig. 8.20. Connections between intersections have been changed to gray for display purposes. Intersections containing FMNs are shown in *light green* and links from intersections that contain directional antennas are depicted in *light green*.

After all nodes and beacons were entered into the model, areas receiving adequate signal for communications are drawn to show complete radio coverage of primary and secondary EWs (see *magenta lines* in Figs. 8.21–8.23). There are several links that are not covered, but intersections on both sides are covered. Each map shows an area that is expected to have degraded radio coverage, but this is not in the primary EW or along the belt. For each broadcast location, a database file is generated that shows all other locations in the mine and the maximum signal that is available in each of those locations. It also traces the path taken (thick magenta lines in Figs. 8.21–8.23). These database files are used in the next step.

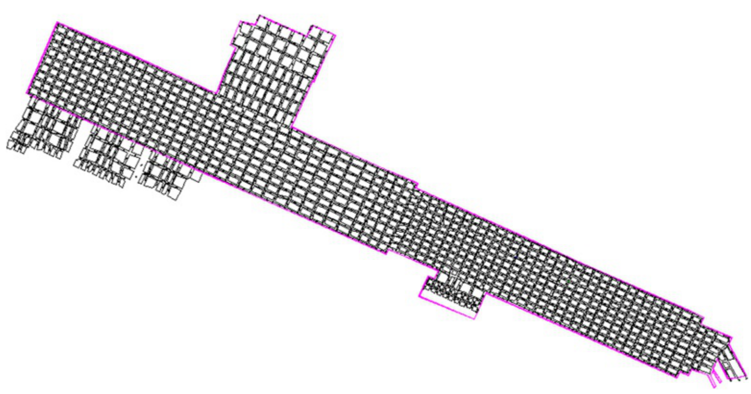


Fig. 8.17 Isolated mains area of Example Mine with COMMs network.

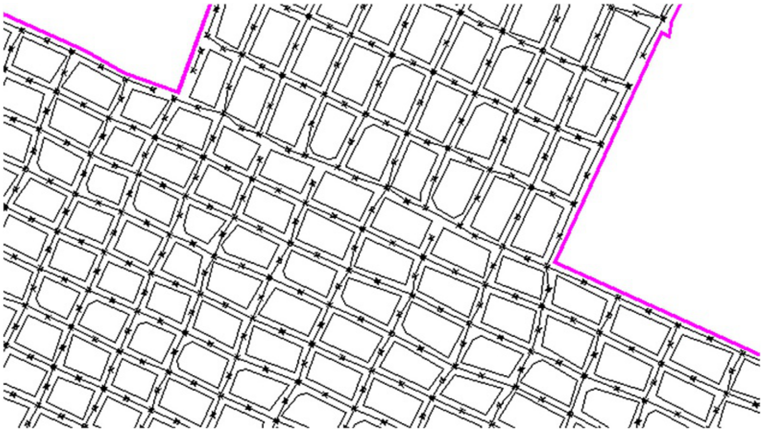


Fig. 8.18 Detailed area of the COMMs network.

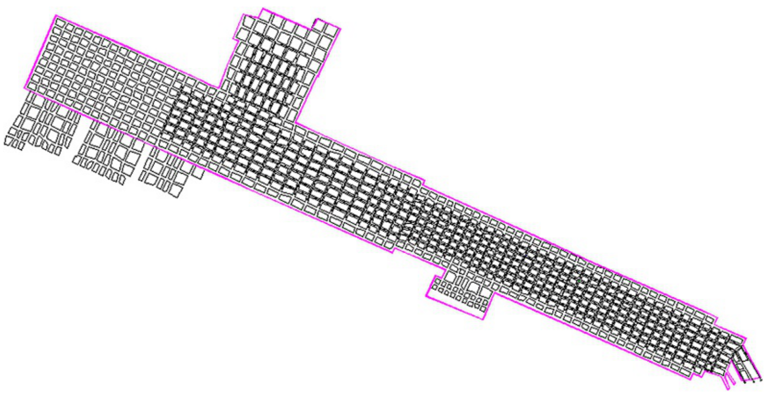


Fig. 8.19 Reduced COMMs network.

Locations along primary and secondary EWs were selected as inputs into the tracking simulation. This simulation interrogates the signal strength that is available from all signal sources in the model. These values were calculated in the previous step as well as a variation value in the form of a confidence interval. This average value and confidence interval are used by a pseudorandom number generator that is capable of outputting numbers that meet a prescribed statistical model. In this case, a uniform distribution random model was picked and a sufficient number of random values were made to show the randomness of the system, 250. COMMs will also output the header and position files that are used by the tracking simulator. The tracking simulation will then output a coordinate in *X*,*Y* pairs that are in mine coordinates. This means that for an intersection in the mine (GTP), 250 signal strength estimates generate 250

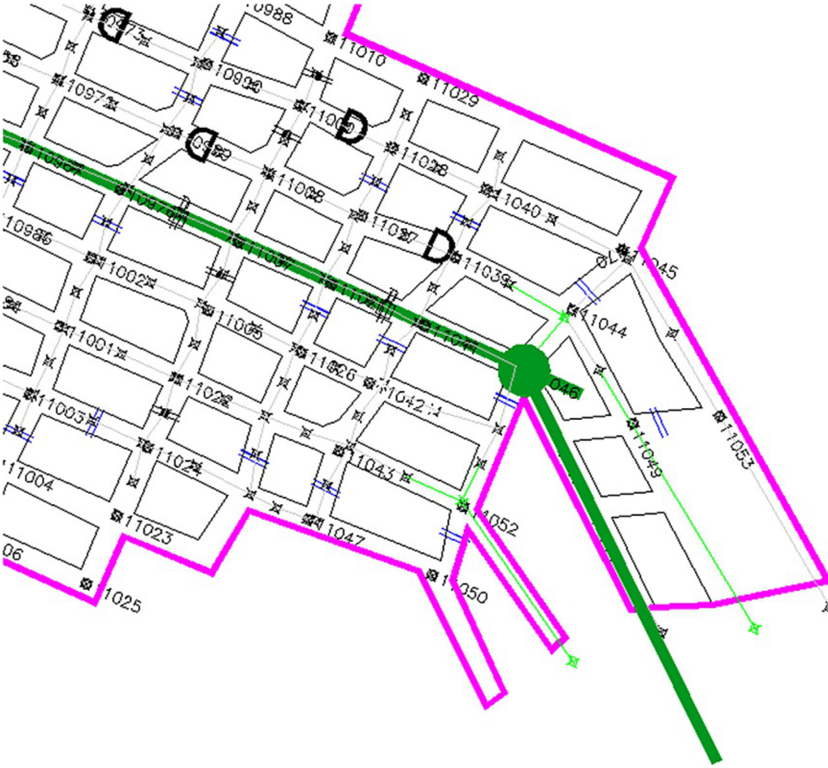


Fig. 8.20 Example layout including antenna directions.

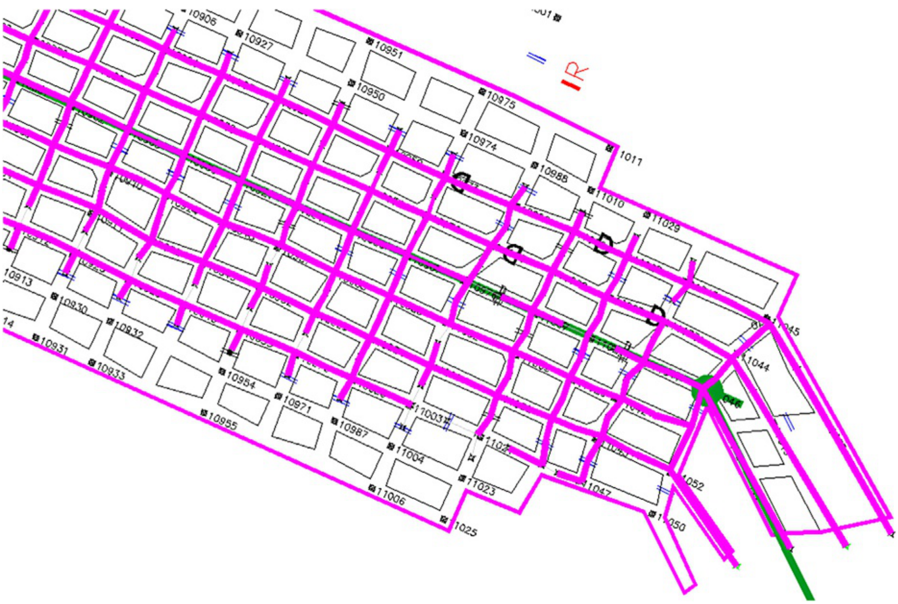


Fig. 8.21 Radio coverage—Map 1 of 3.

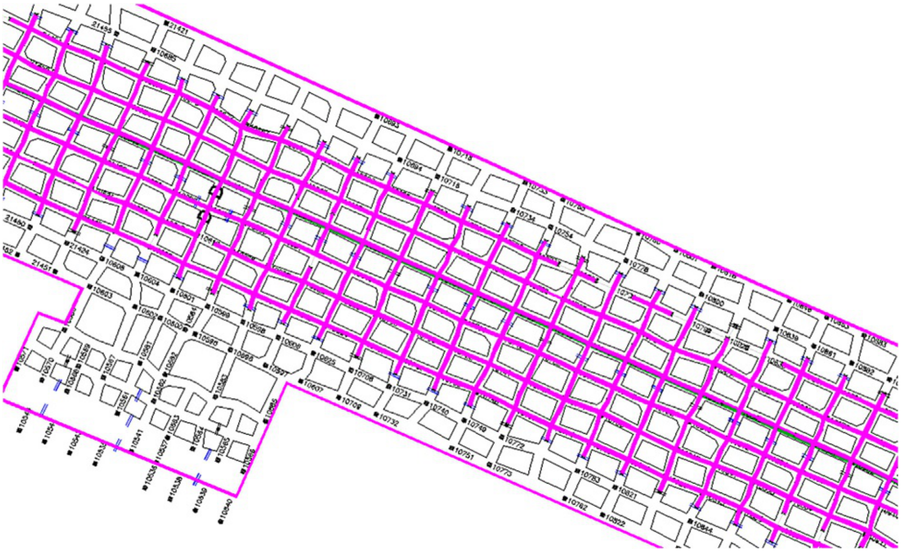


Fig. 8.22 Radio coverage—Map 2 of 3.

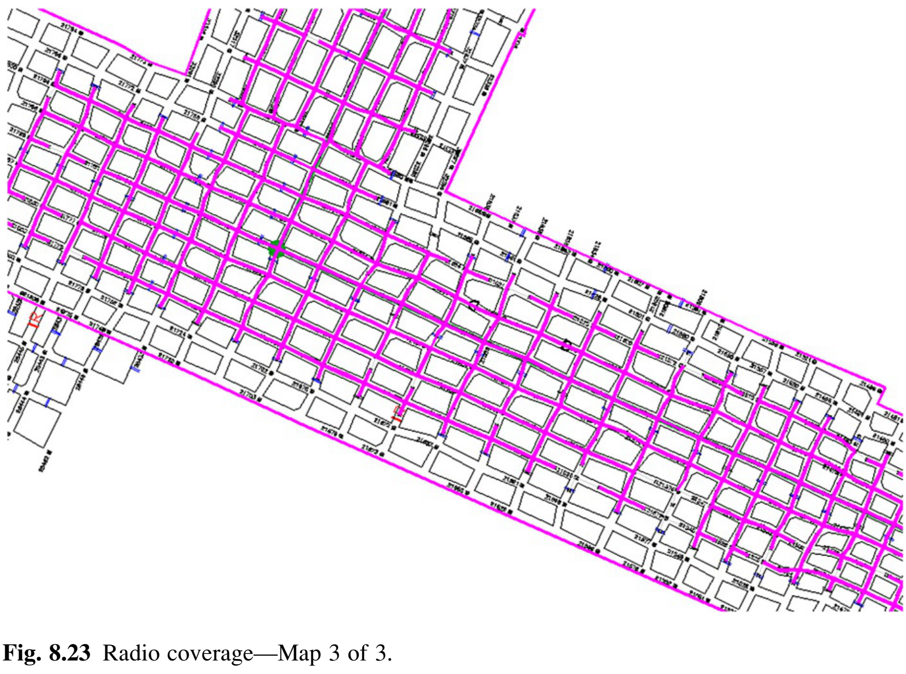


Fig. 8.23 Radio coverage—Map 3 of 3.

coordinates (TSPs). These values are used in subsequent calculations. Estimates represent the answer from the model; they are not inputs to the design. If values are unacceptable, areas of the mine must be examined to modify tracking accuracy. This process is consistent with the current standard practice.

8.3.2 Metric values in example layout

Following are examples of metric calculations to describe the tracking system as installed. Some metrics are not described because they are not predicted in this analysis. For instance, latency will not be described. The scenario-based metrics—reliability, availability, susceptibility, and robustness—are not described. For all of the following metrics, areas of the mine are modeled. For instance, all of the intersections along the primary EW are listed out as the GTPs of interest. For each of these GTPs, 250 TSPs are generated. These TSPs are used in the calculations.

For the static testing area in the test mine, the active TCA is highlighted *blue* in Fig. 8.2. The Inferred TCA will sporadically occur in the entries adjacent to the escapeways as shown in Figs. 8.21–8.23. No escapeway inside the test area will have a tracking error greater than 2000 ft and no strategic area will have a tracking error greater than 200 ft. Therefore, this FMN configuration will mean that the static testing area in test mine will be included in the compliant TCA. The compliant TCA is the area inside the TCA where tracking quality guidelines are met.

For the primary EW in the Test Mine, it is expected that the instantaneous accuracy (IA) of measurements ranges from 1.14 (nearly perfect IA) to 1237 ft. For the secondary EW, IA is calculated to range from 1.33 to 985 ft. The beltline is covered, but in spite of its much higher attenuation, rendering lower signal strengths, the IA range from 1.25 to 896 ft remains comparable to that of the escapeways.

Based on this estimated installation of Test System equipment, the simulation predicts the primary EW in Test Mine will have an AA of 267 ft and AA will be 334 ft in the secondary EW. The calculated AA in the beltway is 383 ft.

The primary EW simulation IAs have an SDA of 248 ft and the secondary EW is comparable with SDA of 217 ft. The beltway SDA is 221 ft.

Figs. 8.24–8.26 graphically show the average error vector and the average cluster radius for the three areas of interest in the mine. The primary EW has an ACR of 94 ft with an AEV of $\{212,-26\}$. The secondary EW has an ACR of 73 ft with an AEV of $\{152,-7\}$. The belt entry has an ACR of 89 ft with an AEV of $\{95,-25\}$. These AEVs are consistent with the angle of the mine, meaning the tracking system is calculating the TSPE in the correct entry, but the distance inby is variable. The linearity of the primary and secondary EWs is therefore expected. The belt entry shows a greater spread, and this can also be anticipated because the entry does not contain any transmitter equipment. Therefore, some TSPEs tend to be drawn to the primary and secondary EWs where the signal is stronger, located in this mine, respectively, on either side of the belt entry. This is the value of two-dimensional metrics; they show bias in the system that can be engineered out by changing antenna and node locations.

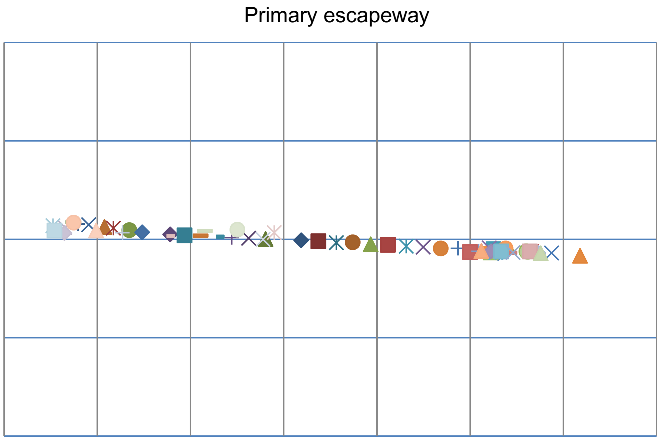


Fig. 8.24 Average TSPE plot along primary EW.

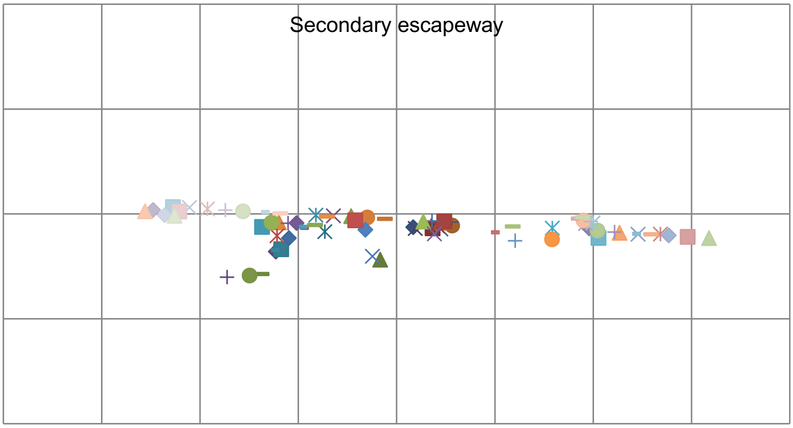


Fig. 8.25 Average TSPE plot along secondary EW.

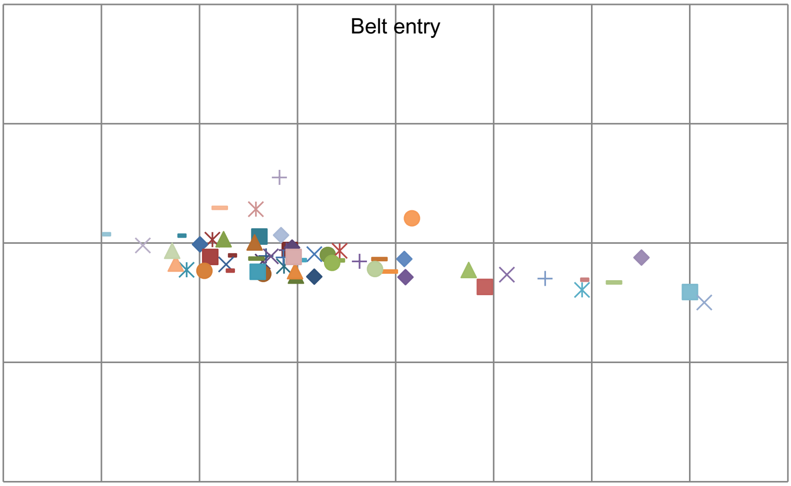


Fig. 8.26 Average TSPE plot along belt entry.

The thousands of values calculated are filtered by area and then ordered by IA. Because of the large number of samples, the 90th percentile is found at an IA. These simulations predict a 90% CD of 185 ft in the primary EW, 162 ft in the secondary EW, and 254 ft within the belt entry.

8.4 Measurement of tracking system performance

The tracking system described earlier was installed with care taken to keep infrastructure devices in simulated locations. Data were collected over the course of several months and several surveys to compare with simulation results. Several tests were designed to isolate specific effects that may impact the tracking system. In addition, the Test System was installed in the Example Mine on behalf of the research project and is the secondary system, allowing the research team to modify the system. The system is not installed to the working face in the Example Mine. The most inby node is located at a turn in the main entries.

8.4.1 Measurement of tracking system variations when stationary

In order to gain an understanding of the variation in the tracking system with the least number of perturbations, a device was hung from a roof bolt on 10-25 from 11:44:00 until 12:47:57. The device was hung at the location 1907839, 350827. Measured results at this location were: AA of 99.5, SDA of 19.4, AEV of $\langle 40,-4 \rangle$, and ACR of 84. Selected records from the tracking database are in Table 8.2 with a map of the GTP and TSPs in Fig. 8.27.

Next, the device was hung from a roof bolt on 10-26 from 10:15:54 until 11:57:00. The device was hung at the location 1905962, 350799. Measured results at this location were: AA of 79.7, SDA of 33.6, AEV of $\langle -46,3 \rangle$, and ACR of 48.8. Selected records from the tracking database are in Table 8.3 with a map of the GTP and TSPs in Fig. 8.28.

The handset hung in the secondary EW is compared to the prediction values for that same location. Fig. 8.29 shows locations calculated by the tracking system in *red* and predicted values in *black*. The actual location of the handset is circled in *red* and the prediction location is a *blue point*. Tracking system locations are taken from surveys performed in the area, not including values from the stationary handset test described earlier. TSP values in the following figures and tables are taken from surveys conducted with the survey buggy, described later. Figures indicate that prediction values describe the same sort of distribution, but are trending more toward the other escapeway than calculated values. Table 8.4 shows metric values for this single location. AA, SDA, 90% CD, and ACR values from the prediction and the measured are within an acceptable range. AEV describes the predicted values to be further toward the primary EW.

The stationary handset in the primary EW (see Fig. 8.30) was placed at the location circled in *red*. *Red points* are calculated by the tracking system and *black points* are predicted points. The clustering of locations in this escapeway is the opposite of

Table 8.2 Stationary handset test data for Fig. 8.27

Time inserted	Reported time	TSP	IA	Time inserted	Reported time	TSP	IA
11:43:46	11:44:07	1908018,350816	179	12:23:27	12:25:17	1907943,350820	104
11:44:07	11:44:16	1908035,350813	196	12:25:17	12:25:37	1907930,350820	91
11:44:16	11:44:26	1907985,350816	146	12:25:37	12:25:47	1907943,350820	104
11:44:26	11:44:37	1907933,350820	94	12:25:47	12:25:57	1907920,350820	81
11:44:37	11:44:46	1907913,350820	74	12:25:57	12:26:37	1907939,350820	100
11:44:46	11:44:56	1907703,350833	136	12:26:37	12:26:47	1907752,350829	87
12:12:47	12:12:57	1907739,350829	100	12:26:47	12:27:17	1907943,350820	104
12:12:57	12:13:47	1907946,350820	107	12:27:17	12:27:27	1907733,350829	106
12:13:47	12:13:57	1907739,350829	100	12:27:27	12:27:57	1907939,350820	100
12:16:17	12:17:47	1907939,350820	100	12:27:57	12:28:47	1907953,350820	114
12:20:57	12:21:07	1907930,350820	91	12:28:47	12:29:07	1907939,350820	100
12:21:07	12:21:17	1907949,350820	110	12:35:17	12:35:27	1907907,350820	68
12:21:17	12:22:37	1907936,350820	97	12:35:27	12:35:47	1907933,350820	94
12:22:37	12:22:57	1907920,350820	81	12:35:47	12:37:17	1907946,350820	107
12:22:57	12:23:07	1907746,350829	93	12:37:17	12:37:27	1907959,350820	120
12:23:07	12:23:17	1907936,350820	97	12:37:27	12:37:57	1907933,350820	94
12:23:17	12:23:27	1907916,350820	77	12:46:47	12:48:17	1907943,350820	104

Fig. 8.27 Map of stationary handset test.

predicted values. Although the prediction does show the same sort of distribution, it is trending in the opposite direction.

Calculating metrics for this point shows errors that are apparent in Fig. 8.30. Table 8.5 shows calculated values. AEV clearly shows the major difference in cluster locations. Also, predicted values have much more variation and are an average distance further away. Most important, the 90% CD is much better than the predicted value. The prediction value is much worse than the measured value, but it well within the 2000-ft guideline established by MSHA.

In order to determine the cause of the difference between predicted values and measured values, two other specific points were investigated in the primary EW. Location 844, shown in *red* in Fig. 8.31, was drawn with *red points* for the tracking system's locations and *black* for predicted points. Metric values for this location are in Table 8.6 and show the same variation between predicted and measured values. Again, the predicted cluster is much further outby from the measured cluster. This location is also showing a greater pull toward the other escapeway.

In the last two locations, there are two factors not adequately accounted for in simulations and predictions. Near the location of the personnel door to the right of the circled location in Fig. 8.31 is a significant topographical change in the coal seam. Within two crosscuts outby that location is a reversal of elevation from floor and roof. This roll is completed to the right of the *red circle* in Fig. 8.30, but in the opposite direction. Accurate elevation and thickness data were not available at the time of the simulation and predictions were not entered into the model.

Fig. 8.32 shows Location 199 circled in *red with black points* for prediction values and *red points* for tracking system calculations. Table 8.7 gives metric calculations for the location shown in Fig. 8.32. They are in acceptable agreement. As seen in the secondary EW, AEV and ACR are not in complete agreement, but the other metric values are suitably close. This location is parallel to the static handset location in the secondary EW. At this place in the mine, there is consistent height and the topography is consistent.

Table 8.3 **Stationary handset test data for Fig. 8.28**

Time inserted	Reported time	TSP	IA	Time inserted	Reported time	TSP	IA
10:15:43	10:16:03	1905787,350806	175	11:52:04	11:52:14	1905869,350803	93
10:16:03	10:16:13	1905754,350806	208	11:52:14	11:53:04	1905886,350800	76
10:16:13	10:16:33	1905781,350806	181	11:53:04	11:53:14	1905869,350803	93
10:16:33	10:16:43	1905810,350803	152	11:53:14	11:53:34	1905892,350800	70
10:16:43	10:16:53	1905751,350806	211	11:53:34	11:54:14	1905873,350803	89
10:16:53	10:17:03	1905899,350800	63	11:54:14	11:54:24	1905918,350800	44
10:17:03	10:17:13	1905925,350820	42	11:54:24	11:55:04	1905905,350800	57
10:17:13	10:17:33	1905886,350800	76	11:55:04	11:55:24	1905889,350800	73
10:17:33	10:17:43	1905892,350816	72	11:55:24	11:55:34	1905895,350816	69
10:17:43	10:17:53	1905889,350800	73	11:55:34	11:55:54	1905909,350800	53
10:17:53	10:18:13	1905912,350800	50	11:55:54	11:56:04	1905892,350816	72
10:18:13	10:19:13	1905886,350800	76	11:56:04	11:56:14	1905895,350800	67
10:19:13	10:19:23	1905922,350800	40	11:56:14	11:56:24	1905882,350800	80
10:19:23	10:19:33	1905869,350803	93	11:56:24	11:56:34	1905863,350803	99
10:19:33	10:20:33	1905882,350800	80	11:56:34	11:56:44	1905846,350803	116
10:20:33	10:21:13	1905909,350800	53	11:56:44	11:56:54	1905918,350800	44
10:21:13	10:21:23	1905935,350800	27	11:56:54	11:57:04	1905935,350800	27

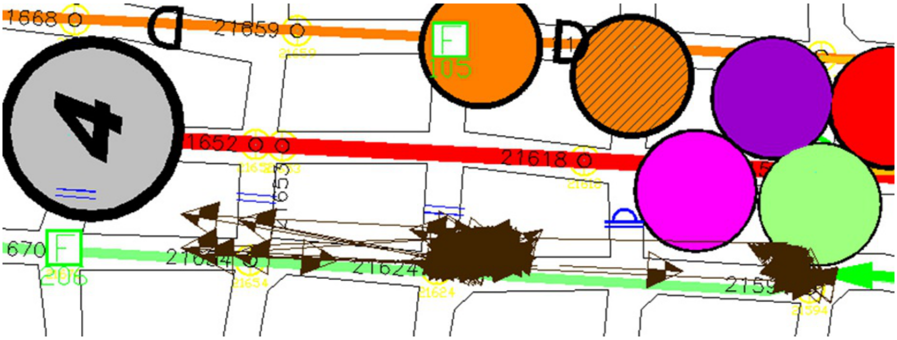


Fig. 8.28 Detailed map of stationary handset test.

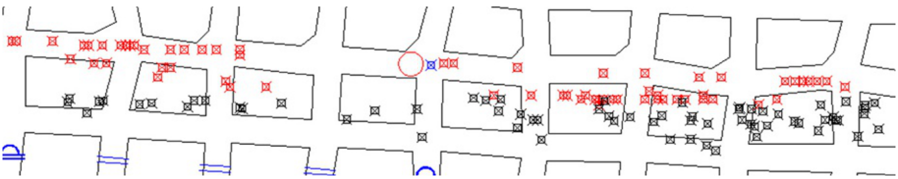


Fig. 8.29 TSP and predicted TSP for stationary handset in the secondary EW.

Table 8.4 Predicted and measured metrics for secondary EW locations in Fig. 8.29

Location	Type	AA	SDA	90% CD	AEV	ACR
193	Predicted	241	105	376	⟨178,−36⟩	154
193	Measured	275	162	421	⟨28,−9⟩	180

Fig. 8.30 TSP and predicted TSP for stationary handset in the primary EW.

Table 8.5 Predicted and measured metrics for the primary EW location in Fig. 8.30

Location	Type	AA	SDA	90% CD	AEV	ACR
729	Primary	412	165	547	⟨400,−2⟩	149
729	Primary	157	69	233	⟨−25,16⟩	154

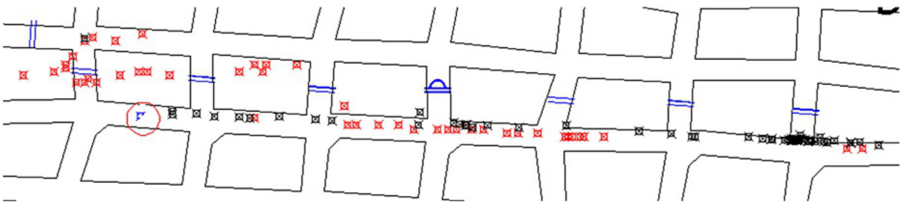


Fig. 8.31 TSP and predicted TSP for a location in the primary EW.

Table 8.6 Predicted and measured metrics for the primary EW location in Fig. 8.31

Location	Type	AA	SDA	90% CD	AEV	ACR
844	Primary	402	215	550	⟨220,19⟩	251
844	Primary	193	125	370	⟨146,13⟩	140

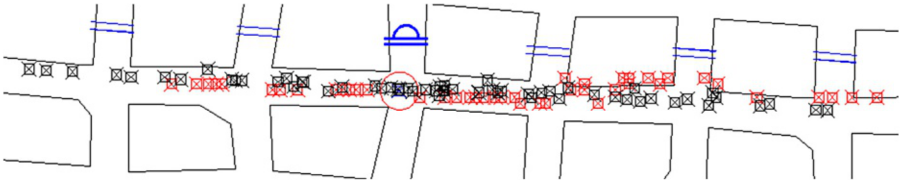


Fig. 8.32 TSP and predicted TSP for a location in the primary EW.

Table 8.7 Predicted and measured metrics for the primary EW location in Fig. 8.32

Location	Type	AA	SDA	90% CD	AEV	ACR
199	Primary	119	99	336	⟨−6,2⟩	119
199	Primary	103	66	220	⟨59,1⟩	64

Stationary handset tests were performed in order to understand the variation in calculations of the tracking system's location under static conditions. The data show that there is a general variation of roughly 200 ft in a direction following the escapeway, or 400 ft in either direction. The 400 ft of variation is still 1/5 of the 2000-ft guideline set by MSHA. For both primary and secondary EW, a tracked device can be located in the general area of its actual location. A description from the tracking system could be relayed to a miner underground and at walking speed they would have encountered the tracked device within minutes, assuming a search along the escapeway.

8.4.2 Measurement of tracking system variations when in motion

Measurements were taken using a SkyMark Survey Buggy. The device is a dead reckoning tool, which, when taken to a location underground, is capable of tracking its location and recording various sensor data with accurate time stamps. The tracking system does not communicate a device's location with the tracked device, but the tracked device locations are logged at the main tracking system computer. The buggy is used to calculate the GTP of a set of handsets. Fig. 8.33 shows the general configuration of the cart while being towed. Radios are rotated around to be as far away from the tow vehicle and as high in the mine as possible without hitting the roof. Handsets are arranged in two orientations with four held vertically and two held horizontally.

The survey buggy records a single time and a location, while the tracking system records a calculated location and two times. Tracking system times are the time that a tracked device entered into a state or location and the last time it reported being at that location. Handsets report the communication infrastructure RSSI at a predetermined interval. This report is used by the tracking system computer to calculate the position. The survey buggy records the GTP by a survey number. Surveys are numbered by the operator; each survey was conducted to either measure values in an area or to conduct a specific test.

In this chapter, only surveys that are measuring areas are used. Several surveys were conducted in areas of the mine and then the buggy was taken outside. When the area of interest was exited, the record continued. Recorded values that did not have the operator actively working the survey buggy are excluded. Data collected from these surveys are used in stationary handset discussions.

Predicted values are created at specific locations in the mine. In order to compare these values to continuous measurements done by the survey buggy, continuous measurements were filtered. Both values are in the mine's coordinate system, so discrete points of the simulation were used as anchor points. Any GTP from the cart within

Fig. 8.33 General test configuration while towing.

15 ft of an anchor point was selected from data sets to be used. For each of these GTPs, there is an associated time it was recorded. Also with this time is a set of handsets that were located on the buggy. For each time and handset, TSP values are queried from the tracking database. This method of comparing may cause inconsistency with metric values reported for the simulation-only data earlier in this chapter; however, for the purposes of comparison, data comparing measured and predicted results will only include data points that meet the criteria described here.

An example of these surveys is Survey 148, which was a survey of the secondary EW. Fig. 8.34 is a map of this survey. The *red line* is the path traveled as reported by the survey buggy. Spads are drawn as *yellow blocks* and are included as reference. *Green arrows* are drawn every one minute of survey time from the GTP to the TSP for each device.

Metric values for this particular survey are shown in Table 8.8. The device is the identification number for the individual radio. GTPs are recorded at a very high density since they are logged every time the buggy changes location, which can be several data points per foot; the total number recorded is shown in the count column. GTP is calculated as part of the survey buggy software. AA is calculated by taking the arithmetic average of all IAs as calculated from each recorded GTP. SDA is calculated in the same manner, except it is the standard deviation by population. 90% CD is calculated by sorting distinct IA values into percentile ranks, the 90th percentile rank is the cutoff value that is reported. Averaging the difference in X and Y coordinates between GTP and TSP yields AEV. The GTP plus the AEV yields the center of the TSP spread. The average of the distance of a TSP from the center of the TSP spread is the ACR. ACR is only calculated from a TSP that is not at the center of the TSP spread and distinct

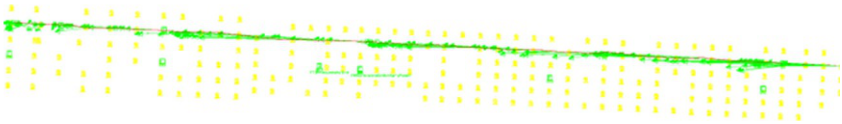


Fig. 8.34 Survey 148.

Table 8.8 Metrics for survey 148

Device	AA	SDA	90% CD	Count	AEV	ACR (count)	ΔGTP
9537	189	27	226	1978	⟨160,5⟩	55 (168)	93
9548	207	26	243	1983	⟨193,−2⟩	48 (173)	93
9593	282	29	322	1976	⟨278,−9⟩	46 (157)	99
9647	321	29	361	2256	⟨316,−12⟩	44 (158)	100
9775	249	26	285	2198	⟨244,−6⟩	42 (179)	88

values to avoid weighting. The count of TSPs used in the ACR is shown in parentheses. This is generally the number of times the tracking system calculated a location during the survey. Delta GTP is the straight-line distance from minimum GTP to maximum GTP. It is not the distance the survey buggy traveled, rather it is the diagonal distance of the bounding box area. For example, the buggy could be used to cover a 5-ft area 1000 times for a distance traveled of 5000 ft, but a Delta GTP of only 5.

Prediction points are at discrete locations, as described earlier. GTPs from surveys were queried to find points that were within 15 ft from the predicted TSP point and compared favorably to measured TSP results. This is shown in Fig. 8.35 where *green points* are GTPs collected by the buggy for surveys included in the data set that are within 15 ft of Location 680. *Red points* are TSPs and *black points* are predicted TSPs. The 15-ft distance was chosen because the average intersection interval is 100 ft and there are locations at halfway points between intersections, such as 680, or roughly every 50 ft. A radius of 15 ft yields a total travel distance of 30 ft along the escapeway, with a sufficient buffer to prevent double counting a GTP.

Belt entries and entries away from escapeways are areas that do not need to be covered by the tracking system and that increase the variability. This is shown by two surveys, 1127 and 1130, that were conducted in the secondary EW of the mine. Survey 1127 is shown in Fig. 8.36 and Survey 1130 was conducted perpendicular to survey 1127 and is shown in Fig. 8.37.

As part of the overall study, 21 surveys were conducted from September to February. These surveys have over 800,000 data points after filtering by node point. Randomly selected locations, with their predicted values and measured values, are presented in Tables 8.9 and 8.10. Values are calculated as described in the description of Table 8.8, but count values have been excluded for ease of formatting. In general, there are 2–4 times more measured values at any particular location than there are predicted values.

Table 8.11 summarizes predicted and measured results by only the one-dimensional values AA, SDA, and 90% CD. It shows that predicted values are higher than measured values, but they are descriptive of the same system because both values

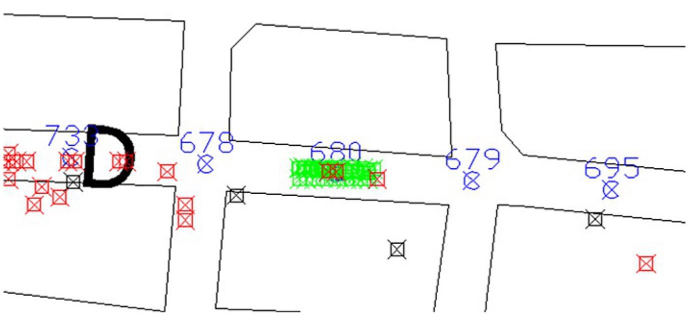


Fig. 8.35 GTP, TSP, and predicted TSP for Location 680.

Fig. 8.36 Survey 1127.

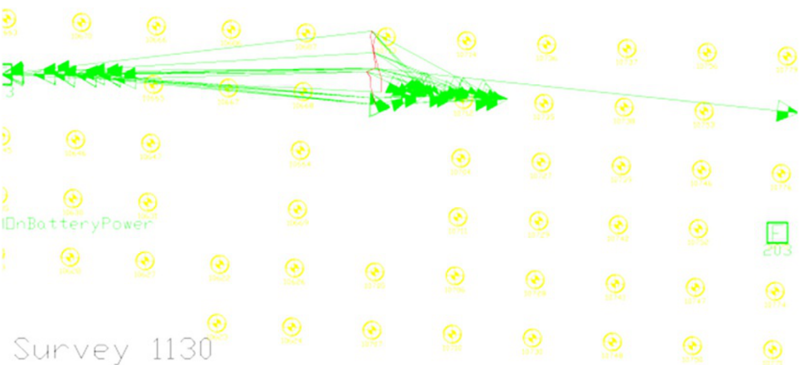


Fig. 8.37 Survey 1130.

are significantly lower than target values for the tracking system. Along the entirety of escapeways, the 90% CD should be less than 2000 ft and the average of both escapeways is one quarter of this target. Both predicted and measured AA describe a tracked device's location within two crosscuts with two crosscuts of potential error (SDA). A crosscut along the travel way (100 ft) can be crawled in less than a minute (34 s assuming 3 ft/s crawling speed), meaning a potential search area of 400 ft can be slowly traversed by a rescue team in 2–3 min. These prediction results are only for prediction locations that were visited by the survey buggy during one of the surveys in the data set. This may lead to some inconsistencies with previously reported results; however, values are of a consistent order of magnitude and indicate the importance of the data set that is used to calculate these metric values.

Table 8.9 **Selected GTP and TSP with prediction values for the primary EW**

Location	Predicted					Measured				
	AA	SDA	90% CD	AEV	ACR	AA	SDA	90% CD	AEV	ACR
13	149	128	609	$\langle 14,-1\rangle$	152	180	94	292	$\langle 143,-7\rangle$	136
17	151	101	539	$\langle 44,5\rangle$	147	207	114	341	$\langle 159,-6\rangle$	165
24	150	132	643	$\langle 30,0\rangle$	150	157	85	268	$\langle 133,-6\rangle$	97
26	231	131	672	$\langle -133,17\rangle$	170	130	77	231	$\langle 129,-5\rangle$	88
121	116	98	508	$\langle 29,0\rangle$	116	219	138	378	$\langle 180,-6\rangle$	181
199	119	99	336	$\langle -6,2\rangle$	119	103	66	220	$\langle 59,1\rangle$	64
293	212	114	395	$\langle -161,11\rangle$	146	152	45	206	$\langle -37,14\rangle$	146
316	208	135	470	$\langle 12,0\rangle$	205	222	118	412	$\langle -53,7\rangle$	239
362	143	99	334	$\langle 23,5\rangle$	139	244	98	420	$\langle -72,5\rangle$	231
427	90	65	224	$\langle 44,9\rangle$	83	123	114	291	$\langle -128,12\rangle$	99
601	154	71	371	$\langle -153,19\rangle$	55	215	109	295	$\langle 108,10\rangle$	281
702	115	88	287	$\langle 53,5\rangle$	112	116	55	181	$\langle -35,12\rangle$	109
734	407	132	475	$\langle 373,-11\rangle$	141	223	74	326	$\langle 27,12\rangle$	202
844	402	215	550	$\langle 220,19\rangle$	251	193	125	370	$\langle 146,13\rangle$	140
1002	59	11	77	$\langle 50,-18\rangle$	25	146	32	179	$\langle -16,-139\rangle$	28

Table 8.10 Selected GTP and TSP with prediction values for the secondary EW

Location	Predicted					Measured				
	AA	SDA	90% CD	AEV	ACR	AA	SDA	90% CD	AEV	ACR
4	221	116	397	⟨−55,−5⟩	209	263	159	392	⟨−71,4⟩	172
62	264	163	525	⟨−180,−19⟩	193	336	187	666	⟨−183,11⟩	280
75	129	57	263	⟨−106,−6⟩	70	284	212	585	⟨198,−15⟩	278
129	218	141	480	⟨−37,−36⟩	205	329	205	573	⟨207,−16⟩	257
213	180	72	334	⟨126,−57⟩	106	311	149	547	⟨131,−13⟩	80
266	116	93	326	⟨56,−9⟩	96	306	360	453	⟨−43,5⟩	200
325	115	66	241	⟨82,−34⟩	74	382	262	386	⟨97,−17⟩	237
365	123	74	248	⟨68,−20⟩	102	302	283	438	⟨−57,−10⟩	208
406	80	52	325	⟨21,−19⟩	70	317	325	459	⟨−155,0⟩	131
517	206	114	381	⟨123,−36⟩	163	376	139	563	⟨−140,4⟩	320
579	203	122	362	⟨9,−17⟩	200	319	141	405	⟨−108,9⟩	295
620	89	60	218	⟨20,−20⟩	85	136	109	297	⟨−18,−4⟩	153
679	284	139	493	⟨236,−47⟩	152	133	64	204	⟨−31,−6⟩	129
733	65	46	199	⟨29,−13⟩	56	98	107	261	⟨47,−13⟩	132
770	517	156	550	⟨−4,−63⟩	176	218	96	473	⟨26,31⟩	52
839	462	57	466	⟨451,−53⟩	34	230	142	466	⟨133,5⟩	223

Table 8.11 **Summary of predicted and measured results**

		Predicted	Measured
Primary	AA	221	170
	SDA	193	112
	90% CD	605	420
Secondary	AA	209	158
	SDA	205	126
	90% CD	504	601

8.5 Conclusions

This chapter describes a realistic and simulated deployment of a communications and tracking system with complete analysis of the system's performance. There are 13 FMNs underground, 2 FMNs above ground, and three BCNs located at strategic areas. Using anticipated loss parameters for the Test System radio signals, fresh air and belt air entries of the mine are simulated in order to calculate anticipated performance metrics. Several dozen physical tests of the designed and described Test System are compared to predicted results.

Predicted results proved to be higher in most metric values that are measured; however, measured results are collected by a device that is continually measuring locations, but predictions are from single point values. A comparison technique was used to gather all data collected in both prediction and measurement cases. Variations in this comparison technique have a large impact on measurements. This is especially true when a survey was conducted for a long period of time in a small area. Values collected during that time will have a greater impact on the overall average because there are more of them, than in an area that was visited less or for a shorter period of time. This is further complicated by the internal reporting intervals of the tracking system. Many systems report the current location of a tracked device and the duration at that location. They do not report the number of times the device was reported to be at that location. A smoothing algorithm that is easy to understand needs to be developed to solve the time and location weighting problems that can be caused during measurements.

This chapter describes a method of predicting the performance of a tracking system that is in line with the observed performance. More importantly, it demonstrates that the predicted measures and observed values using standardized metrics do describe the same tracking system because general values and trends are correct. The magnitude of changes can be adjusted with simulation input parameters, changing values to be more specific to that location.

Communication and tracking systems have rapidly become critical to operations at the most efficient mines in the United States. The Test System presented is significantly smaller than the typical system found in a producing mine; however, the techniques for planning, verification, and optimization of tracking systems described in this chapter can be utilized for any size tracking system. While communication and tracking systems are a significant cost to any mining operation, with the proper maintenance and planning they can pay for themselves in productivity gains.

Acknowledgments

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Out-of-seam dilution: Economic impacts and control strategies

9

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9.1 Introduction and background

9.1.1 Out-of-seam dilution defined

Out-of-seam dilution (OSD) results when strata above and/or below the coal seam are intentionally or unintentionally included in the run-of-mine (ROM) product. Some coal seams contain rock partings, the mining of which generates in-seam dilution. With few exceptions, parting and coal seam thicknesses rule out selective mining that does not extract in-seam dilution, so it is generally considered a natural part of ROM product. Extraction of OSD, however, can be avoided if sufficient care is taken in the mining process.

The industry's understandable focus on improving productivity [1] has led to more powerful machinery that advances at faster rates often without concern for increasing levels of OSD being extracted. The three most common reasons given for not paying attention to OSD are as follows: (a) The more powerful machinery is larger and needs a certain height to operate in, which is more than the coal seam height; (b) the immediate roof strata are often weak and need to be removed to prevent them from falling and causing injury; and (c) all of the ROM product will be processed in the preparation plant, which is fully capable of rejecting OSD. While these are the reasons given by mine operators for extracting out-of-seam material, they do not match what is actually observed in mines; for example, mining height often exceeds required equipment height by as much as 1.0 ft (0.3 m) or as much as 10 in. (0.25 m) of dilution being removed from the floor.

There are legitimate reasons for extracting out-of-seam material, such as grading belt entries for proper belt alignment and cutting overcasts and undercasts to provide ventilation; however, these necessary excavations can often be managed such that dilution material is placed underground and not sent out of the mine with the regular product. Thus, while each mining operation has unique conditions that justify producing some OSD, in most cases, there are tremendous opportunities to significantly improve mine profitability through moderately conservative reductions in OSD.

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9.1.2 Impacts of dilution

As shown in Fig. 9.1, dilution has a pervasive impact on each step of the coal production process. Not only does OSD have an obvious impact on face production costs due to extracting extra material, but also it negatively impacts the cost of all downstream processes. Roof rock is typically much harder than coal and mining; it results in higher power costs and increased maintenance and wear on extraction equipment. Materials handling costs are affected by the dilution material, which decreases conveyor belt life and increases power costs. Processing costs are also impacted as a result of the added waste material flowing through the processing plant. The OSD material is eventually rejected, where it adds to the costs of constructing, maintaining, and reclaiming a suitable waste disposal area.

Some less obvious, but nevertheless significant, impacts of dilution on coal processing are related to clays removed from the mine floor and to carbonaceous black shale removed from the mine roof. Floor clay impacts the processing system in the following three ways:

1. It completely breaks down in solution leading to increases in the viscosity of the medium in dense media circuits. This might improve media stability at the low end of the gravity spectrum; however, for the more common higher separation gravities, increased viscosity is detrimental as it decreases separation efficiency leading to lower clean coal quality.
2. Increased clay implies higher thickening and dewatering cost along with correspondingly higher flocculant costs for water clarification.
3. Higher clay content impairs the ability of flotation cells to clean and dewater ultrafine coal, which can be as much as 5% of raw coal production.

Another major impact of OSD, which is frequently ignored because it is not very well understood, is its influence on clean coal quality. If preparation plants were 100%

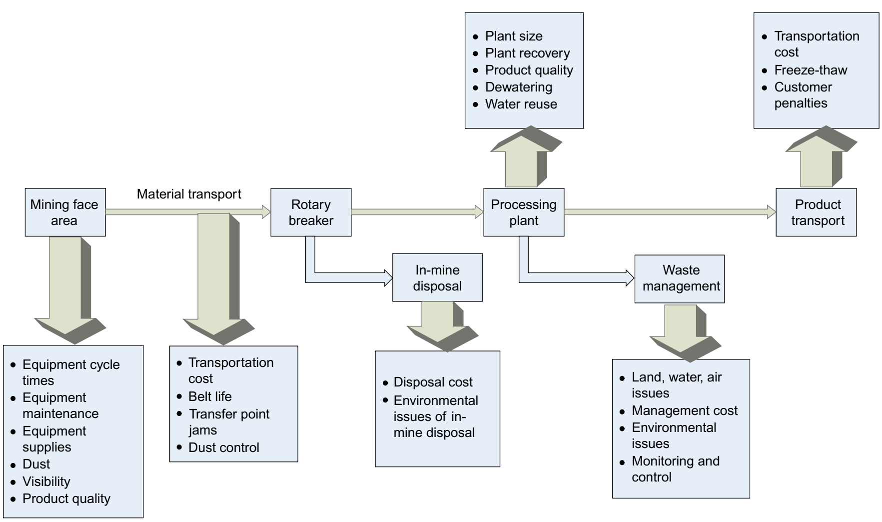

Fig. 9.1 Impact of dilution on various components of the coal-mining system [2].

efficient, OSD, having a higher specific gravity (SG) than coal, would be entirely rejected and not influence clean coal quality; however, there is always some degree of imperfection in even the most efficient coal preparation processes. Probable error (E_p) is a measure of the degree of imperfection of a density-based separator based on the amount of heavy material it misplaces to the float and the amount of light material it misplaces to the sink. E_p values for typical density-based separators range from 0.02 to 0.05 for heavy-media processes and from 0.15 to 0.25 for water-only processes. Even though the misplaced percentages are small for heavy-media systems, large amounts of OSD in the raw coal feed can result in significant amounts of this material reporting to the clean coal product. The impact is particularly severe when lighter OSD materials such as black shale (SG ~1.8) are encountered. Most separators have d_{50} cut points set close to this density implying that as much as 50% of such OSD material can report to the product. Compounding the problem, OSD material is known to contain significantly higher amounts of pyrite, mercury, and other trace elements [3]. Therefore, the negative impact on product quality can be very high. This aspect of OSD's impact is further detailed in this chapter, but additional research is still needed.

Other costs related to OSD have been highlighted by Luttrell et al. [4], who estimated the cost of OSD from a blending perspective. It was argued that approximately 3 tons of 35%–40% ash middlings can replace 1 ton of pure rock without changing the total mass of ash in the coal product. Thus, in coal blending operations, coal preparation middlings can be added by increasing separator cut points if OSD is reduced. The analysis predicted that for 100 tons of ROM coal, 21 tons of middlings can be substituted for a 7 ton reduction in pure OSD rock. Assuming a selling price of $25 per ton, this net gain of 14 tons of product per 100 tons of feed results in increased profitability of approximately $14 million per year for a 1000 tons-per-hour preparation plant.

9.1.3 Importance of achieving OSD control

Increased competition due to the recent industry downturn and the resulting focus on reducing production costs have led some mines to more closely examine the feasibility of reducing OSD to benefit from favorable economic impacts of that strategy. At one Illinois Basin mine, a 2.5 in. (0.06 m) reduction in OSD was demonstrated after mine management, and the project team educated the face crew on the impacts of OSD [1]. Data collected during that study demonstrated that even without any focus on or incentive to reduce OSD, desired levels were inadvertently achieved about 15% of the time. This suggests that substantial reductions in OSD are possible if conscious efforts are made.

Most of the field data presented in this chapter resulted from research conducted in the Illinois Basin, a region that saw coal production levels decline by one-half in less than a decade following legislation passed to curb sulfur emissions. Trying to stay competitive, Illinois Basin coal producers focused on costly "clean coal technology" for preparation plants and power plants instead of reducing OSD so that less "cleaning" was necessary. Consequently, marketing decisions favored lower-cost producers in the Powder River Basin for many years. Similar market dynamics should be expected as a result of policies and regulations dealing with mercury, particulate matter, and carbon emissions, just to name a few. The industry needs to prepare in advance to comply, and decreasing the amount of OSD produced is always a viable option for reducing costs and environmental impacts of coal mining.

9.2 Experimental modeling of an OSD control procedure

9.2.1 Sample collection

This section summarizes a study [2] conducted at five Illinois coal mines where channel samples were collected from 13 mechanized mining units. Each sample included roof and floor strata and coal from three horizons within the seam (i.e., top 3.0 in. (0.08 m), bottom 3.0 in. (0.08 m), and the rest of the coal seam) as shown in Fig. 9.2. At a few locations where a rock parting was present, a sample of the same was also obtained. Four of the 13 units sampled were in the Illinois No. 5 seam; nine of the 13 units sampled were in the Illinois No. 6 seam. In total, 57 channel samples were collected.

Samples were collected according to the procedure described in USGS Circular 735 [5]. First, a fresh face of coal was exposed on the pillar by chipping with a chisel and hammer. Then, a longwall hammer was used to cut a 5 in. (0.13 m) wide by 3 in.

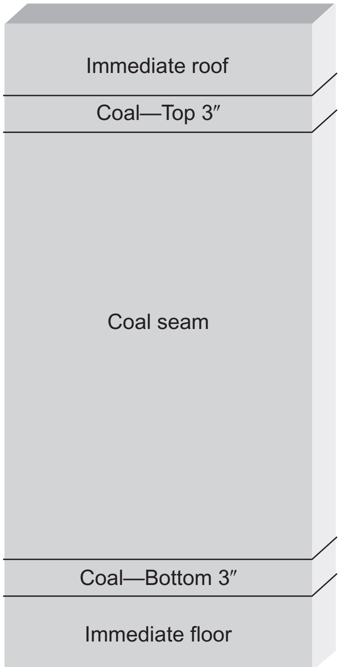

Fig. 9.2 Channel sample horizons [2].

(0.08 m) deep channel into the cross section of the face being sampled. A brattice curtain was laid on the floor to collect coal and rock material as it was chipped from the channel. Approximately 10 lb (4.5 kg) of the material were collected at each sampling location. These samples were placed in buckets and sealed with tight lids. For each roof and floor sample and for top and bottom coal horizons, separate 2 lb (0.9 kg) samples were collected, placed in airtight bags, sealed, and labeled.

9.2.2 Sample preparation

In the laboratory, samples were prepared by crushing in a jaw crusher and screening at 1 mm and 100 mesh sizes to extract the 1 mm × 100 mesh size fraction. A Jones riffler was used to obtain a representative (~150 g) split of this sample for conducting a washability analysis.

9.2.3 Washability analysis

ASTM procedures [6] for conducting coal washability analyses were carefully followed. This involved fractionation at specific gravities of 1.4, 1.5, 1.6, 1.7, 1.8, 1.9, and 2.0. Densities were prepared using typical mixtures of organic liquids such as perchloroethylene and dibromomethane. Samples were placed in the lowest-density liquid, and the portion that floated to the top was collected. The remaining portion was placed in the next lowest density, and the process was repeated yielding a "float" fraction for each specific gravity and a "sink" fraction for the highest specific gravity. At each specific gravity, if a sufficient quantity of material required for further analysis was not obtained in the float fraction, it was combined with the floated fraction from the adjacent specific gravity, and the weighted average of densities for the combined sample was calculated. For samples where all material sank at 2.0 specific gravity, an additional gravity cut was made at 2.2 specific gravity to fractionate the sample. This separated each of the 57 channel samples into as many as seven float fractions in increments of particle densities and one sink fraction with particle densities greater than 2.0 specific gravity yielding a total of approximately 450 sample fractions.

9.2.4 Advanced procedure for washability analysis

Traditionally, the average gravity of material for each fraction is calculated as the arithmetic mean of upper and lower gravities. For example, a material that sinks at 1.4 specific gravity but floats at 1.5 specific gravity is assumed to have an average specific gravity of 1.45. However, it is possible that all of the 1.4 × 1.5 density fraction may actually be in the 1.4 × 1.45 density fraction with an average specific gravity closer to 1.425. Thus, to obtain a truly accurate specific gravity requires maximizing the number of fractions by narrowing fraction ranges to very small intervals, which can quickly become unreasonably expensive.

To achieve optimal use of research funds, an alternate approach [7] was adopted in Patwardhan's study. Gravity fractions were obtained in traditional fashion and then subjected to ash analysis. The ash content in coal is known to be a function of mineral matter content, which is a function of specific gravity. Hence, the ash content of different gravity fractions can be regressed against the average specific gravity as determined using the traditional approach. The resultant regression equation can then be used to calculate a corrected average specific gravity of the fraction based on its ash content. This approach has been shown to be very reliable, particularly when the regression is applied to samples from one location [8]. Another advantage of this approach is its ability to correct errors in washability analysis, which are relatively frequent when working with fine size fractions.

9.2.5 Sample analysis

Each of the specific gravity fractions collected in the above fashion was rinsed, dried, weighed, and analyzed for ash, sulfur, and 51 trace elements. The ash analysis was conducted according to ASTM standards in a muffle furnace at a university laboratory. Trace element analyses, which included analysis for sulfur, were conducted at a commercial laboratory using aqua regia digestion followed by inductively coupled plasma mass spectrometry (ICP-MS) and inductively coupled plasma atomic emission spectroscopy (ICP-AES).

9.2.6 Modeling gravity separation partition curves

A density cut point of 1.9 was simulated on ROM material with a corresponding probable error (E_p) value of 0.057. With this information, partition numbers were calculated using Whiten's classification function [9], which is as follows:

$$\text{Partition Number} = 100 \times \frac{1}{1 + \exp\left(\dfrac{\ln 3(\mathrm{SG} - \mathrm{SG}_{50})}{E_p}\right)} \tag{9.1}$$

This partition number represents the percentage of the density fraction "SG" reporting to product when a density cut of "SG_{50}" is made. Comparing actual yield predicted by partition curve modeling with theoretical yield predicted by the washability analysis provides an estimation of organic efficiency, which is a measure of achievable product quantity and quality.

9.2.7 Modeling cost impacts of dilution

OSD's impact on the overall operating cost of a coal mine was examined by separating it into four components.

9.2.7.1 Mining costs

The mining cost component is separated into extraction costs and material handling costs. The extraction cost impact is a result of the fact that OSD material has no value to the customer and is expected to be discarded in the coal preparation process. Mining this material results in an effective "clean coal" productivity loss, which can be quantified using any production modeling tool. In this case, a simple deterministic production model developed by the authors [10] was used. The analysis assumes that extraction costs are the same for all sources of dilution and ignores the reality that mining harder roof strata generally have a higher cost than mining softer coal and floor strata due to increased bit replacement frequency and greater energy needs. In some cases, mining floor strata may be more expensive depending on moisture and plasticity characteristics. Extracted OSD incurs material handling costs as it is conveyed from the point of extraction to the point of processing or loading for shipment to the customer. Obviously, OSD is conveyed together with coal; however, for modeling purposes, it is treated separately to account for higher densities characteristic of OSD, which have a direct bearing on conveyor energy costs.

9.2.7.2 Processing costs

The processing cost impact includes several components. The purpose of the rotary breaker at the feed end of most coal preparation plants is to remove large inorganic material, a large percentage of which is OSD. In most instances, eliminating OSD would eliminate the need for a rotary breaker. OSD size distributions, weight densities, and abrasive and chemical characteristics increase wear on processing equipment and infrastructure as it flows through each processing circuit resulting in higher maintenance costs. OSD extracted from the floor is mostly clay, which breaks down into ultrafine particles ultimately handled by a thickener. Eliminating OSD does not eliminate the need for a thickener, as was the case with the rotary breaker, but it could allow for a smaller thickener to be used. Regardless of thickener size, OSD requires more flocculant and flotation chemicals. In addition, clay material also affects media viscosity in heavy-media operations, which has a negative impact on the separation efficiency of those circuits. Cost estimates generated in a related study of these effects [11] were utilized in this case.

9.2.7.3 Quality impact costs

Quality impacts of OSD are the result of near-gravity material in OSD being misplaced to the clean coal product. All coal separation devices have imperfection (I) and probable error (E_p) ratings, a measure of the amount of misplacement they allow. Modern heavy-media systems have lower I and E_p (i.e., are more efficient) than older water-only systems. Misplaced material results in higher ash and sulfur levels in the final product. These translate to lower yields, reduced heating values, and higher sulfur emissions. For pollutants that are capped, such as sulfur, there are costs for mitigation or purchase of allowances. To estimate cost impacts of quality deterioration due to increased sulfur levels in the product, an allowance price of $390 per ton of SO_2 [12] was used. Misplaced material is typically higher in undesirable trace elements, which also has an impact on product quality. Although samples were analyzed for several trace elements, mercury (Hg) was used as a surrogate to determine quality impacts because it has come closest to being a regulated pollutant in the United States.

When the Clean Air Mercury Rule (CAMR) passed, allowances were expected to trade for approximately $2000 per ounce ($32,000 per pound) [13]. This value was used to determine cost impacts due to increased levels of trace elements from mining OSD. Finally, in assessing quality impacts, average transportation costs of $7 per clean ton and a selling price of $1.50 per million British thermal units (MMBTU) were used for Illinois coal based upon one of the previously referenced studies [11].

9.2.7.4 Waste disposal costs

Since most of the mined OSD material is rejected during processing, disposal must be considered in assessing cost impacts. In the United States, coal mine waste material is typically disposed of in "gob" piles and slurry impoundments, although serious consideration is being given to backfilling, which is more common practice internationally. Previously developed disposal cost estimates for coarse and fine refuse [14] are used in this case. They consider only operational costs of waste disposal and ignore indirect disposal costs associated with land acquisition, permitting, and reclamation, all of which have risen dramatically since the turn of the century [15].

9.3 Modeling results

9.3.1 Product quality impacts of OSD

To show the depth of analysis on product quality developed in Patwardhan's study, results obtained for one channel sample collected at one mine are presented. Then, summary results for all samples are provided.

9.3.1.1 Detail of quality analysis for mine 1 #2 sample

As described earlier, the #2 channel sample collected at mine 1 included separate roof rock, coal seam, and floor clay fractions. Measured thicknesses of these fractions were 10.2, 58.9, and 5.0 in., respectively. Specific gravities were 2.1, 1.4, and 2.6, respectively. Based on these data, percentages of each fraction in the ROM coal were calculated at 18.0%, 71.2%, and 10.8%, respectively. By simulating a density cut point of 1.9, product quantity and quality were estimated as shown in Table 9.1.

The model predicts that mining and processing (at a 1.9 specific gravity cut point) the entire 74.1 in channel profile results in a yield of 66.72% achieved at ash, sulfur, and mercury contents of 7.69%, 0.87%, and 0.092 ppm, respectively. If only the coal seam material is mined, the model predicts a yield of 66.0% at ash, sulfur, and mercury contents of 7.14%, 0.87%, and 0.087 ppm, respectively. Thus, extracting 15.2 in. of OSD gains just 0.72% in mass yield but increases clean coal ash content by 7.6%, clean coal sulfur content by 0.95%, and clean coal mercury content by 6.0%. Contents of trace elements As, Pb, and Se also increase by 7.5%, 1.5%, and 64.3%, respectively.

Table 9.1 **Data analysis for mine 1 #2 channel sample**

Corrected average SG	Wt (%)	Partition number	Yield (%)	Ash (%)	S (ppm)	As (ppm)	Hg (ppm)	Pb (ppm)	Se (ppm)
Roof	***18.0%***								
1.96	5.4	23.2	0.226	52.22	1.68	25.6	0.6	43	146
2.08	80.3	3.3	0.475	59.81	1.6	23.3	0.55	40.9	138
2.20	14.3	0.3	0.007	68.32	1.56	21.9	0.68	37.3	121
Cumulative	100		0.71	57.48	1.6	24.0	0.57	41.5	140
Seam	***71.2%***								
1.28	88.5	100.0	63.04	6.29	0.73	2.7	0.08	15.2	2.2
1.48	2.4	100.0	1.73	19.88	2.93	12.8	0.27	101	3.9
1.55	0.7	99.9	0.48	24.53	2.81	3.6	0.11	4.6	3.0
1.72	0.6	97.0	0.40	35.97	4.86	4.8	0.16	8	6.1
1.76	0.5	93.2	0.32	38.98	7.65	6.9	0.23	14.2	7.3
2.19	7.3	0.4	0.02	67.18	10	81.4	0.46	266	6.5
Cumulative	100		66.00	7.14	0.87	3.03	0.09	17.4	2.3
Floor	***10.8%***								
2.02	1.3	8.3	0.012	56.39	1.63	20.6	0.55	37.4	104
2.57	98.7	0.0	0.00	92.74	0.76	2.6	0.05	23.3	0.6
Cumulative	100		0.01	56.47	1.63	20.56	0.55	37.4	104
Total	*100%*		66.72	7.69	0.87	3.26	0.09	17.7	3.78
%Difference total versus seam only			0.72	7.6	0.95	7.5	6.0	1.5	64.3

Cut point = 1.9 SG.

9.3.1.2 Summary results for all samples

Table 9.2 lists average (for all sampled units) ash, sulfur, and trace element contents for each sampled strata. These data reveal that roof rock contains significantly higher trace element concentrations compared with either the coal seam or the floor strata. Mercury content in the roof is almost three times that of the coal seam, while mercury content in the floor is only slightly higher than that of the coal seam. Trace element concentrations in the roof strata are one to two orders of magnitude higher than those in the coal seam and are higher than those in the floor strata despite the fact that ash content of floor strata is somewhat higher than ash content of roof strata. These results establish that concentrations of unwanted constituents in coal, such as ash, sulfur, and various trace elements, are significantly higher in OSD and in roof strata in particular and that mining OSD should be minimized to the extent possible.

The "blue band" is a prominent shale parting commonly found near the bottom of the Illinois No. 6 coal seam. Where this rock band is present, its ash, sulfur, and trace element concentrations are significantly higher than for the coal seam. The "blue band" is technically not OSD due to its position within the seam. Furthermore, selective elimination during mining is not practical. Nevertheless, these results suggest that avoiding any kind of dilution, be it out-of-seam or in-seam, is beneficial from a product quality perspective.

Results from simulated cleaning of this ROM material are presented in Table 9.3. Although there is a high degree of variability in results, it appears that, on average, increases in ash, sulfur, and trace element concentrations in clean coal attributable to OSD are quite significant. Ash, sulfur, and mercury contents in clean coal are shown to increase by 8.0%, 1.9%, and 3.8%, respectively, when OSD is present in ROM coal. Other trace elements also increase substantially with As and Se contents increasing by as much as 12% and 46%, respectively. These results clearly establish the hypothesis that mining OSD has a significant and negative impact on product quality in terms of ash content (which implies a corresponding reduction in heating value), sulfur content,

Table 9.2 Average ash, sulfur, and trace element contents in sampled horizons of Illinois mines

Strata	Wt (%)	Ash (%)	S (%)	As (ppm)	Hg (ppm)	Pb (ppm)	Se (ppm)
Roof	13.53	71.55	2.77	25.77	0.29	34.97	79.59
Coal seam—top 3 in.	3.40	9.42	1.05	1.76	0.07	2.94	4.80
Coal seam—middle bench	75.50	12.94	2.00	4.77	0.11	12.06	2.93
Blue band	4.83	53.10	4.75	10.27	0.39	128.3	5.10
Coal seam—bottom 3 in.	3.63	16.19	3.04	4.16	0.08	7.58	2.13
Floor	7.98	79.38	3.25	6.75	0.12	38.79	2.28

Table 9.3 Summary of percent increases in ash, sulfur, and trace element concentrations in clean coal due to OSD mining

Mine and unit	Yield	Ash	S	As	Hg	Pb	Se
Mine 1 #1	0.23	0.11	0.33	0.68	0.55	1.39	0.05
Mine 1 #2	1.09	7.60	0.95	7.45	5.99	1.49	30.3
Mine 1 #4	0.56	1.59	0.54	3.31	0.77	1.01	13.5
Mine 1 #5	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Mine 2 #1	5.33	22.5	17.6	24.78	3.76	6.62	4.70
Mine 2 #3	10.09	44.3	−0.29	15.51	2.71	23.77	143
Mine 2 #4	7.03	4.73	−0.96	23.40	−0.67	6.12	118
Mine 3 #W	0.18	0.06	0.03	0.61	0.01	0.13	2.06
Mine 3 #2	0.46	0.13	−0.05	0.30	−0.09	−0.07	0.23
Mine 4 #EHG	2.86	3.90	1.29	10.46	3.04	1.08	19.6
Mine 5 #2	1.08	0.62	1.05	2.04	0.08	1.12	1.97
Mine 5 #W	5.38	17.8	3.05	63.71	32.59	24.44	258
Mine 5 #MS	1.18	0.12	0.91	1.37	0.59	2.53	0.06
Average	2.7	8.0	1.9	11.8	3.8	5.4	45.5

Cleaning simulated at 1.9 SG cut point.

and most trace element concentrations including Hg, As, and Se, which have been the subject of potential environmental legislation.

The impact of OSD on clean coal quality is highly variable across different mines and units. This is particularly true for mercury. The increase in mercury content from OSD varied from as low as a fraction of 1% to as high as 33%. Overall, it appears that even when OSD is eliminated and drastic reductions in some very undesirable trace elements, such as mercury, do not occur, their concentration is nevertheless definitely reduced. Due to extremely high mitigation and credit costs projected for these elements, even small reductions have the potential to translate into large quality cost savings. Associated reductions in heating value can also amount to significant penalties. When these are avoided in combination with cost savings from eliminating OSD mining that will be described in the next section, there is certainly sufficient incentive to implement some form of OSD control.

9.3.2 Cost impacts of OSD

Once OSD impacts on product quality were understood, the effect of OSD on mine economics was modeled for a “typical” Illinois coal mine using average OSD mining characteristics, average OSD qualities, and average product quality changes due to OSD as presented in Tables 9.2 and 9.3. Quality impacts were estimated separately for OSD removed from roof and floor strata and include effects of reduced heating value, increased sulfur and mercury contents, and increased transportation costs on a MMBTU basis, as shown in the following computations:

Cost impact of higher ash content (lower heating value)

Cleaned seam-only ash content = 10% (12,960 BTU/lb)

Increase in ash content due to OSD = 8%

Ash content with OSD = $10 \times 1.08 = 10.8\%$ (12,845 BTU/lb)

BTU reduction in 1 ton of clean coal = $2000 \times (12{,}960 - 12{,}845)/10^6 = 0.23$ MMBTU

Cost of BTU reduction = 0.23 × $1.50 = $0.35/ton of clean coal

Increase in ash content in 1 ton of clean coal = 0.008 tons

Increased cost of transportation = 0.008 tons × $7/ton = $0.06/ton of clean coal

Cost impact of higher sulfur content

Cleaned seam-only sulfur content = 2%

Increase in sulfur content due to OSD = 1.9%

Sulfur content with OSD = $2 \times 1.019 = 2.038\%$

Sulfur increase in 1 ton of clean coal = 0.00038 tons

Cost of sulfur increase = 0.00038 × $390 = $0.15/ton of clean coal

Cost impact of higher mercury content

Cleaned seam-only Hg content = 0.10 ppm

Increase in Hg content due to OSD = 3.8%

Hg content with OSD = $0.1 \times 1.038 = 0.1038$ ppm

Hg increase in 1 ton of clean coal = $2000 \times (0.1038–0.1)/10^6 = 0.0000076$ lbs

Cost of Hg increase = 0.0000076 × $32,000 = $0.24/ton of clean coal

9.3.2.1 Overall cost impact due to quality impacts of OSD mining

The above cost impact computations add up to $0.79 per ton of clean coal. Considering that coal preparation plants in Illinois average around 65% yield, the overall cost of quality impacts translates into $1.22 per ton of ROM coal. Separating this cost into roof and floor components is accomplished by realizing that ash- and transportation-related impacts are applicable to both roof and floor dilution, which average 13.5% and 8.0%, respectively (see Table 9.2, wt% column), while sulfur- and trace-element-related impacts are associated primarily with roof OSD. This leads to the determination that the cost of quality impacts is higher for roof OSD ($5.09 per ton of roof dilution versus $1.26 per ton of floor dilution). From a purely economic perspective, this finding leads to the recommendation that if mining height considerations require cutting out of seam material, floor strata should be mined before roof strata.

9.3.2.2 Summary of OSD cost impacts

Overall, cost impact modeling results are shown in Table 9.4. Fig. 9.3 provides a graphic representation of the same data. As noted previously, mining, processing, and waste disposal cost estimates are taken from data generated in previously cited studies [10, 11, 14]. Quality impact cost estimates developed by Patwardhan [2] were described in the previous section of this chapter.

Table 9.4 Incremental cost impacts of OSD in 2010 US$/ton [2, 12, 13]

	Roof dilution	Floor dilution	Total OSD cost (clean ton basis)
Base mining cost	$4.69	$4.69	$1.55
Materials handling	$0.75	$0.75	$0.25
Base processing cost	$1.40	$1.40	$0.46
Rotary breaker	$0.15	–	$0.03
Maintenance	$0.30	–	$0.06
Viscosity effect	–	$0.21	$0.03
Flocculant usage	–	$0.43	$0.05
Quality impact cost	$5.09	$1.26	$1.22
Waste disposal cost	$1.13	$0.54	$0.30
Total	$13.51	$9.28	$3.95

As seen in Table 9.4 and Fig. 9.3, the overall cost of dilution based on OSD mining and recovery assumptions previously explained is $3.95 per clean ton. This would be the expected cost reduction if extraction of OSD material could be completely avoided. Considering that to be an unreasonable expectation and assuming that OSD mining is only reduced by one-half, cost savings are still approximately $2.00 per clean ton. This amount is substantial and could easily have a positive impact

Fig. 9.3 Cost impacts of OSD by component and in total.

on the competitiveness of any individual coal-mining operation that implemented OSD control measures. Similarly, widespread adoption of OSD control measures improves the competitiveness of the entire coal industry.

It can be seen that productivity losses resulting from OSD mining have the greatest impact on cost followed by quality deterioration due to OSD quality impacts. Additional costs incurred due to OSD in order of importance are processing and waste disposal costs. There is general industry recognition of OSD impacts on productivity, processing, and waste disposal; however, the cost of quality deterioration due to the impact of OSD is a matter that has received little attention from the industry. It is hoped that increasing understanding of the quality impact will provide the needed incentive for coal mine operators to pay more attention to and increase efforts aimed at minimizing OSD.

9.4 Controlling OSD

The first step in controlling OSD is to understand what causes it. Each mining operation has its own issues; however, OSD in ROM coal is principally a factor of either weak roof and floor strata or coal seam height. In many cases, both of these factors are involved. Chugh et al. [16] identified the most common causes of OSD as follows:

1. Poor roof and floor conditions due to weak or disturbed geology in the region of mining. This cause is generally beyond the control of mine workers and results in unintentional extraction of OSD.
2. Low coal seam heights that may occur universally throughout a mine or unexpectedly in isolated areas. While seam height cannot be controlled, OSD produced from this cause is typically discretionary in that it results from mining equipment operators cutting into the roof or floor to gain height for operator comfort or equipment clearance (which overlaps with the next cause) when marginal seam heights of 4–5 ft (1.2–1.5 m) are encountered. This is termed operational OSD.
3. Larger and heavier equipment, which create one or both of the following effects:
   a. Roof strata must be extracted to provide for increased clearance requirements.
   b. Damaged weak floor strata are mixed with loose coal during cleanup operations.

   OSD produced from this cause is termed engineered OSD in that it results from mine plans and equipment designs that are engineered to maximize production capacity.
4. As just explained, not only high-capacity production systems generate engineered OSD, but also they cause operational and unintentional OSD as described in the following examples:
   a. Deep cuts tend to inhibit the mining machine operator's ability to see the cutterhead and discern if it is within the coal seam.
   b. Advanced haulage systems give the mining machine operator less time for positioning the machine and checking horizon control.
   c. Modern coal processing systems have led to the perception that mining a clean product is no longer important because any dilution will be removed during processing. This has already been shown to be a gross misperception.

Considering that unintentionally produced OSD is beyond control given current technology (i.e., extractive equipment and mining methods), efforts to control OSD have and should focus on operational and engineered OSD. Operationally, produced OSD is best addressed through workforce education and awareness programs. OSD that results from engineered systems is best addressed with more advanced engineering.

9.4.1 Workforce education and awareness

Working with mining various mining professionals, Chugh et al. [17] developed an educational program on controlling OSD that was delivered to personnel at a southern Illinois coal mine by the mine's management team. Immediately following its presentation, a study was conducted to gauge the program's impact by measuring and quantifying any level of reduction in OSD output.

The first phase of this exercise was to establish a benchmark to improve upon, that is, measure OSD levels resulting from existing practices. An extensive industrial engineering study collected time study data for 30 cuts and dilution measurements at 700 points located across 10 crosscuts of advance in an 11-entry continuous miner production section. Dilution measurements included mining height, coal seam height, and roof and floor dilution thicknesses as shown in Fig. 9.4. Phase 1 measurements indicated average mining height 6.53 ft (1.99 m) with average coal seam height of 5.61 ft (1.71 m). Average extracted OSD measured approximately 11 in.

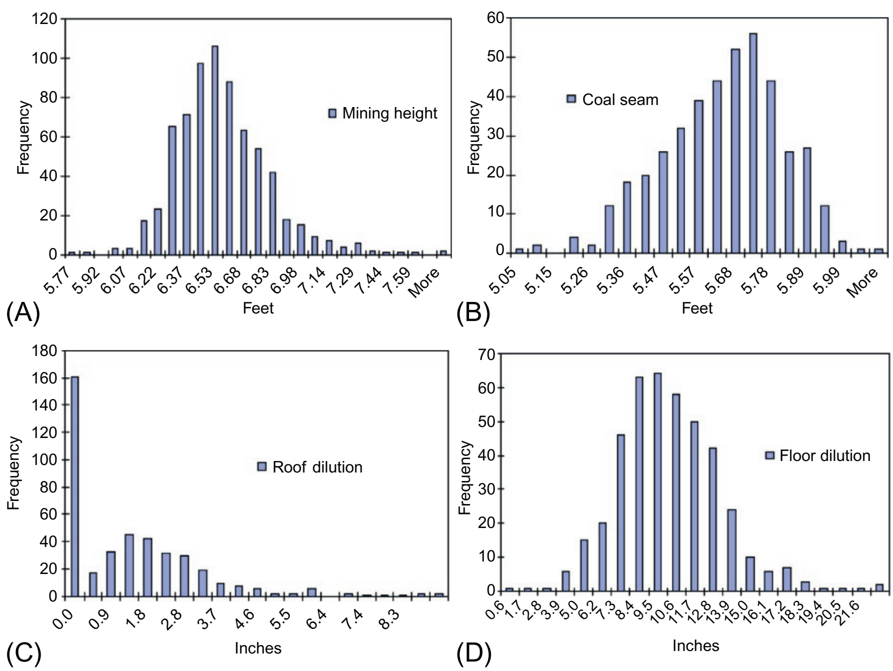

Fig. 9.4 OSD awareness program phase 1 (benchmarking) measurements: (A) mining height, (B) coal seam thickness, (C) roof dilution, and (D) floor dilution [17].

(28 cm), of which less than 1.5 in. (3.8 cm) was roof dilution meaning that OSD at this mine primarily consisted of floor dilution. Mining equipment in use required a minimum mining height of 6.0 ft (1.8 m) giving opportunity to reduce dilution by 6.4 in. (16.3 cm).

With these data in hand, the project team established a goal to reduce dilution by 3.0 in. (7.6 cm) as a first step with the expectation that as mine personnel became comfortable working in slightly lower mining heights, further reductions may be attempted. In establishing this goal, it was noted that it had been achieved 8%–12% of the time during the phase 1 data collection period as indicated in Fig. 9.4. This further supported the feasibility of consciously reducing dilution to achieve the established goals.

Using data collected in phase 1 coupled with engineering design modeling, an OSD educational awareness program (EAP) was prepared. The program included computer-generated models showing out-of-seam mining levels, the influence that existing OSD levels had on product quality, and the associated economic impact that resulted. The program also set forth the OSD reduction goal and explained the effect that achieving it would have on productivity and profitability. Chugh's research team presented the EAP to a mine management team who then delivered it to continuous miner and roof bolter operators working on the mechanized mining unit being evaluated.

Immediately following presentation of the EAP, a second phase of data collection commenced. Phase 2 data collection lasted 2 weeks during which time study data were collected for another 30 cuts and out-of-seam dimensions were measured at 275 locations while the same 11-entry section advanced four more crosscuts. These data were analyzed to determine any productivity improvements and OSD reductions.

Time study results indicated a marginal increase in loading time (38 s per haulage unit in phase 2 as compared with 37 s per haulage unit in phase 1). This difference is not statistically significant at a 95% confidence interval, so it cannot be scientifically concluded that the educational program resulted in any measurable productivity improvements. However, researchers believe the difference is a result of the cautious approach adopted by continuous miner operators in attempting to stay in-seam and reduce OSD. Earlier studies [1, 18] conducted by some of the same research team members showed that longer loading times result in greater utilization of haulage unit capacity, which is the single biggest productivity factor in continuous miner batch haulage systems.

Fig. 9.5 shows the progression of mining heights and dilution from just prior to presentation of the OSD EAP through 2 weeks of phase 2 data collection. The center of the coal seam is represented by the *y*-coordinate of 0 ft. Red, blue, and violet colors represent mined thicknesses of roof, coal, and floor, respectively. The figure indicates a consistent post-EAP reduction in roof dilution; however, floor dilution, although reduced immediately following the EAP, drifted back to and even beyond pre-EAP levels over the 2-week phase 2 data collection period. It should be noted that lithologic properties of the geologic cross section at the study location enabled easier detection of the coal-roof interface whereas the coal-floor interface was not as apparent.

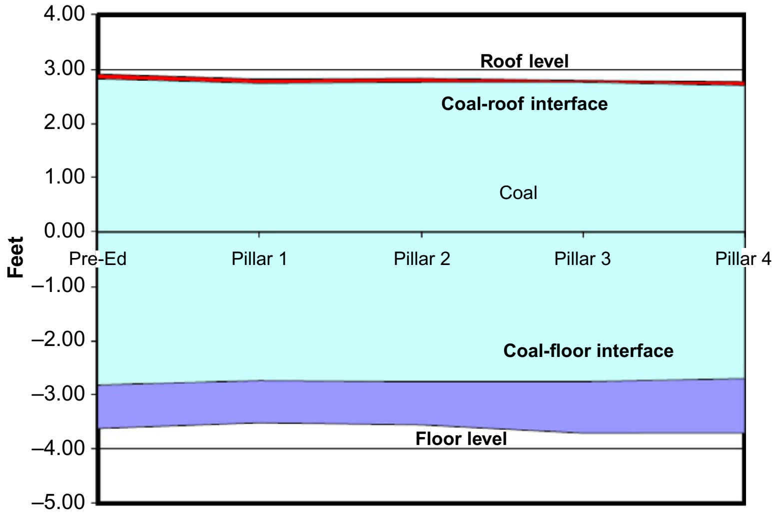

Fig. 9.5 Variation in mining height and OSD extraction from just before OSD awareness training through 2 weeks following the training [17].

Analysis of pre- and post-EAP data (see Fig. 9.6) revealed a definite relationship between OSD thickness and coal seam thickness. OSD is observed to increase with decreasing seam height. The relationship is described by the linear regression equation as follows:

$$\text{OSD thickness} = -0.8331 \times \text{seam thickness} + 5.4945 \quad (9.2)$$

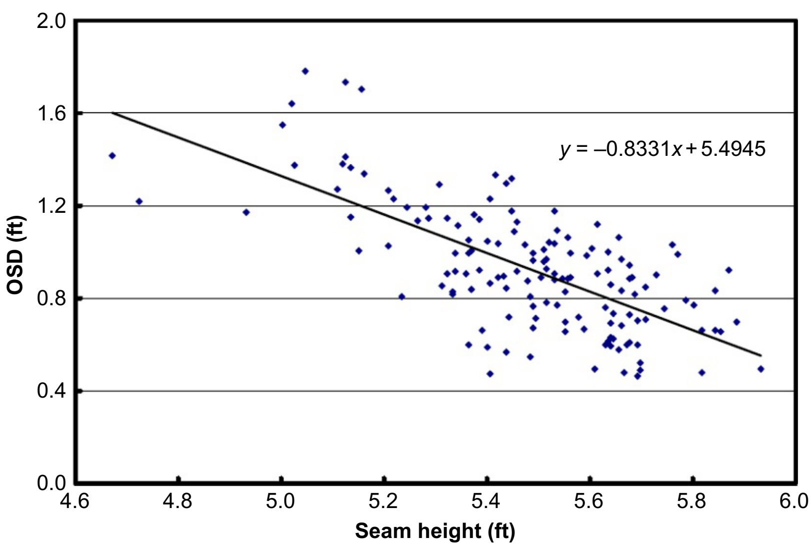

Fig. 9.6 Regression analysis showing relationship between OSD thickness and seam height [17].

Assuming the entire coal seam thickness is extracted, mining height is the sum of OSD thickness extracted (from roof and floor) and seam thickness, that is,

$$\text{Mining height} = \text{OSD thickness} + \text{seam thickness} \tag{9.3}$$

Substituting Eq. (9.2) in Eq. (9.3) gives.

$$\text{Mining height} = 0.1669 \times \text{seam thickness} + 5.4945 \tag{9.4}$$

Thus, change in mining height as a function of change in seam height can be represented as.

$$\frac{\partial(\text{Mining height})}{\partial(\text{Seam thickness})} = 0.1669 \tag{9.5}$$

Accordingly, for every 1.0 in. (2.5 cm) reduction in seam height, mining height decreases by only 0.2 in. (0.4 cm). Comparing phase 1 and phase 2 data, coal seam height decreased by 1.9 in. (4.9 cm), which should have resulted in mining height decreasing by only 0.3 in. (0.8 cm); however, mining height actually decreased by almost 1.0 in. (2.5 cm) as shown in Table 9.5. When only the period immediately after the EAP is considered (pillar 1), instead of the reduced seam height resulting in a mining height reduction of 0.3 in., the actual reduction was almost 1.7 in. These greater-than-expected decreases in mining height are attributed to the effectiveness of education and awareness in controlling OSD.

As previously mentioned, immediately following the EAP, mining height was reduced by ~1.5 in. (3.8 cm); however, by the end of phase 2 data collection efforts, mining height had returned to and even exceeded pre-EAP levels. This emphasizes the need for ongoing monitoring and refresher EAPs. The research team recommended a quarterly refresher EAP that could easily be incorporated with routine safety shares owing to the fact that mining OSD generates higher levels of silica dust exposure [17].

Table 9.5 Summary of dilution measurements (1.0 ft = 0.3 m)

	Floor (ft)	Roof (ft)	Coal seam (ft)	Mining height (ft)
Pre-EAP average	*0.80*	*0.09*	*5.64*	*6.49*
Post-EAP average	*0.87*	*0.06*	*5.48*	*6.41*
Pillar 1	0.78	0.08	5.48	6.35
Pillar 2	0.80	0.07	5.51	6.37
Pillar 3	0.95	0.04	5.51	6.46
Pillar 4	1.00	0.07	5.40	6.47

9.4.2 Advanced engineering controls for reducing OSD

Two critical aspects of controlling OSD are alignment of the continuous miner in the face area and guidance of the machine during cutting. Efforts to automate those functions are discussed in this section. An essential component of any automated coal-mining method is seam-following technology. This is usually achieved through detecting the coal-rock interface. Multiple coal-rock interface detection (CID) technologies have been developed for this purpose [19]. A few common and viable (for Illinois Basin coal) options are briefly described. The National Mining Association and the Office of Industrial Technologies of the US Department of Energy have identified horizon sensing as a critical technology that should be a priority for future coal-mining-related research.

9.4.2.1 Natural gamma radiation (NGR)

NGR is the most advanced CID technology with more than 150 units employed around the world [20]. This method works on the principle that shale, clay, silt, and mud have higher levels of naturally occurring radioactivity than coal due to their containing small quantities of radioactive potassium (K-40), uranium, and thorium. Measured NGR decreases exponentially as a function of coal thickness; thus, attenuation of the NGR sensor toward coal can be used to measure coal thickness between the sensor and the rock interface. NGR technology has many features that make it a viable option in automated mining operations. It can measure coal thicknesses from 1.0 to 20 in. (2.5–50 cm). The unit is compact and easily mounted on mining machines. It has a display panel that is easy to read by operators using remote control devices. The most prevalent applications to date have been on longwall units. There are a few inherent weaknesses that arise from distribution of radioactive material in the coal seam. For example, NGR levels vary from seam to seam requiring units to be calibrated for each seam in which they will be used. A related issue is that NGR levels can vary within a seam depending on levels of radioactive constituents present at the time of geologic deposition. Also, rock partings (in-seam dilution) can show false seam boundaries. The applicability of NGR systems in Illinois may be limited since black shale is a typical immediate roof layer and it has radiation properties that are similar to coal.

9.4.2.2 Vibration-based CID

When coal and rock are cut, different patterns of vibration are generated. By analyzing these vibrations, the CID sensor can detect when the machine has started cutting boundary rock instead of coal. Vibrations analyzed include machine vibrations, in-seam seismic vibrations, and acoustic vibrations [21], with each having strengths and weaknesses depending on the application. When analyzing machine vibration, sensors can be mounted on the machine. This method has good potential when adaptive signal discrimination technologies are used to interpret vibration data. Feedback is immediate when the machine starts to cut rock, so mining can be stopped with minimal dilution extraction. In-seam seismic and acoustic sensors must be attached to the coal itself requiring frequent remounting as mining progresses. This is inefficient and defeats one of the main purposes for having an automated system. Their applicability in Illinois Basin conditions may be limited for two reasons. First, vibration-based CID determines the interface using differences in vibration characteristics, which depend on both rock hardness and fracture. Since fracture characteristics in Illinois are highly variable, it would be difficult to calibrate mining machines for any particular seam. Frequent recalibration may also increase operational cost in addition to making it unreliable. Second, Illinois Basin coal is hard and in some cases has higher compressive strength than roof and floor strata. Thus, differences in vibration frequency may be minimal.

9.4.2.3 Infrared CID

Different types of strata release varying amounts of infrared radiation during extraction. Infrared sensing devices can measure radiation emitted from the cutting zone and detect changes in strata being mined. This method has the distinct advantage that radiation readings can be captured from a remotely mounted sensor located behind the cutterhead, and they function even when it is obscured by dust and water sprays. This method can be used under any type of roof, and response time is instantaneous when the coal-rock interface is reached [20].

9.4.2.4 Optical/video CID

This technology is based on the concept that different types of strata have different light reflectivities. Optical sensing technology by itself is not very accurate, but is greatly improved with the addition of video cameras and image analyzing equipment. These sensors, like infrared sensors, can be remotely mounted and, with appropriate video technology, can penetrate moderate dust and water spray obstructions; however, heavy dust and water spray mist can cause problems. Another benefit of this system is that video data can be employed for guidance purposes [22].

9.4.2.5 Radar-based CID

This technology utilizes a single antenna, which transmits and receives Doppler radar pulses. A network analyzer controls signal frequency. Signals are attenuated as they pass through coal and bounce off the density interface of the confining rock, which is interpreted by the network analyzer to determine the distance to that interface. This system has reliable accuracy and operates well under most roof conditions. It is also suited for monitoring rib thickness between adjacent holes in highwall mining. Two disadvantages of this system are that it does not work well in coal seams with wave-dispersing properties and it requires the transmitter to be located within 4 in. (10 cm) of the coal [23].

9.4.2.6 Pick force CID

This CID method measures changes in the force exerted on one or more of the picks on a continuous miner cutterhead. The energy required to break differing types of rock results in varying forces being applied to any given pick. This phenomenon can be used to determine when the mining machine cuts into a different type of strata. This type of system can be conveniently integrated into the mining machine keeping components compact and protected. It is also capable of instantaneous feedback. To date, a commercial unit of this type has not been developed for advanced testing [19].

9.4.2.7 Application of CID technology in the Illinois Basin

One of the newest underground coal mines in the Illinois Basin supplies fuel to an electric-power-generating plant that is located on the same property and was designed to burn ROM coal. Control OSD is critical to the operating and economic success of this operation. To achieve target levels of OSD, mine engineers in collaboration with equipment manufacturers have tested CID technology combined with artificial intelligence (AI) to the control continuous miner's cutterhead height. CID sensors capture data on coal-roof interface profiles, which are remembered by AI systems and used to constrain machine control functions within established tolerances. A commercial CID sensor is mounted on the mining machine together with a set of inclinometers that identify the cutterhead position at any given point in time. Graphic displays of the cutterhead position and coal-rock interfaces in the roof and floor enable the machine operator to selectively mine coal while minimizing dilution.

9.5 Conclusions and recommendations

This chapter presents novel data describing OSD impacts on both clean coal quality and on costs attributable to OSD including the cost of quality impacts, which are summarized in the following conclusions:

1. OSD has a significant impact on clean coal product quality. This impact can be variable from mine to mine, but on average, results will not differ much from those obtained in the study described earlier. That study found that for five Illinois Basin mines, OSD's average impact on clean coal product quality was an 8% increase in ash content and a 2% increase in sulfur content over inherent content of these variables in the coal seam.
2. Analysis of channel samples revealed that roof and floor strata contained significantly concentrated levels of mercury and most trace elements of note to the coal industry. Mercury was used as a surrogate for all trace elements, and the study determined that OSD's average impact on clean coal product quality was a 4% increase in mercury content. Increases in several other trace element concentrations were also measured, some of which were startling, for example, cadmium (Cd) and chromium (Cr) registering 850% and 150% increase, respectively.

3. The costs of quality impacts from OSD extraction were estimated at $5.09 and $1.26 per ton of mined roof and floor strata, respectively. Including all other factors (mining, processing, waste disposal, etc.), OSD costs were estimated to be $13.51 and $9.28 per ton of ROM material. The difference between roof and floor costs suggests that if out-of-seam extraction is required for height or any other reason, floor should be mined instead of roof.
4. The total combined cost impact of OSD was estimated at $3.95 per ton of clean coal. Accepting the fact that OSD cannot be entirely limited, even a 50% reduction becomes a worthy target that if achieved would result in a cost savings of ~$2.00 per ton.

Having established the need for and value of reducing OSD extraction, two control strategies were presented. The first and easiest to implement is an educational awareness program (EAP). EAPs target operational OSD, that is, that caused by production practices that limit visibility or that result from the common assumption that processing eliminates it from the final cleaned product. An effective EAP requires detailed analysis of existing conditions and practices in order to establish benchmarks and targets that emphasize economic impacts. Given accurate information, most mine workers will respond with conscientious efforts to minimize OSD extraction; however, the mining environment is very dynamic, and it is easy to lose focus when breakdowns and other operational issues halt production altogether. Thus, refresher EAPs should be conducted on a frequent (e.g., quarterly) basis.

The second strategy is technology-based, and it targets OSD caused by engineering designs that include the use of larger, heavier equipment. Larger equipment requires increased mining clearances often necessitating the extraction of some OSD. Heavier equipment damages weak floor strata that become mixed with loose coal during cleanup operations. Numerous technologies have been developed and tested with the most successful being some form of coal interface detection (CID). CID systems use radiation (gamma, infrared, Doppler, etc.), vibration, energy, or force to detect coal seam boundaries and guide mining machine operators in avoidance of those boundaries.

The message of this chapter is that OSD control is of vital importance. Effective strides have been made in understanding causes and impacts of OSD, but these have been on a macro level; whereas to be effective in controlling OSD, this understanding needs to occur at or be transferred to the micro level within companies and at individual mines. One way to do this is by integrating OSD analysis into permitting activities, which seems to have usurped the attention of most mining engineers. Associating OSD control with permit applications can satisfy both regulators focused on minimizing environmental impacts of waste disposal and mine operators focused on minimizing production costs. Efforts in both directions will enhance the competitiveness of the coal industry and improve the marketability of coal as a low cost and clean burning fuel of choice.

Acknowledgment

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Coal mine methane: Control, utilization, and abatement

10

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10.1 Introduction

Coal, as the most abundant and economical fossil fuel, has been an important source of global primary energy production for the past two centuries, and will continue to be an essential component of the global energy mix for the next few decades. Without coal resources, the United Nations' development goals are not achievable [1]. Total minable reserves of coal to a depth of 1000 m (3300 ft) are estimated at 1 trillion tonnes (1.1 trillion tons), while estimated reserves to a depth of 3000 m (10,000 ft) range from 17 to 30 trillion tonnes (19–33 trillion tons) [2]. Total known mineable coal deposits are sufficient for at least 300 years of use, a timeframe that is approximately fivefold that of known natural gas resources and tenfold that of known crude oil resources. Approximately 60% of world steel production and 40% of current electricity production are powered by coal. Historically, coal has played a significant role in the advancement of civilization, and it will continue to be a major fuel source in developing countries for at least the next quarter century [3]. Coal is, therefore, central to the energy security of many countries and will continue to play a significant role in ending energy poverty around the world.

World coal production in recent years has reached almost 8 billion tonnes (9 billion tons) per year. At this rate, coal and the associated methane gas that is released are likely to remain a dominant source of energy in coming decades; however, it will not be without some resistance due to the debate regarding climate change. Despite the swift deployment of renewable energy, mainly in response to that debate, coal has been responsible for the largest upsurge in meeting energy requirement of all energy sources. To maintain that position, it is imperative that the industry implement modern mining principles in planning, designing, and extracting coal to control, utilize, and abate vented and leaked methane gas.

Coal seams are formed over millions of years by the biochemical decay and metamorphic transformation of plant materials. When plant materials such as roots, bark, and wood are deposited in swampy lakes, they undergo bacterial and chemical changes to make peat deposits. Coalification results as these deposits are subjected to high pressure and high temperature over time. The coalification process produces large quantities of byproduct gases such as methane (CH_4) and carbon dioxide (CO_2). The chemical composition of coal is therefore determined by its derivation from plants and comprises carbon, hydrogen, and oxygen, with nitrogen and sulfur as minor

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components. For instance, low-rank bituminous coal can be characterized by the formula $C_{10}H_7O$.

As shown in Fig. 10.1, as the coalification process progresses, coal of increasing density is formed under layers of sand and mud over millions of years. Coal rank is based on density, which proceeds from low to high as follows: peat, lignite, subbituminous, bituminous, and anthracite. The amount of byproduct gases increases with coal rank; i.e., it is highest for anthracite.

Most of the gases produced during the coalification process escape to the atmosphere, but a small fraction is retained in the coal. The amount of retained gases depends on a number of factors, such as burial depth, coal rank, type of immediate roof and floor rock strata, local geological anomalies, and the tectonic pressures and temperatures prevalent at that time [4].

Methane is the major component of gases in coal, comprising 80%–90% or more of the total gas volume. As coal is formed, the decomposing organic material produces methane gas, as well as CO_2, hydrogen, oxygen, nitrogen, and lower proportions of other gases like ethane, propane, butane, and argon. During diagenesis, carbon increases from 60% to 90%, whereas hydrogen decreases from 5.5% to 3%, meaning that large volumes of methane are released. Biogenic methane is first to form by anaerobic bacteria in the early stage of coalification, followed by thermogenic methane at temperatures of 120–150°C (248–302°F). Although much of the methane generated by the coalification process escapes to the atmosphere or migrates into the adjacent reservoirs or rocks, a significant volume remains trapped within the coal.

Coal mine methane (CMM) is therefore a term given to methane gas emitted due to coal-mining activities either from the coal seam or from surrounding gassy rock formations. Thus, coal and CMM are syngenetic in origin. In fact, CMM may be emitted from active and abandoned underground and surface coal mines, and as a result of postmining activities such as coal processing, storage, and transportation. While this book is focused on coal, it is not out of place to mention that encountering methane gas in metal and nonmetal underground mines is also common.

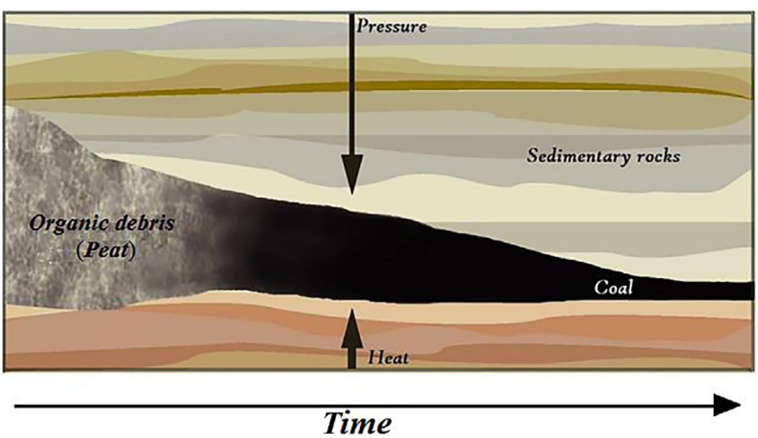

Fig. 10.1 Coalification process during geological time.

The amount of CMM generated at a specific mining operation depends on the productivity of the coal mine, the gassiness of the coal seam and other underlying and/or overlying formations, mining methods, operational variables, and geological conditions. CMM can ordinarily be emitted into the mine environment and exhausted from the mine shaft along with ventilation air. However, it may be captured by drilled boreholes that augment the mine's ventilation system.

Large amounts of methane released during mining present concerns about adequate mine ventilation to ensure worker safety, but they can also create opportunities to generate energy if this gas is captured and utilized properly. The implementation of cost-effective CMM utilization can yield substantial economical and environmental advantages such as improved miners' safety and mine productivity besides reduced greenhouse gas (GHG) emissions.

With increasing worldwide coal production, more attention is being given to the health, safety, and environmental impacts of methane released during coal mining. Methane not only creates unsafe working conditions in underground mines, but also heightens environmental concerns as it is a potent GHG. Recent research done by the United Nations has shown that the impact of methane on the atmosphere is more far reaching than was originally thought, and coal mines are the fourth largest source of methane emissions after oil and gas, landfill, and livestock industries [1]. Consequently, a major focus is currently being directed at minimizing methane emissions from the entire coal industry value chain from production through utilization.

In order to minimize environmental impacts of coal mining, it is important to ensure safe extraction, as well as control and abatement of CMM throughout the mine life cycle. Currently, technological advances have made it possible to significantly reduce CMM emitted even from the gassiest mines. Applying an engineering strategy for control and useful utilization of CMM not only improves working conditions at mines, but also provides an affordable and clean burning fuel.

This chapter focuses on everything that has anything to do with CMM capture and utilization including methane chemistry, CMM-related disasters in coal mining, the main factors affecting CMM accumulations in underground coal mines, methods for capturing CMM using boreholes into and from coal mines, removing methane from abandoned mines, benefits of capturing and controlling CMM, and the role of CMM in energy production and environmental concerns. Recent advances are summarized in an effort to help to the industry eliminate disasters and fatalities in coal mining and also minimize the environmental impact of CMM emissions, which leads to productive, safe, and responsible coal mining. It is hoped that those practical principles discussed will be useful in reducing global methane emissions especially from coal mines, and to advance the abatement, recovery, and use of CMM as a valuable clean energy source.

10.2 Coal mine methane

The main component of the primary coal seam gas is methane in concentrations of 80%–90%, which develops during the coalification or carbonization process.

In general, coal can store about six to seven times more methane than the equivalent volume of rock in a conventional reservoir.

From the author's viewpoint, it is possible to categorize gases derived from coal mines into four key forms: (i) coal mine methane (CMM) or working mine methane (WMM), (ii) coal seam methane (CSM) or coal bed methane (CBM) collected from unmined coal beds, (iii) abandoned mine methane (AMM) drained from depleted or inactive mines, and (iv) syngas from underground coal gasification (UCG) by which coal is gasified in situ.

As mentioned, CMM is a general term for all methane released mainly during and after coal-mining operations. Therefore, CMM is a type of coal gas present in active working mine sites, and may be captured through a drainage system or vented from coal mines by a ventilation system; i.e., ventilation air methane (VAM). In drainage systems, CMM may be captured via surface vertical premining boreholes, horizontal premining boreholes ahead of the coal face, or postmining wells drilled into the gob area.

10.2.1 Methane chemistry

Methane is a colorless, odorless, tasteless, and flammable chemical compound with the chemical formula CH_4. Methane is the second most abundant GHG accounting for about 15% of global GHG emissions produced by human activities and is responsible for more than a third of total anthropogenic radiative forcing. Methane is more than 20 times more effective at trapping heat than CO_2, making it one of the most potent GHGs.

Methane is lighter than air, having a specific gravity of 0.554. It is only slightly soluble in water. It burns readily in air, forming CO_2 and water vapor; the flame is pale, slightly luminous, and very hot. The boiling point of methane is $-161.5°C$ (258.7°F) and the melting point is $-182.5°C$ (296.5°F) [5]. Methane is not toxic when inhaled, but it can produce suffocation by reducing the concentration of oxygen inhaled. In general, methane is very stable, but mixtures of methane and air, with the methane content between 5% and 15% by volume, are explosive. Explosions of such mixtures have been frequent in coal mines, causing many disasters worldwide. It is worth mentioning that methane content more than 15% may not necessarily be explosive.

Even though methane-air mixtures under 5% are not explosive, a considerable margin of safety must be provided (usually to less than 2%) due to the compounding effect of coal dust. An effective mine ventilation system will therefore ensure that the volume of gas mixture is minimized below the explosive range in a safe level (usually less than 1% or up to 1.25%). In addition, due to the fact that inert gases such as nitrogen or CO_2 cannot chemically react with methane, they can be added to an explosive methane-air mixture to make it nonexplosive [4].

Methane, which is also known as methyl hydride, is a group 14 hydride and the simplest alkane, a series of hydrocarbons. It is the main constituent of natural gas. In coal mining, methane is the main constituent of marsh gas (swamp gas) and fire-damp (flammable gas), and may be captured commercially from gaseous coal seams before, during, or after mining. Therefore, unlike other GHGs, methane is the primary component of natural gas and can be converted to usable energy.

Methane is commercially synthesized by the distillation of coal and by heating a mixture of carbon and hydrogen. It can be produced in the laboratory by heating sodium acetate with sodium hydroxide and by the reaction of aluminum carbide (Al_4C_3) with water. Reactions of methane with chlorine and fluorine are triggered by light.

Environmentally, methane is considered a short-term radiative forcer, meaning that it has a relatively short (approximately 9–15 years) lifespan in the atmosphere. While methane does not linger as long in the atmosphere as CO_2, and is emitted in smaller quantities than CO_2, its ability to trap heat in the atmosphere or to absorb the sun's heat, which is called its global warming potential (GWP), is more than 20 times greater than that of CO_2 [6]. Over the period of its short lifespan, methane is 84 times more potent as a GHG than CO_2 [7].

10.2.2 Methane emission sources

Methane is mainly emitted during the production and transportation of natural gas, oil, and coal. Nonetheless, coal mining is not the major source of methane emissions. CMM from underground mining operations is typically vented or flared; whereas in surface mining, it is released directly to the atmosphere. However, underground mining can produce substantially greater levels of CMM than surface mining because deeper coals are under greater pressure and can hold more methane. Some CMM remains in the coal after mining and is released by subsequent processing and transportation during postmining activities. CMM emissions from abandoned mines are not quantified, but may be significant in some cases.

In addition to coal mining, related activities such as the extraction and processing of natural gas as well as the handling of coal at coal-fired power plants and coal processing plants result in the release of significant amounts of methane into the atmosphere. Emissions also result from the decay of organic matter in municipal solid waste landfills, some livestock manure storage systems, and certain agroindustrial and municipal wastewater treatment systems. However, without more stringent measures to reduce sources, methane emissions are expected to increase approximately 45% to 8500 million metric tonnes of CO_2 equivalent ($MMTCO_2E$) by 2030 [6].

By 2020, global methane emissions from active coal mines alone are estimated to reach nearly 800 $MMTCO_2E$, accounting for less than 10% of total global methane emissions. China leads the world in estimated CMM emissions with more than 420 $MMTCO_2E$ (more than 27 billion cubic meters annually) in 2020. Other leading global CMM emitters are the United States, Russia, Australia, Ukraine, Kazakhstan, and India [8].

10.2.3 CMM emission models

The flow of methane through coal seams substantially differs from the gas flow mechanisms of conventional gas reservoirs. CMM transport in coal has three distinct properties: desorption from coal surfaces, diffusion through the coal matrix, and flow through the coal seam fracture system. The two more common methods used in modeling the rate of CMM emissions are Fick's and Darcy's laws.

In order to perform a mathematical analysis using Fick's method, the coal seam is assumed to be composed of small spheres, at the surface of which diffusion occurs. Fick's law describes diffusion from spheres along the concentration gradient as:

$$\frac{\partial C}{\partial t} = D\left(\frac{\partial^2 C}{\partial r^2} + \frac{2\partial C}{r\partial r}\right) \tag{10.1}$$

where C is CMM concentration in coal in ft^3/ft^3, t is time in second, D is coefficient of diffusion in ft^2/s, and r is radial coordinate in ft. Solving this equation for various initial and boundary conditions results in different solutions that can be used to model CMM emissions.

In Fick's law, the size of the hypothetical sphere where diffusion takes place is important since it determines the degree of fragmentation of the coal seam. In addition, combining the radius of the sphere (a) with the coefficient of diffusion (D) results in a new diffusion parameter D/a^2 that determines both the rate at which a coal seam will diffuse methane and the fraction of seam gas content that can be drained in a given time.

On the other hand, Darcy's dynamic flow equation describes gas transport through the fracture system in coal where the driving force is the pressure gradient. In fact, Darcy's law holds that the flow rate of CMM through a porous medium is proportional to the potential or pressure gradient, which is simplified for low pressures in homogeneous medium, linear case, and laminar flow as:

$$\frac{\partial^2 P^2}{\partial x^2} = \frac{\mu\phi}{kP} + \frac{\partial P^2}{\partial t} \tag{10.2}$$

where P is gas pressure in atm, x is distance into coal seam from face in ft, μ is absolute viscosity of methane in lb mass/ft s, ϕ is pseudoporosity, k is permeability of coal in millidarcies, and t is time in second.

According to these equations, it is apparent that the net rate of CMM emission under normal conditions is a function of reservoir pressure, permeability, coal gas content, porosity, and diffusivity.

10.2.4 Methane's importance in industry

Methane offers a unique opportunity for the coal industry to proactively address the climate change issue and simultaneously increase available energy supply. The relative abundance of methane on Earth makes it an attractive fuel, though capturing and storing it poses challenges due to its gaseous state under normal conditions for temperature and pressure. Capturing CMM has the potential to be a cost-effective method to reduce GHGs, increase energy security, provide coal mines with economic and environmental benefits, recuperate ventilation air quality, and improve miners' safety.

On the other hand, methane is an important source of hydrogen and some organic chemicals. Methane reacts with steam at high temperatures to yield carbon monoxide and hydrogen; the latter is used in the manufacture of ammonia for fertilizers and explosives. In the chemical industry, methane is a raw material for the manufacture of methanol (CH_3OH), formaldehyde (CH_2O), nitromethane (CH_3NO_2), chloroform (CH_3Cl), carbon tetrachloride (CCl_4), and some freons. The incomplete combustion of methane yields carbon black, which is widely used as a reinforcing agent in rubber, the principal component of automobile tires.

As a whole, methane is not a bad gas in and of itself, but mankind is pushing more quantities of it into the atmosphere than ever before. Understanding where it comes from and developing technologies to mitigate adverse effects will play an important role in maintaining the Earth's climate for the future. Therefore, modern technologies need to be developed and employed to remove fugitive CMM from underground coal mines and to use it in profitable and practical ways. Beside various other industrial uses, CMM can potentially be utilized as a clean energy source supplied directly to neighboring communities as town gas or used to generate electricity at electric power plants built adjacent to mines.

10.3 Methane emission control in coal mines

CMM emissions into mine workings essentially result from pressure differences between gases trapped in the coal seam and the atmosphere of the mine. In the past century, much effort has been expended to control CMM emission in underground coal mines. The earliest attempts were to control CMM accumulations in active workings by mixing it with air and ventilating it to the outlet roadway or tailgate. In recent years, efforts have mainly focused on coal seam degasification and gas drainage systems to collect CMM from active coal mines at the surface or in advance of mining underground. These efforts have led to using CMM to supplement mine ventilation systems and other commercial uses.

Mechanization and increasing productivity, especially in longwall coal mining, have led to greater volumes of CMM emissions. In longwall mining, it is necessary to control CMM emissions at the face area, at T-junctions, and along gate roadways and rooms. In this respect, having knowledge of geological and mining conditions such as coal rank, gassiness of the coal seam and rock formations, productivity of the coal mine, mining method, depth of mining, and rate of advancement can be an enormous help in controlling excess gas emissions and consequently preventing disasters and explosions. It is important to mention that achieving and maintaining these high levels of productivity requires proper control of CMM.

Methane in coal mines will always be a hazard, but the risk of explosion has been greatly minimized by increased safety regulations, sensitive gas detectors, improved ventilation, and methane drainage systems. Considerable research has gone into effectively and commercially controlling CMM in underground coal mines. The main technique used for controlling CMM concentrations is ventilation; however, other emission control measures such as horizontal and crossmeasure (inclined) boreholes, gob wells, and vertical degasification wells have also been developed and are currently used in mines with high methane emissions. In underground coal mining, as CMM emits into the mine area from the cracked coal or rock strata, timely monitoring of hazardous gases and proper implementation of well-designed ventilation and methane drainage systems must go hand in hand with taking steps to reduce frictional ignitions and educating all miners regarding safety regulations. When this is done, even highly gassy mines can be safely and successfully operated.

10.3.1 Ventilation control measures

CMM emissions can adversely affect both safety and productivity of underground coal mines. Ventilation is the universal methane control technique used in underground coal mines where it is essential to circulate fresh air across active faces using mechanical fans. Therefore, underground coal mining requires carefully designed and well-maintained ventilation systems that can control large amounts of methane and dust produced during coal extraction. In this respect, it is necessary to implement systems that closely monitor CMM levels to ensure that methane concentrations are kept at low levels.

According to figures obtained by the National Renewable Energy Laboratory [9], the average amount of CMM released from surface and underground coal mines in the Illinois Basin of the United States are 1.91 and 4.23 g of methane per kilogram of mined coal, respectively. This region is not known for being gassy. Therefore, face ventilation systems at underground coal mines must be able to safely handle gas flows by focusing on the peak CMM levels, not overall levels.

To reduce CMM concentrations within the mine, methane is diluted by fresh air that is circulated in sufficient quantity by the ventilation system. By mixing CMM with ventilation air, VAM results. The maximum concentration of methane in mine air is restricted to 1% (or at maximum up to 1.25%) in roadways where personnel travel, and less than 2% in the face area. In order to keep CMM concentrations below these acceptable levels, usually more than 10 t of fresh air have to be circulated through the mine for every tonne of coal mined. This, in turn, results in venting large quantities of VAM with methane concentrations typically under 1% that are currently exhausted to the atmosphere at most mine shafts worldwide. An alternative is to use VAM by feeding it into a boiler and generating heat or power. Such an option might be attractive at mines that generate their own power.

As mentioned, the density of CMM is roughly half that of air, which results in a buoyant high-concentration layer at the mine roof that does not readily mix into the ventilation air stream. This phenomenon is largely a result of inadequate ventilation. Depending on the emission rate, entry width, seam slope, etc., an air velocity of 0.4–0.8 m/s (1.3–2.6 ft./s) measured at the mine roof was enough to prevent CMM layering at that point. In the absence of other means to promote mixing, raising air velocity is a highly effective way to reduce CMM ignition risk. Higher air velocity promotes better mixing in addition to lowering the average CMM concentration. CMM layering is not as critical at the mine floor or rib since the gas readily mixes into the ventilation air stream, losing its buoyancy [4]. However, it is important to manage velocity at

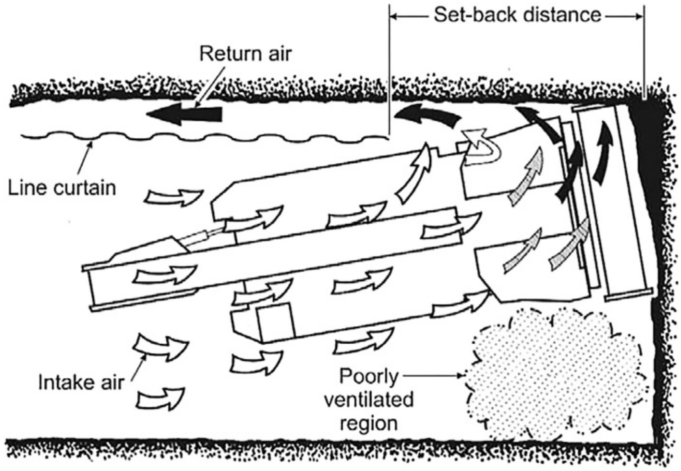

Fig. 10.2 Ventilation setback distance in exhaust systems near a continuous miner [4].

certain locations in the ventilation system. As an example, the United States Mine Safety and Health Administration (MSHA) has regulations requiring exhaust systems to have a maximum setback distance of 3 m (10 ft) to prevent CMM concentration at the face (Fig. 10.2).

The productivity and safety of longwall mining can far exceed that of room-and-pillar mining; however, total CMM emissions per volume extracted by shearers in longwall panels are generally higher than those for continuous miners in room-and-pillar mines [10]. Therefore, wide mine faces and high production rates characteristics of longwall mining present unique ventilation problems. For this reason, a number of supplemental operations such as degasification and drainage systems are designed to reduce the amount of CMM emitted at the face during longwall mining.

10.3.2 Degasification/drainage control measures

Mine ventilation is not the only method of CMM control in underground mining, particularly in very deep or gassy mines. In fact, using a ventilation system alone to control CMM emissions in deep or gassy mines would be prohibitively expensive, and it is necessary to supplement the ventilation system with a degasification system consisting of a network of underground and/or surface boreholes. A methane drainage system is unavoidable when the ventilation system cannot dilute CMM emissions to a level below statutory limits. In such cases where explosive CMM levels are likely, CMM can be drained prior to, during, and even after mining operations in order to mitigate unwanted disasters within the mine. These systems have the added benefit of reducing environmental concerns with regard to GHGs.

Coal mine degasification was originally developed to improve worker safety in mines. However, apart from alleviating environmental concerns and enabling productivity improvements, capturing CMM has many economical benefits that can justify and offset investment costs. International studies determined that 30%–40% of all coal mines produce suitable CMM that can be effectively used for power generation with gas engines. Therefore, drainage systems can drain and produce commercial CMM as well as reduce and prevent health and safety risks.

The purpose of methane drainage is to capture CMM in high purity form. It is frequently performed in one of two ways. First, in advance of mining, boreholes are drilled into the unmined seam (Fig. 10.3) and in situ CMM is collected. Second, after coal exploitation, boreholes are drilled into the gob area (Fig. 10.4) and liberated CMM is collected. The former is called a “predrainage system” meaning that CMM is drained ahead of mining. The latter is called a “postdrainage system” meaning that CMM is drained after mining. Pre- and postdrainage systems can be performed by both in-seam horizontal or crossmeasure boreholes drilled underground from active workings or by vertical wells drilled from the surface.

As shown in Fig. 10.4A, it is possible to combine both pre- and postdrainage approaches. In this scenario, the drainage system consists of drilling underground horizontal or crossmeasure boreholes from the mine workings into the unmined coal seam or into gob areas. As horizontal boreholes degasify the unmined coal seam, crossmeasure boreholes may also be drilled to effectively degasify the fractured rock strata above the coal seam in the gob area. Underground boreholes are typically 10–100 m (33–330 ft) in length, and within a single mine several hundred boreholes

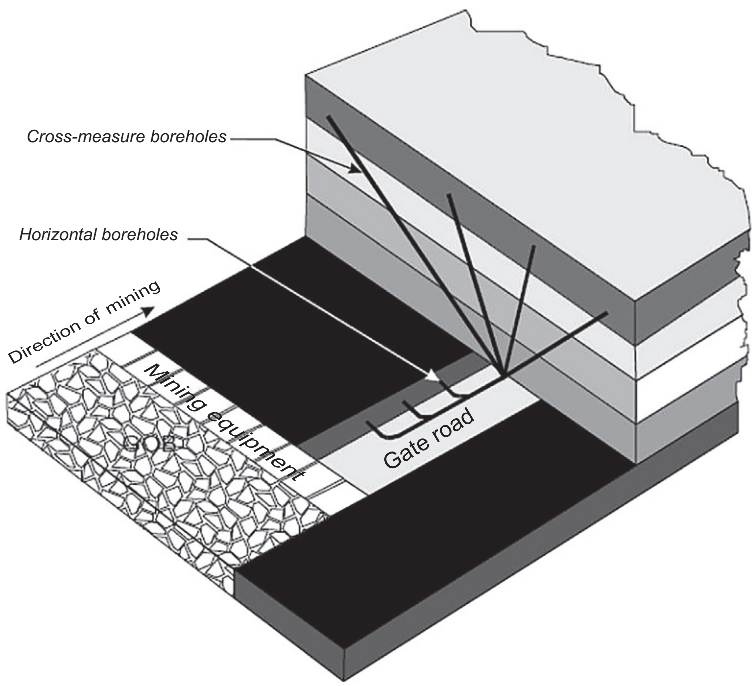

Fig. 10.3 Schematic of predrainage system using horizontal or crossmeasure boreholes [11].

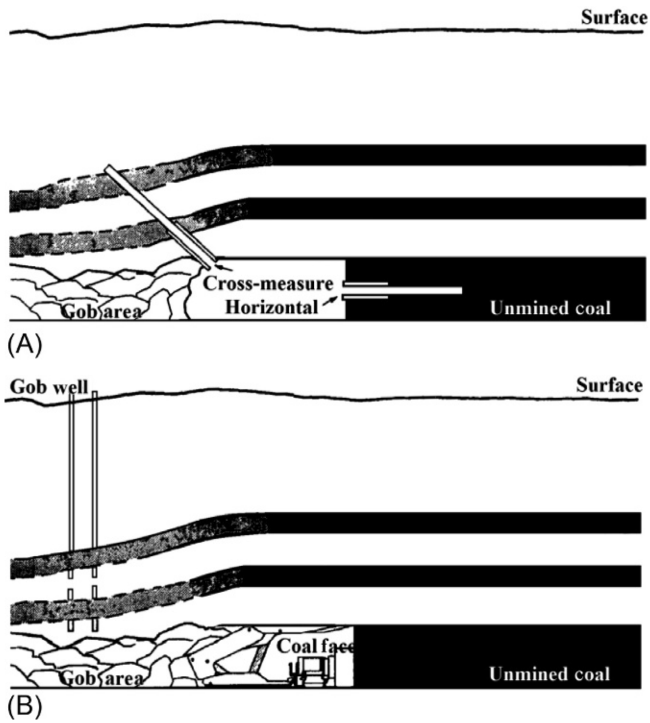

Fig. 10.4 Schematic of postdrainage system using (A) postdrainage crossmeasure along with predrainage horizontal boreholes, and (B) postdrainage vertical gob wells (not to scale) [12].

may be drilled. The boreholes are connected to an in-mine vacuum piping system, which transports released CMM out of the mine.

When a coal seam is fully extracted by longwall mining or after pillar extraction in room-and-pillar mining, the immediate roof strata tend to fracture due to stress concentration and coal extraction resulting in caving of the overlying rock strata [13]. This caving releases CMM into the gob area, which can be extracted through gob wells as shown in Fig. 10.4B. Fractured roof strata in the gob area are a significant source of CMM; and in deep, gassy mines, the ventilation system is unable to sufficiently dilute CMM emitted from the gob area into mine workings. Actually, the CMM that originates and accumulates in the gob area above the mined-out panel is the main source of methane emissions during longwall mining. In these situations, vertical wells are drilled from the surface to drain CMM from the gob area. As shown in Fig. 10.5, these wells are generally drilled to a point 2–15 m (7–50 ft) above the coal seam prior to the mining. As the mining face advances under these wells, methane-charged coal and strata around the well will fracture, which increases permeability of any gas-bearing strata. Then, using a vacuum system, CMM emitted from the fractured strata flows into these gob wells and to the surface. The rate of CMM emission in the gob area mainly depends on the rate of advancement, geological conditions, panel size, gas

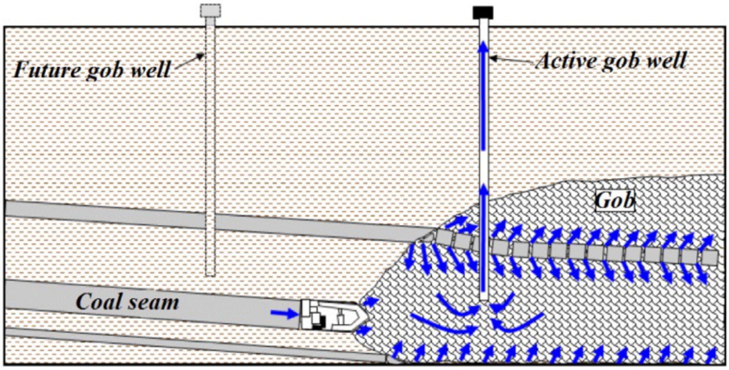

Fig. 10.5 Development of vertical gob wells in advance of mining [14].

content, and coal seam thickness. Overall, methane concentrations in gob gas vary from 30% to 90%.

Predrainage systems can also be combined with postdrainage vertical gob wells as an interesting commercial CMM capturing method. In this situation, CMM is first captured through a predrainage system in regions where methane flows exceed the capacity of the mine ventilation system. When CMM emissions remain high even after face advancement, a postdrainage system is then used to capture gases emitted in the gob area. As depicted in Fig. 10.6, pre- and postdrainage systems are combined to capture CMM in gassy coal seams prior to mining, capture CMM from fractured coal seams and rock strata during mining, and capture CMM accumulations in the caved gob area after mining. The predrainage system can potentially recover as much as 90% of the CMM with composition more than 90% pure methane, which is desirable for injection to pipelines. CMM in the gob area is less pure, but is of sufficient concentration to power fans and pumps used to create the vacuum needed to extract it. Therefore, there is a strong motivation for maximizing CMM capture through a combination of both

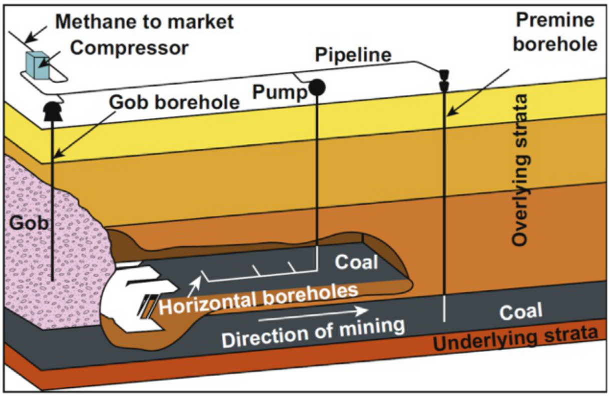

Fig. 10.6 Combination pre- and postdrainage system in an active underground coal mine [15].

pre- and postdrainage systems to achieve enhanced safety, environmental mitigation, and energy recovery.

Selection of horizontal boreholes, cross-measure boreholes, vertical boreholes, or gob gas vent holes is somewhat dependent on the gassiness of the coal seam and the fractured rock strata. The number of boreholes, their locations, and their degasification durations can be changed based on site-specific factors [16].

Degasification techniques are mainly dependent on reservoir properties of coal seams being mined. Good methane control planning depends on accurate information on these reservoir properties and the total gas emission space created by mining operations [4]. Reservoir properties are highly dependent on the depth and rank of the coal seam, which are good indicators of the gassiness, but direct measurement of gas content (amount of gas contained in a tonne of coal) is highly recommended. In general, reservoir properties governing CMM emissions can be divided into two groups [4]: (i) properties that determine the capacity of the seam for total gas production; e.g., adsorbed gas and porosity, and (ii) properties that determine the rate of gas flow; e.g., permeability, reservoir pressure, and diffusivity of coal.

Thakur et al. [2] summarized the advantages for coal seam degasification as follows:

- reduced CMM concentrations in the mine air leading to improved safety;
- reduced air requirements and corresponding savings in ventilation costs;
- faster advance of development headings and economy in the number of airways;
- improved coal productivity;
- additional revenue from the sale of CMM;
- additional uses of degasification boreholes; e.g., water infusion to control respirable dust;
- exploration of coal seams to locate geological anomalies in advance of mining; and
- liberation of CMM into the atmosphere is avoided.

10.3.3 Auxiliary control measures

Ventilation has long been the primary means of controlling CMM emissions at the mining face. However, as mining has progressed into gassier areas, supplemental means have become of interest for continued safe and productive mining operations. Ventilation models use predictions of CMM inflows in face areas and gate roadways to help design systems that will prevent unwanted disasters. To develop these predictive data, CMM is monitored by means of intermittent sampling with portable methane detectors and continuous monitoring with machine-mounted methane monitors.

Aside from improving the ventilation to reduce CMM accumulations in eddy zones, the chance of a methane ignition can be reduced by directly addressing the ignition source. When a shearer cutter bit strikes rock, abrasion from the rock grinds down the rubbing surface of the bit, producing a glowing hot metal streak on the rock surface behind the bit. The metal streak is often hot enough to ignite methane, causing a so-called frictional ignition. There are many well-known methane ignition sources in coal mines, ranging from frictional ignitions caused by cutting bits to electrical sparks, roof falls, aluminum impacting on iron, smoking materials, explosives and detonators, spontaneous combustion, and naked flames. In addition, Kissell [4] investigated other less-recognized ignition sources such as hot solids, thermite sparking from light metal alloys, adiabatic compression, static electricity, lightning strikes, and sliding friction between blocks of rock or between rock and steel. In general, technical instructions and regulations must be strictly followed during mining operation to avoid unwanted disasters and to minimize the possibility of human error. For instance, the surface temperature of permissible electrical and diesel engines should not exceed 150°C (302°F). Also, in very gassy longwall panels, plows are less likely to produce sparks than shearers, and the shallow cut made by a plow releases less CMM per pass than that of shearer.

The spray fan system is an auxiliary ventilation system that makes use of the air-moving ability of water sprays. Moving droplets in the spray with a suitable water pressure and an appropriate spray location will drag the surrounding air forward to create a considerable airflow.

According to Kissell [4], CMM ignitions at longwall faces may be controlled by:

- installing methane detectors in critical locations to prevent CMM accumulations;
- installing water sprays behind each cutter bit to quench the hot metal streak;
- providing better ventilation around the shearer to control methane build up;
- mounting arranged water sprays on the shearer to eliminate eddy zones;
- replacing worn bits regularly;
- changing the attack and tip angles of conical bits; and
- using radial bits instead of conical bits.

10.4 CMM utilization

In underground coal mines, CMM has the potential to be extracted before, during, and after mining as a valuable byproduct. Consequently, the mining industry's interest in recovering CMM for sale or for on-site utilization is increasing.

CMM is most often used for power generation, district heating, or boiler fuel; but it can also be used as town gas or sold to natural gas pipelines. The production of gas from coal has a long history and town gas, which is a potent mixture of methane, hydrogen, and carbon monoxide, and was commonly used as a domestic fuel until natural gas became widely available. CMM can be injected into natural gas pipelines if the infrastructure is available, since CMM is usually sweet (not sour like natural gas) as it does not contain hydrogen sulfide (H_2S). However, enrichment of the gas may be required before CMM is considered pipeline quality.

CMM can also be used to generate power using a number of technologies, including gas turbines, internal combustion engines, and boiler or steam turbines. This on-site capability is valuable because the mining operation needs electrical power to operate machinery and ventilation fans, coal cleaning plants, coal dryers to remove moisture, and other surface facilities. Ventilation fans at an underground mine can consume more than 70% of the total electricity used at the site.

There are a number of practical technologies for liquefying and compressing natural gas as LNG or CNG for vehicle fuel. In some countries, proper technologies that can be economically applied for CMM projects are under construction. As an interesting example, there is one vehicle fuel project at the Furong Mine in Sichuan, China, which uses CMM to fuel buses.

CMM can be used in boilers for space and water heating. For example, CMM has the potential to be used onsite or nearby a gassy mine for residences that require hot water. Furthermore, it is desirable to heat ventilation air in the winter before pumping into the mine. Conversely, heat exchangers may be used to cool the air in deep, high-temperatures mines.

CMM may also be used as a chemical feedstock, such as in methanol or carbon black production. Other practical applications are coking coal development, fuel for aluminum hydroxide roasting furnace systems and glassworks factories, a fuel source for fuel cells, and a feedstock for producing dimethyl ether (C_2H_6O) [17]. In addition, CMM is sometimes used in coal preparation plants to fuel thermal dryers that heat the air used to remove surface moisture from coal.

Hence, capturing CMM not only mitigates concerns for climate change, but also delivers other important cobenefits, including improvement of mine safety and productivity, localized energy production, and improvement in local air quality. Moreover, CMM has the potential to provide a cleaner burning fuel for use at the mine or for sale.

Specific CMM end uses depend on the gas quality, especially the concentration of methane and the presence of other contaminants. One problem with CMM is great variability in its composition; VAM may contain 0.1%–1% methane, whereas CMM drained from the unmined seams prior to mining may contain 60% to more than 95% methane. CMM drained from fractured formations above gob area may also contain 30%–95% methane depending on borehole locations and other operational and completion parameters [18]. Since pipeline grade natural gas must be at least 96% pure methane, lower-quality mine gas must be upgraded for distribution by removing water and inert gases.

Technology is now readily available to recover high-quality CMM that can be used as fuel while simultaneously reducing mining hazards. In 2008, over 200 CMM projects developed in 14 countries capturing more than 3.5 billion cubic meters (124 billion cubic feet) per year [19,20]. Hence, CMM recovery is a clean technology that can reduce mining costs and make operations much safer and more economical by turning a safety hazard into a valuable energy resource.

10.5 CMM abatement

As a whole, methane emissions related to coal mining can be categorized into five groups based on emission sources: (i) vented VAM, (ii) AMM that is seeping out, (iii) surface mine emissions, (iv) emissions from degasification systems, and (v) fugitive emissions from postmining operations. The greatest amounts of post-mining methane emissions occur when coal is crushed and sized, which results in increased surface area allowing the methane to rapidly desorb and be emitted to the atmosphere especially during transportation.

Methane accounts for 20% of global anthropogenic GHG emissions, and coal mines constitute 8% of methane emissions or about 400 $MMTCO_2E$ annually [1]. The capture of CMM provides major benefits while mitigating environmental risks. CMM drainage at first began as a technology for improving the safety and productivity of underground coal mining by preventing explosions. Not only does it provide the same service now, but it also decreases GHG emissions from coal mines and mitigates air pollution because it is a clean-burning fuel.

Coal mine gases that affect air quality when naturally liberated are CH_4, CO_2, CO, and H_2S. Added to these is coal dust, nitrogen oxides (NO_x), sulfur oxides (SO_x), and diesel particulate matter [21]. Capturing and recovering CMM using drainage systems when methane content is above 25% by volume is normally feasible; however, feasible or not, degasification is necessary in sections with over 8 m^3/min (283 ft^3/min) of CMM emissions.

A major concern is that even after coal mines are shut down, CMM continues to be released. It is worth mentioning that CMM is the riskiest gas released from abandoned underground coal mines because each tonne of coal removed can result in about 15 t of CMM emissions. Therefore, it is important to control AMM emissions to less than 1% by volume in order to minimize the risk of GHG emissions and explosions in abandoned mines. The methane content of AMM ranges from 30% to 80%; however, it typically contains no oxygen and its composition changes slowly. The main hazard in abandoned coal mines is accumulation of AMM in discarded underground mining structures. Falling barometric air pressure causes expansion of methane, which can result in AMM overflow. Therefore, applying suitable mine reclamation strategies is essential to abate any destructive events. Filling of near-surface mine openings and sealing by injections along with degasification systems can mitigate the risk of AMM explosions or emissions to the atmosphere. Actually, many factors can affect potential AMM volumes, including time since abandonment, gas content and adsorption characteristics of the coal seam, methane emission rates during active mining, mine flooding, presence of vent holes, and mine seals.

Flaring of CMM or VAM is an abatement option that may be considered if it is not feasible to utilize recovered gas, mainly due to low concentrations of methane or unattractive markets. With the proper equipment and procedures, unused drained CMM can be safely flared to minimize GHG emissions, because converting CH_4 to CO_2 lowers the greenhouse effect. Based on the overall oxidation reaction of CMM when CH_4 is fully oxidized, burning 1 kg (2.2 lb) of CH_4 produces 2.75 kg (6 lb) of CO_2. In other words, when 1 kg (2.2 lbs) of CH_4 is mitigated, 20.25 kg (45 lbs) of CO_2 emissions are reduced in terms of the GHG impact. In addition, flaring converts methane with an average GWP of 25 to CO_2 with a GWP of one.

It is important to dilute emitted CMM with low methane content through mixing with the ventilation air. Despite low methane concentrations (1% or less), collectively VAM is the single largest source of CMM emissions globally, so there is concern with just venting it to the atmosphere. When methane content is high, CMM should be captured by drilled boreholes to enhance the ventilation system and secure miners' health. In fact, there are many varieties of respiratory disorders that can result from the inhalation of CMM and coal dusts in underground coal mines. The more common disorders usually observed in coal miners include pneumoconiosis, lung cancer, chronic obstructive pulmonary disorder, and asthma [3].

Thermal oxidation technologies have been introduced at commercial scales to abate VAM emissions or utilize VAM to produce electricity. However, current VAM technologies are generally not able to process methane concentrations below 0.2% without use of additional fuel to augment the methane content. Other technologies to mitigate VAM emissions include catalytic oxidation, lean fuel combustion, and rotary kilns. These are emerging and under development [1]. In some countries like the United States and Australia, VAM is recovered in commercial scale using either a thermal or catalytic oxidation technique that produces no flame, yet in addition to abating VAM emissions, they can produce thermal energy or electricity using a steam turbine. Although VAM is generally less than 2%, it is possible to produce electricity at concentrations more than 0.5% by circulating water through the oxidizer and capturing superheated steam, which may be used to power a steam turbine.

Increasing atmospheric concentrations of methane will have important implications for global climate and perhaps for the stratospheric ozone layer and background levels of tropospheric ozone. In fact, the GWP of CH_4 is at least 84 times greater than CO_2 over a 20-year period. However, the GWP decreases to around 25 times greater when calculated over a 100-year period. CMM drainage systems are an effective way of reducing CMM from mining operations. The 200-plus CMM recovery and utilization projects described earlier also serve as key abatement strategies worldwide. For instance, one underground coal mine in the United States employing a CMM drainage system will reduce emissions by more than 453,500 metric tonnes/year CO_2 equivalent, which equates to taking approximately 100,000 cars off the road [22].

10.6 Economics of CMM recovery

Key factors that may affect the economics of CMM recovery are: (i) quantity and quality of CMM, (ii) capital and operating costs of the project, (iii) elasticity of demand and selling price for recovered CMM, and (iv) availability of environmental benefits and credits. Although the economical analysis of CMM recovery is case based, preliminary cost-effective analyses indicate that under certain conditions, methane recovery is economically attractive to the mining industry. In addition, there are some environmental benefits to abating methane as doing so will reduce the amount of GHGs. Despite many efforts to put a price on carbon, as yet there are no quantifiable economic numbers associated with these environmental benefits.

Boger et al. [23] indicated three primary reasons for recovering CMM: (i) decreasing explosions within underground coal mines and consequently increasing mine safety, (ii) improving mine economics due to using or selling CMM as a byproduct and reduce production delays, and (iii) mitigating global and local environmental risks associated with carbon emissions.

Effective CMM drainage reduces the risk of explosions and gas outbursts, ventilation costs, and unwanted accidents, and miners' diseases. Reducing these risks in turn diminishes their associated costs, which can usually impose high losses in revenues of typical high-production longwall mines. Additional costs related to CMM emissions within the mines or into the atmosphere that may be avoided are lost production, long- or short-term delays, legal costs, worker's compensation, punitive fines, and even mine closure. With CMM recovery now a practical and affordable way to reduce GHG emissions, improve safety and productivity, and enable coal operators to profit from on- or off-site CMM utilization, what was once a waste product and solely a miner's curse is now a valuable byproduct, provided that it can be properly controlled and managed.

In evaluating the economics of CMM recovery, it must be recognized that in order to establish a new drainage system in a mine, a major investment is required to cover cost items such as the CMM extraction plant, surface facilities, drilling, piping, pumping, installation of drainage equipment, and on-line monitoring systems. Although capital costs for CMM are somewhat less than those for CBM, they are still substantial and depend on a number of factors, including depth of mining, distance between mining districts to extraction plants, technical layout (such as pipe diameters, optimal borehole spacing, etc.), method of monitoring and control, and site-specific geological and mining conditions [24].

In the case of VAM, local electricity sales prices and persuasive governmental support may be the determining factor for economical feasibility of generating electricity from VAM. In cases where CMM or VAM cannot be economically recovered and used due to impractical site-specific conditions or unattractive markets, it may be possible to generate revenue from carbon credits through destroying CMM or VAM by converting CH_4 to CO_2.

10.7 Conclusion

CMM is stored within coal as a result of the coalification process whereby plant material is converted to coal. Due to coal-mining activities, pore pressure decreases and trapped methane is released from the coal and surrounding rock strata into the mining atmosphere. This leads to the build-up of CMM in mines, which potentially creates an explosive hazard. Ventilation or degasification systems are used to prevent CMM accumulation and to ensure its release to the atmosphere for safety reasons. CMM is therefore both a safety hazard and a potent GHG. On the other hand, CMM has the potential benefit of being used as a clean fuel.

The main technique for controlling CMM concentrations is ventilation. Despite low concentrations of methane in vented mine air, collectively VAM is the single largest source of CMM emissions globally. In order to prevent emissions into the atmosphere, VAM released through ventilation shafts can either be destroyed by converting CH_4 to CO_2 (e.g., flaring), or be captured and compressed for commercial uses (e.g., electricity generation).

CMM can be drained prior to, during, and even after mining operations in order to mitigate any unwanted disasters within the mine and reduce environmental concerns with regard to GHG. Recovered CMM is most often used for power generation, district heating, or boiler fuel; but it can also be used as town gas or sold to natural gas pipelines.

Investing in CMM drainage systems is a practical and affordable option that results in less downtime, safer mining environments, productivity improvements, and the opportunity to generate additional revenues resulting from utilizing CMM and reducing GHG emissions. Hence, CMM that was once a waste product and solely a miner's curse is now a valuable byproduct, provided that it can be properly controlled and managed. Modern coal mining will truly reach maturity when it recognizes the benefits of adopting a CMM management system that constructively integrates CMM control, utilization, and abatement in such a way that incurred costs are classified as a worthwhile investment.

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Diesel particulate matter: Monitoring and control improves safety and air quality

11

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11.1 Diesel use in mining

Nearly all mine workers are exposed to aerosols—both mechanically generated and from combustion [1–4]. Diesel engine exhaust is a primary source of submicron (particles with diameter $<1\ \mu m$) mine aerosols [5]. Diesel is an efficient fossil fuel and the energy efficiency of diesel makes it an attractive fuel choice for many industrial and domestic applications [6]. The use of diesel equipment in the mining industry is an attractive option not only because of the ability to convert a large fraction of available energy into useable work, but also because diesel engines are fuel efficient, rugged, and dependable. It is very common for diesel engines in heavy-duty trucks to have a life of 1,600,000 km [7]. Generally, in underground mines, diesel equipment provides more flexibility and maneuverability as compared to electric-powered systems. In the United States (US), it is estimated that diesel-powered equipment is used in 14,000 mining operations [6]. Considering the pace of developments in other energy alternatives, it can be assumed that the underground mining industry will maintain its reliance on diesel-powered equipment for the near future [8].

Because diesel vehicles are one of the primary components of underground mining systems, they are the main source of miners' exposure to diesel exhaust aerosols and gases [9]. Diesel engines produce submicrometer-sized carbonaceous aerosols that become part of the respirable and total particulate mass in the mine air [10]. Diesel equipment operators and other miners spend most of their working time within close proximity to this equipment causing their exposure to harmful diesel exhaust mixtures. In view of the large population of workers who are exposed to DPM in underground mines, DPM has become a subject of increased concern. Documents, which support Mine Safety and Health Administration (MSHA) regulations, clearly demonstrate that underground miners and other personnel who work in confined spaces are exposed to higher DPM than any other occupation [11,12]. Underground miners' exposure to diesel exhaust can be 100 times more than its normal environmental concentrations and 10 times more than concentrations present in other diesel engine work environments [13,14].

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11.2 DPM characteristics

DPM is a general term used for submicrometer aerosols, which are emitted by diesel engines as a product of incomplete combustion. The US Code of Federal Regulations [15] (30 CFR Parts 7, 36, and 72) describes the composition of diesel engine exhaust as a complex mixture of several compounds, which contains both particulate and gaseous fractions. In any mine, the exact composition of diesel exhaust is variable. The physical, toxicological, and chemical properties of DPM are controlled by the type/design of engine, engine life, engine maintenance, engine tuning, equipment operator, type of fuel used, load cycle, and exhaust after-treatment devices. In addition to these factors, different environmental settings in which diesel engines are used also affect the gaseous and particulate matter composition of diesel engine exhaust. In a DPM mixture, the gaseous concentrations of diesel exhaust are oxides of nitrogen, carbon, and sulfur; alkenes and alkanes; aldehydes; and monocyclic and polycyclic aromatic hydrocarbons; whereas particulate constituents are diesel soot and other solid aerosols, including metallic abrasion particles, ash particulates, silicates, and sulfates [16].

The major particulate fraction of diesel exhaust consists of very tiny individual particles with a solid elemental carbon (EC) core that absorbs many toxic substances. In general, DPM is primarily composed of an EC core and other organic and inorganic aerosols [17]. EC core particles are slowly covered by a thin layer of volatile material [18–20]. The organic carbon (OC) fraction of DPM forms different compounds with OC differing from EC because it is composed of volatile and semivolatile organic material. >1800 different organic compounds have been known to adsorb on an EC core. A part of these organic fractions result from incomplete fuel combustion in the diesel engine and are formed when lubricating oil is not completely oxidized during the process of combustion [21]. Diesel particles also contain a fraction of nonorganic absorbed compounds [16]. The process of formation of EC during combustion and expulsion is mainly governed by temperature, oxidant availability, and residence time [22].

In terms of size, aerosols contributed to the environment by diesel engines are typically polydispersed and log normally distributed in one, two, or even three distinctive modes: [18,23] (a) nucleation mode (3–30 nm), (b) accumulation mode (30–500 nm), and (c) coarse mode (>500 nm). The approximate geometric mean diameters are given in parentheses, but all of these modes are more appropriately defined by their distinct nature than by their fixed-size boundaries [18]. The residence time of diesel aerosols in the atmosphere depends on the size and concentrations of other particles in the air. Aerosols between 100-nm and 10-μm have the longest residence time, which is typically about 1 week [18]. The typical residence time for 10-nm particles is about 15 min. These particles primarily coagulate with larger particles from accumulation, coarse modes, and dust [17].

11.3 DPM health effects

Health concerns related to DPM are a relatively new concept in mining, beginning in the late 1970s to early 1980s [24–27]. As per MSHA, all diesel aerosols can be classified as respirable aerosols [17]. Particle size and distribution has a major impact on the transport of aerosols and ultimately on the health endpoints associated with exposure. This is primarily because the deposition efficiency of the particle in the respiratory tract depends upon particle size [28–31]. Coarse-mode aerosols are mainly deposited in the anterior nose and extrathoracic regions of the human respiratory tract. Due to the small particle size, diesel aerosols are mainly deposited in bronchial, bronchiolar, and alveolar regions of the human respiratory tract. The nucleation and accumulation-mode diesel aerosols readily penetrate into the alveolar regions where gas exchange occurs [31].

A harmful aspect of the use of diesel as a fuel is its resultant emissions as an adverse environmental agent. DPM is a major cause of a large number of occupational diseases [32]. Diesel emissions contain various respiratory irritants in both the gas and particulate phases and can cause various acute effects. Published studies [33–35] report the severe effects of DPM exposure in which particular concern is the reported chronic health effects of DPM exposure. Diesel particles are very small in size and generally less than a micron [18]. Because of their small size, DPM particles penetrate deep into the human lungs [17]. Long-term and continuous exposure to DPM can result in severe health issues, which include respiratory diseases, lung cancer, reduced lung capacity, pneumonia, and heart disease [26,33,34,36]. In occupational settings, human epidemiological studies have demonstrated an association between increased lung cancer rate and diesel exhaust exposure [37–39]. The National Institute of Occupational Safety and Health (NIOSH) regards diesel exhaust as a "potential carcinogen," and states that reductions in workplace DPM exposure reduces cancer risks [33]. The International Agency for Research on Cancer (IARC) has declared that "diesel engine exhaust is carcinogenic to humans" [40]. A reported study suggests that the risk of lung cancer among workers that are heavily exposed to respirable EC (between 640 and 1280 $\mu g/m^3$) was five times more than the risk in the lowest exposure category ($<20\,\mu g/m^3$) [38]. In addition, acute exposure to diesel exhaust can also cause deleterious health effects like eye and nose irritation, headaches, nausea, lightheadedness, vomiting, numbness, and asthma [41,42]. Other diesel exhaust exposure effects include bronchial irritation, cough, phlegm, and neurophysiologic symptoms [43].

11.4 Regulatory impact on underground mines diesel equipment

MSHA regulates the usage of diesel equipment in US underground mines. MSHA initiated rulemaking regarding DPM in order to reduce the DPM exposure of underground miners. Based on mining commodities, MSHA has divided the mining industry into two separate sections for regulation purposes: (a) coal mines and (b) metal-nonmetal (M/NM) mines.

11.4.1 Impact on underground coal mines

In underground coal mines, MSHA imposes distinct requirements for diesel-powered equipment usage. This is due to the possible presence of coal dust, explosive gas mixtures, and other related safety matters. MSHA also requires that any diesel equipment used in an underground coal mine must be approved by MSHA. Underground coal mine diesel equipment has been categorized by MSHA into two types: Type (I) permissible diesel equipment and Type (II) nonpermissible equipment.

Type (I) permissible equipment is required in those underground mines that potentially have methane gas and/or coal dust explosive mixtures. Permissible equipment is mainly used in areas of the mine that are inside the last open crosscut (called inby areas). Permissible equipment consists mainly of heavy-duty (HD) production equipment that must be explosion proof and have stringent requirements for exhaust cooling systems and surface temperature controls. As of July 2002, according to MSHA, the requirement for each piece of permissible diesel equipment used in any underground coal mine is a maximum emission of 2.5 g/h. of DPM [16]. Most permissible diesel equipment use exhaust filtration systems in order to meet the MSHA requirement of 2.5 g/h.

Type (II) nonpermissible diesel equipment can be used in the areas of the underground coal mines where use of permissible equipment is not essential. Nonpermissible equipment does not require surface temperature and exhaust cooling controls. MSHA further categorizes nonpermissible diesel equipment into two types: (a) nonpermissible light-duty (LD) equipment and (b) nonpermissible HD equipment [16]. As of January 2005, MSHA requires that each nonpermissible HD diesel equipment used in an underground coal mine must emit <2.5 g/h. of DPM and each nonpermissible LD diesel-powered equipment added in an underground coal mine after May 2001 must emit <5.0 g/h. of DPM [16]. Other diesel engine vehicles, which meet the US Environmental Protection Agency (EPA) emission standards, are considered in compliance with LD provisions, even if they exceed DPM emissions of 5.0 g/h. LD diesel engines that meet US EPA standards and emit >5.0 g/h. can be an important source of DPM in the underground coal mine air [17] even though MSHA regulations allow their usage without emission control.

11.4.2 Impact on M/NM mines

After July 2001, MSHA requires that any diesel-powered engine introduced in a metal-nonmetal mine in the US must be either certified to meet or exceed the particulate matter (PM) emission requirements specified by the EPA [32]. This requirement does not pertain to diesel engines used in ambulances or in firefighting equipment as these vehicles are used according to the mine's firefighting and evacuation plans.

11.5 US DPM regulations and permissible exposure limits (PELs)

MSHA published two rules related to miners' exposure to DPM in underground coal [16] and M/NM mines [32]. These regulations, implemented in the two mining sectors, differ significantly in term of implementation and DPM exposure determinations. The DPM regulatory approach implemented in M/NM mines is focused on monitoring DPM exposure in mine atmospheres. They encourage the use of personal protective equipment (PPE) and different administrative and engineering controls. The US DPM

PEL for M/NM mines was implemented in May 2008. MSHA monitor DPM by employing a sampling methodology developed by NIOSH to measure total carbon (TC) concentration [44]. TC is the cumulative mass of both EC and OC. TC is used as a measure of DPM because reported studies have shown that DPM is generally (70%–90%) made up of TC [45–47]. The relationship between EC and OC fractions in untreated exhaust from diesel engines depends on operating conditions, type of engine, fuel composition, and many other factors. The EC fraction in DPM increases with engine speed and operating load. A study conducted in underground metal mines where diesel-powered equipment is extensively used in the mining process reported that EC concentrations average about 75% of TC concentrations; whereas TC concentrations were found to make up, on average, 72% of total DPM concentrations [10,48]. Another study conducted in several underground M/NM mines showed strong linear correlation between EC and DPM and between EC and TC concentrations when Diesel Particulate Filter (DPF) systems were not used [49]. The PEL for TC on an average eight-hour shift basis is 160 $\mu g/m^3$. TC is defined as the sum of EC and OC when both EC and OC are analyzed by using the standard NIOSH 5040 method [32,50].

Contrary to M/NM mines, DPM regulations implemented in underground coal mines are not compliance based, as the primary focus is to control DPM concentration by utilizing different available technologies that reduce DPM at the point of generation. Therefore, MSHA does not enforce DPM monitoring for compliance determinations in underground coal mines.

11.6 DPM exposure measurements

In underground M/NM mines, MSHA and operators routinely measure ambient concentrations and personal exposure to CO, CO_2, NO, NO_2, DPM, and other pollutants to verify compliance with exposure limits [32,44]. Routine measurements of in-use emissions of CO and measurements of personal exposures to CO and NO_2 are a requirement when operating diesel engines in underground coal mines [51].

In order to assess the potential environmental and health impact of diesel emissions, ambient and indoor exposure measurement is primarily used. In occupational settings, three basic types of sampling techniques are used: (a) personal, (b) breathing zone, and (c) general air [52]. In the case of personal sampling, a measurement or sampling device is worn by the miner during his normal work shift. In breathing zone sampling, other individual samplers not worn by the miner measure concentrations from the breathing zone of the miner for whom exposure is to be obtained. These are typically hung from the roof and moved from place to place as the miner's work location changes. Area sampling or mine air sampling is a technique where concentrations are sampled or measured from fixed locations in the mine. In underground M/NM mines, a miner's exposure is usually determined by the personal sampling technique [32,51]. This is due to the nonexistence of an established relationship between area sampling and personal sampling concentrations [52,53]. The following section describes methods used to determine DPM concentrations in US M/NM mines as coal mines do not use ambient exposure monitoring for DPM compliance.

11.6.1 Shift average-based monitoring

11.6.1.1 The NIOSH 5040 method

The most accurate method to determine the concentration of DPM in the mine air is the NIOSH 5040 method. This is an established technique to measure DPM concentration in underground mines. The NIOSH 5040 technique is an analytical method, which is used to measure EC and OC components of a DPM sample collected on a quartz fiber filter [54]. The NIOSH 5040 method differentiates carbon content into organic and elemental components, which makes it more versatile as compared to the other carbon analytical methods. MSHA considers TC to be the most appropriate surrogate for DPM as TC contributes over 80% of the particulate matter present in diesel exhaust [55,56]. Major interferences that are typically found in underground M/NM mines can affect TC analysis. These interferences are mechanically generated dust that may contain OC and/or EC, cigarette smoke, oil mist, ammonium nitrate/fuel vapors from explosive material, and welding fumes [3,32,57]. However, because dust in M/NM mines generally does not have high carbon content, results of TC and EC measurements are not significantly influenced by carbon from mineral dust [58]. Size-selective samplers used during DPM sampling can effectively remove EC and OC contamination from mineral dust, but these are not effective in removing submicrometer aerosols from cigarette smoke or oil mist [54,59]. Avoiding cigarette smoke or oil mist is not always possible when taking personal samples and because these contaminants interfere only with the OC analysis, MSHA proposed using submicrometer EC as a surrogate for DPM [60]. In a typical M/NM mine environment, no other sources of submicrometer EC are known. Thus, EC can be used as a surrogate for DPM [60].

The current DPM sampling protocol requires that both personal and area samples be collected simultaneously over the miner's full shift [44]. The personal sample is used to assess personal exposure to TC and EC. One area sample (or sometimes more than one) is used to assess the relationship between TC and EC. As a part of the alternative procedure to determine TC exposure levels, EC concentrations obtained by personal exposure sampling are multiplied by the ratio of TC to EC obtained from area sample measurements [32,44]. This EC-based approach to determining TC exposures is used to minimize the potential for overestimating a miner's exposures to TC due to various artifacts that primarily affect the measurement of the OC fraction of TC. The TC/EC ratio is determined from one or more samples collected in the main exhaust course of the mine because OC interferences are expected to be negligible in those samples [32,44].

The NIOSH 5040 method requires four sampling components: a specialized wearable pump designed to deliver a constant volume flow with an accurate timing device, tygon tubing, a 10-mm nylon cyclone, and a DPM filter cassette. Field samples are collected by a sampling medium that consists of a 37-mm cassette preceded by a 10-mm nylon cyclone. Air is drawn at a flow rate of 1.7 l per minute using a calibrated constant flow sampling pump through the nylon cyclone. The nylon cyclone has a cutpoint of 4.0 μm and an impactor with a cutpoint of 0.8 μm at a volume flow rate of 1.7 l per min. These settings serve to physically eliminate most of the mechanically generated mineral dust from diesel aerosol. Before sampling, the cassette and cyclone assemblies are connected to the calibrated sampling pump by using plastic tubing. Personal DPM samples are collected by fitting these sampling trains on miners, whereas area sampling is conducted by installing the monitoring setup at desired locations in the mine. After the collection of a DPM sample, the cassettes are sealed and sent to a laboratory for analysis. Cassettes are opened and a 1.5-cm^2 rectangular shaped portion of the filter is removed using a metal punch. This procedure allows three individual sample analyses for each sample collected [61]. In the laboratory, a thermo-optical method is utilized to analyze the sample. In the thermal-optical method, separation of OC and EC is accomplished through temperature and atmospheric control [62,63]. The NIOSH 5040 method is a direct approach to measure DPM; it can quantify organic and elemental carbon at low (5 μg) levels and it is less likely to suffer from interferences by mineral sources or other combustibles. Thus, it can quantify diesel particulates in situations where using other shift average methods may not be suitable [54,61,64].

11.6.2 Real-time monitoring

DPM regulations endorsed in the US by MSHA triggered development of instruments that can estimate DPM exposure in real time. Generally, these monitors use photoacoustic methods or condensation counters to measure respirable combustible dust and then display an equivalent DPM concentration. Although the NIOSH 5040 is a standard method used for DPM compliance determination in US M/NM mines, this method is based on determining shift average concentrations. Thus, it inherently involves an issue of "lag time" since it requires a postcollection laboratory analysis for DPM determination. It may take 2 weeks to get laboratory results and miners could be potentially overexposed to DPM during that time. Like any other shift average-based measurement method, the NIOSH 5040 method is not suitable to detect rapid changes in DPM levels, which may occur over the course of monitoring. In order to determine any change in DPM levels caused by changing mining activities, more than one mine air sample may have to be collected using the NIOSH 5040 method. This may increase the amount of work and cost involved in DPM monitoring. These limitations of the NIOSH 5040 method can be addressed by the use of real-time DPM monitoring devices. Although use of real-time DPM monitors is relatively new in the mining industry, real-time DPM monitors can almost instantly quantify the generation rate of DPM as well as its relative magnitude, and highlight mine situations where DPM levels are relatively high for substantial time periods.

Most real-time monitors (both prototypes and commercial units) developed for determining DPM concentrations in mines have faced serious challenges to accurately estimate desired DPM concentrations. This is due to their increased vulnerability to mine atmospheric conditions like oil mist, mineral dust, presence of moisture, and other particles. Another big challenge in using real-time DPM monitoring devices is their applicability in different mining conditions and the lack of a standard/unified calibration method for these units. NIOSH has been closely involved in the development of various instruments that measure airborne DPM concentration [65]. Although several real-time DPM monitoring devices are now in use, the following section focuses on the FLIR Airtec. This is a commercial real-time monitor that has been substantially tested under different mining conditions and has shown to function satisfactorily.

The Airtec measures the EC component of DPM. It has been tested in the laboratory and in different mining conditions where its performance was reported to be satisfactory [66–72]. The Airtec has four key components: an impactor, a filter, a pump, and an optical measuring circuit. Air is drawn at a set flow rate through an impactor and through a filter within the instrument, collecting EC from the air sample. The light intensity transmitted through the filter is then measured by an optical sensing circuit. Increase in EC accumulation on the filter cassette causes the output voltage to decrease. Laser absorption is related with EC concentration by comparing the drop in voltage due to EC accumulation with a calibration curve. The monitor provides a five-minute rolling average of EC concentrations, which is calculated by recording a data point every minute. This also provides eight-hour time-weighted average (TWA) EC concentrations. Laboratory results showed that mineral dust can interfere with the monitor only in the absence of a size selector; whereas no influence of oil mist or humidity was observed. The monitor has not detected the presence of cigarette smoke unless DPM was already present on the filter; however, in enclosed cabs where the operator was smoking cigarettes, sampling results indicate that smoke has been shown to cause a bias to the measurement of the Airtec. In laboratory testing, the Airtec instrument meets the NIOSH accuracy criteria [71]. Khan and Gillies reported some difference in results obtained by utilizing the Airtec and NIOSH 5040 measurements under similar mining conditions [73]. Khan suggests a modified calibration method for Airtec monitors when used to estimate DPM from exhausts produced in equipment burning a fuel with a high percentage of biodiesel [74].

11.7 Controlling DPM in mines

DPM is essentially one of the critical constituents of the mine air mixture and control of DPM concentrations is an important aspect of mine ventilation. DPM is a complex diesel exhaust mixture that has deleterious health effects and the accurate measurement of DPM is a challenging task. In order to control a miner's exposure to DPM, there is no single best available option. Thus, the best approach is to adopt a combination of all control strategies and technologies that are described in this text. There are many different ways to control DPM in mines. Some approaches focus on the design and types of diesel engines, some highlight the need for adequate ventilation, some enforce the concept of exhaust after-treatment devices, some encourage the use of alternative fuels, and some promote environmental cabs and/or PPE. Different control technologies and strategies adopted in the mining industry to control the concentration of DPM in underground mines are categorized in the following five sections.

11.7.1 DPM curtailment before generation

Control of diesel emission by controlling its generation is an effective strategy to prevent or reduce the underground miner's exposure to DPM. DPM control before its generation can be achieved in the following ways:

- *Engine design*: DPM reduction can be achieved by changing the engine design and manufacturing such engines that promote complete fuel combustion. Different types of engines are currently in use by the mining industry with final Tier IV engines expected to generate minimum DPM concentrations.
- *Improving fuel properties and quality*: DPM emissions in mines can be controlled by improving fuel properties and quality as well as by decreasing the amount of impurities present in fuel.
- *Alternative fuels*: One noteworthy way to control DPM generation is by the use of alternative fuels. Several biodiesel fuels have been found to produce lower DPM concentration than regular diesel, so the use of biodiesel in mines should be promoted.
- *Improving engine repair and maintenance*: DPM emissions from any diesel engine depend upon the fuel combustion efficiency of the engine. A single poorly performing engine can create a huge DPM imbalance in the mine atmosphere. In mines, DPM concentrations can be reduced by maintaining an effective engine maintenance program and by hiring skilled labor as proper repair of engines is critical in controlling DPM emissions.

11.7.2 DPM curtailment at the point of generation (source)

Diesel emissions can be controlled just before their release into the mine atmosphere by utilizing different available technologies. These are lumped into the broad strategy of DPM control at the generation source achieved using various exhaust after-treatment devices.

- *Exhaust after-treatment technologies*: Different exhaust after treatment devices can be used in order to control the DPM concentration. These include diesel particulate filters (DPFs), diesel oxidation catalysts (DOCs), and selective catalytic reduction (SCR). Several after-treatment technologies/devices have been reported for controlling DPM concentrations efficiently.

11.7.3 Engineering controls

Several engineering control are commonly implemented in the mining industry in order to control miners' exposure to DPM. Some of the engineering controls are described here:

- *Dilution by ventilation*: Ventilation of underground mines is an important aspect in any mining operation as it is required to provide a safe and comfortable working environment for miners. McPherson stated that ventilation is the lifeblood of a mine; and underground facilities that require personnel to stay below ground cannot operate safely without an effective ventilation system. In any underground facility where personnel are required to enter, the dilution of pollutants by providing sufficient airflows is a main objective of an underground ventilation program [75]. In mines that are deeper than 60 m, ventilation and air conditioning systems usually rely on mechanical means. Total airflow requirements in a mine depend upon the mine's total diesel operating power, types of engines in use, and the airflow distribution system. Dilution of DPM concentrations by ventilation is the most important and common way to control DPM in mines. A proper ventilation system should be present in any mine to direct the airflow in the mine. Short circuiting of the mine air should be avoided and leakage of mine air through stoppings and mine doors should be kept to a minimum level. A number of metal-nonmetal mines use booster and auxiliary fans to circulate air in different parts of their mines. These fans, if not utilized with proper distribution systems, can be a big contributor to mine air recirculation. As a result, miners assume that air is moving appropriately; whereas in reality, the airflow is simply being recirculated and fresh air is not being supplied to working areas of the mine properly. Recirculation can cause a significant increase in pollutant concentrations in certain parts of mines.

- *Environmental cabs*: Environmental cabs are effective in order to reduce a worker's exposure to DPM. Environmental cabs can reduce DPM exposure from 60% to 80% and reduce miners' exposure to noise and dust. Environmental cabs are useful only for workers who stay inside the cab; those workers involved in mine activities that are executed from outside the cab are not protected by environmental cabs. Studies showed that DPM removal efficiency of environmental cabs can be over 90% if the cab is used properly [68]. The environmental cab's benefits largely depends upon its proper use.

11.7.4 Administrative controls

Various administrative controls can be implemented in order to control DPM emissions in underground mines and to reduce miners' exposure to DPM. Some administrative controls are described as follows:

- *Downsizing diesel fleet*: If possible, downsizing the present diesel fleet by removing underutilized, unnecessary, and high-emitting diesel engines in the mine can help in controlling DPM. This approach can be most effective for LD vehicles, as HD diesel vehicles are usually a necessary component of mine production and other operations.
- *Workers' training*: Training of all parties involved in DPM control efforts is essential. They should have a clear understanding of all possible health hazards and other risks associated with DPM exposure. Everyone involved in this process should be well aware of all factors associated with implementation of DPM control technologies and other strategies. Since aggressive driving tendencies normally result in over fueling of engines, high DPM emissions, and lower engine efficiency; operators should be trained to avoid aggressive driving or operating practices.
- *Limiting engine idling*: Engine idling refers to equipment that is running, but not working. DPM emissions from diesel-powered equipment can be reduced by limiting unnecessary engine idling. In mining operations, engine idling is common. Workers should be educated and well informed about potential impacts on mine air quality from idling diesel equipment. Diesel equipment operators should turn off engines to avoid unnecessary idling.
- *Scheduling*: Scheduling of work activities is another important and effective way to control miners' exposure to DPM. Mine management should try to schedule work activities in such a way that they do not concentrate all HD diesel equipment in a single working stope. In case of difficulty in scheduling, management can consider the use of vehicles operated by remote control as this can reduce operator exposure to diesel pollutants by removing them from the source.

11.7.5 Use of PPE

The use of PPE is a "last line of defense" to control miners' exposure to DPM. It has been estimated that in about 20% of all work institutions, 5% of all US workers wear respirators during some of their job time [17]. Two types of respirators are generally used to protect workers from gaseous and particulate pollutants: (a) air-purifying respirators (APRs) purify air by removing and blocking contaminants from the air, and (b) air-supplying respirators (ASRs) supply clean, pollutant-free air from another source. Although respirators are effective in reducing mines' exposure to DPM and other mine air contaminants, they should be used after executing all other possible available controls.

11.8 Summary and conclusions

DPM-related issues are currently high profile in underground mines worldwide. A number of DPM control measures and ambient monitoring approaches are currently practiced in underground mines. Although shift average-based DPM monitoring techniques have been available for some time, these approaches do not help in gaining a full understanding of DPM levels over short time periods. Real-time monitors can almost instantly quantify DPM levels, produce data required for engineering assessments, and provide detailed information about DPM levels for short- and long-term intervals. Different DPM control technologies and strategies have proven to be effective in reducing miners' exposure to diesel engine exhaust. The most effective approach to control DPM in mines is the adoption of multiple measures, since there is no single measure that completely resolves the issue.

DPM regulatory compliance is a big challenge especially in large underground M/NM mines. In the US, DPM regulations are different for underground coal and M/NM mines. For regulatory compliance, coal mine operators must add a certain amount of fresh air in the mine before adding even a single diesel engine, whereas M/NM mine operators must ensure the presence of good-quality breathing air in all working areas of a mine. Under existing regulatory arrangements, MSHA does not require air quality monitoring in underground coal mines. The expectation is that the addition of a specific amount of fresh air in the mine will provide good-quality breathable air; however, exceptions can occur in the presence of short circuiting and recirculation. In such cases, the addition of fresh air in the mine cannot guarantee good quality air. Although MSHA does not currently measure air quality in underground coal mines, it seems likely that underground coal mines may need to adopt M/NM DPM rules in order to ensure suitable air quality.

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Engineered noise controls for miner safety and environmental responsibility☆

12

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12.1 Background

Despite more than 30 years of noise exposure regulation (i.e., establishing and enforcing permissible exposure limits (PELs)), noise-induced hearing loss (NIHL) continues to be one of the most prevalent diseases in the mining industry. A recent study conducted by the National Institute for Occupational Safety and Health (NIOSH), in which over one million audiograms were analyzed, revealed that the mining industry has the highest prevalence in hearing loss among all other industries surveyed [1]. In fact, the mining sector has the highest prevalence of hazardous workplace noise exposures (76%) among all industrial sectors [2]. Despite engineering and administrative controls implemented to reduce noise, miners continue to exhibit a high prevalence (24%) of hearing difficulty [3]. Within the mining industry, underground coal miners are particularly at risk of noise overexposure due to confined environments in which they work and the close proximity of equipment operators to the mining machines they run. As a result, underground coal miners have the highest self-reported rate of hearing loss. In this context, there is a clear need to develop, evaluate, and implement effective noise controls for various pieces of equipment in order to reduce noise-induced hearing loss in the mining industry.

12.2 Approaches to noise control

In general, there are three approaches to reduce noise exposure levels encountered by miners: (1) controlling noise at the source by making physical changes that modify and attenuate the noise generation mechanism, (2) implementing noise controls in the transmission path between the noise source and the receiver; i.e., the miner, and (3) implementing noise controls at the receiver; i.e., personal protective equipment.

☆The findings and conclusions in this report are those of the authors and do not necessarily represent the views of the National Institute for Occupational Safety and Health. Mention of any company name, product, or software does not constitute endorsement by NIOSH.

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The most preferred approach is the first approach—controlling noise at the source. However, this approach requires a thorough understanding of the noise-generating mechanism, and usually involves modifying this generation mechanism. For example, reducing the airflow speed from an aero-acoustically generated noise can result in significant noise reduction.

When noise control at the source is not possible, then noise controls in the transmission path should be considered. These controls usually have the form of acoustic enclosures absorbing some of the noise and/or isolating the noise source from the receiver (miner). It is very important that these enclosures are designed with heat transfer considerations in mind to prevent overheating of noise-generating components such as electric motors, gear boxes, etc. A drawback of acoustic enclosures in underground environments is the reduced visibility that they can cause, thus creating safety concerns.

Finally, personal protective equipment (PPE) such as ear muffs and earplugs should always be worn when noise levels are equal to or exceed 85 dB(A). A variety of styles and materials for hearing protection devices (HPDs) are commercially available with features that allow them to adapt to being worn with other PPE such as safety glasses and hard hats. Importantly, care should be exercised to guarantee proper fit and consistent wearing of these devices.

All HPDs have a noise reduction rating (NRR) on the packaging, which provides an estimated noise attenuation for that product when it is used correctly. However, although they will protect the worker from hearing damage due to noise, compliance credit is not given by the Mine Safety and Health Administration (MSHA) for their use. In other words, MSHA's rules for compliance do not permit subtraction of the HPD's stated NRR from the actual noise exposure level of the miner. Therefore, although HPDs may in theory reduce the noise reaching the ear of the miner, the employer is still responsible for reducing the noise exposure of that worker, as if HPDs were *not* being used. This is one reason why engineered noise control solutions are still needed despite the wide availability of HPDs.

12.3 Current technology

Advances in different areas of science and engineering provide specific tools that can be used to identify noise sources, develop noise controls, and evaluate solutions both in the laboratory and in underground environments. These advances in technology result in the availability of various types of tools and materials that can be used for purposes of noise control. This section presents some of these tools that have been successfully implemented and used to develop noise controls in the mining industry.

12.3.1 Engineering software for acoustics modeling

Small- and medium-sized mining machines are usually tested in hemi-anechoic and/or reverberation chambers to locate noise sources, and to determine sound power levels, respectively. However, for larger machines, it can be difficult or even impossible to conduct testing in such chambers due to the large dimensions of these machines relative to the dimensions of these chambers. In these cases, tests may have to be conducted in the field, and for mining equipment, this often means testing underground and in confined spaces. For large complex machines, test protocols may become quite complicated with a large number of measurements required. In some instances, it may not be possible to conduct measurements under actual operating conditions. Because of concerns about ignition sources, most acoustic measurement systems cannot be used at the working face of a coal mine. In such cases, modeling of machine dynamics and noise radiation may be the best approach.

Numerical models for dynamic and acoustic prediction constitute the most common means to conduct noise control development for different types of mining equipment. When numerical models of a significant sound-radiating component are created, the first step is to validate them using data obtained from experimental tests. This validation process guarantees that the models are an accurate representation of actual parts/components.

Once validated, numerical models can be used to explore various noise control alternatives readily. Therefore, some of the benefits of using these models involve increasing the efficiency of the process and reducing the cost incurred in the fabrication of physical prototypes that would have to be built and tested if these models were not available.

There are different methods to build these numerical models; however, the most commonly used are the boundary element method (BEM), the finite element method (FEM), and the hybrid finite element/statistical energy analysis (FE/SEA). It is not the purpose of this chapter to elaborate on each of these methods, but rather to provide an overview of the benefits and applicability of each of these methods in the mining industry, as described later. There is extensive literature that explains in detail each of these methods.

In acoustics, the boundary element method provides a way to solve the wave equation by discretizing the boundary and solving integral equations for each element that are mathematically equivalent to the original partial differential equation [4]. The main advantage of this method is that only the boundary of the domain needs to be discretized. This advantage is more significant when the domain is exterior to the boundary, such as in sound radiation and scattering problems. In terms of analysis frequency, the BEM constitutes an effective tool for low to mid frequencies (below 1000 Hz) due to discretization requirements. As a rule of thumb, six elements per wavelength are recommended when using the BEM in order to obtain acceptable results [5].

The finite element method also provides a way to solve the wave equation governing acoustic phenomena. In contrast to the BEM, this method requires that the entire domain be populated with elements. This requirement increases significantly the number of unknowns, especially for three-dimensional problems. However, matrices that arise from this formulation have a sparse structure that simplifies their solution and reduces the computational effort.

The hybrid finite element/statistical energy analysis combines a deterministic method (finite element analysis) with a statistical method (statistical energy analysis).

The hybrid method is described in several papers [6–8]. This particular implementation of the hybrid FE/SEA method specifically involves coupling an FE structural model to a SEA acoustic model. In this case, the acoustic space is an infinite space (as opposed to an enclosed cavity). The structure "feels" the effect of the fluid and radiates sound. The SEA model of the fluid structure interaction allows for a few more approximations than the other methods. Exact approximations depend on how surfaces are meshed. Surfaces are broken up into simply connected regions, called faces. Faces are the key to the hybrid coupling and determine where assumptions are made. Assumptions made on each face are: (1) each face is assumed to be uncorrelated from adjacent faces, (2) the curvature of each face is ignored, and (3) each face is considered to have baffled boundary conditions. Making the faces as large as possible typically makes the analysis more effective, mostly due to the first assumption, but somewhat due to the third assumption. However, sometimes large surfaces of a structure are not well approximated as flat faces. Gentle curves do not present a problem, but some curvatures have an impact. Nevertheless, even with these assumptions, the hybrid method was developed for and should have good accuracy in the midfrequency region (500–2000 Hz). The hybrid method can be as much as two orders of magnitude faster than some of the other methods.

12.3.2 Microphone phased arrays and beamforming

One of the first steps in any noise control program is the identification and location of dominant noise sources. It is critical that control measures attenuate the sound radiated by these dominant sources or they will not be effective. This is especially true in large machines where there are numerous noise sources and where treating a lower-level source may have no effect on the overall noise level.

Microphone phased array (MPA) technology is an effective and very efficient tool to identify the physical location and the frequency content of dominant noise sources in mining equipment. The main advantage of MPA technology is the improved speed for the noise source identification process. In general, it takes a few minutes to collect and process the data using MPA technology, in contrast to several hours or even days of data collection required when using acoustic intensity measurements to identify dominant noise sources.

MPA technology comprises two components: (1) hardware, which consists of an array of microphones distributed in a predetermined pattern and a data acquisition system capable of sampling microphone data simultaneously up to the maximum frequency of interest; and (2) a computational algorithm known as beamforming. This algorithm adjusts the phase of the microphone signals based on a grid of assumed source locations, and a source model. The most commonly used model is that of a monopole source.

In its most simple form, the beamforming algorithm assumes that a sound source (e.g., a monopole source) exists at each grid point location. Then, for each assumed noise source, the time delay between the grid point and each of the microphones is computed. Next, the measured microphone signals are time shifted according to the computed time delays and summed up. If an actual noise source is located at the assumed source location, the shifted microphone signals will be in-phase and the summation of the signals divided by the number of microphones will represent the acoustic signal at the center of the microphone array. However, if no actual noise source exists at the assumed source location, the summation of the time-shifted signals will diminish.

Most beamforming algorithms take advantage of the computational efficiency of the fast Fourier transform (FFT) and thus process the data in the frequency domain [9]. However, for moving noise sources, time domain beamforming algorithms have been traditionally used. Nevertheless, a new technique has been developed to conduct beamforming on moving sources of sound that process the data in the frequency domain and is therefore significantly less computationally intensive than traditional time domain beamforming [10,11]. This technology has been demonstrated to be a very effective tool to identify noise sources in mid- and high-frequency ranges (above 1000 Hz). It can be used at frequencies below 1000 Hz; however, the resolution of the acoustic maps decreases significantly requiring additional postprocessing algorithms. Some of these algorithms involve deconvolution methods [12] while others take advantage of the spatial coherence characteristic of noise sources [13].

12.3.3 Source path contribution analysis

There are many cases where due to the complex machine geometry, large dimensions of a machine, and the presence of different noise sources, it is not clear through what paths noise is being transmitted from source to receiver. In general, sound can be transmitted via structure-borne and/or airborne paths. In this context, a test-based approach known as source path contribution (SPC) analysis has been shown to be very helpful.

Several SPC methods are available and they all fall into one of two categories: (1) the synthesis approach; or (2) the decomposition approach. In the synthesis approach, noise arriving at the receiver is calculated as a sum of the contributions from each source; i.e., source strength multiplied by the transfer function between that particular source and the receiver. These transfer functions are measured experimentally using a volume velocity source at the receiver and microphones at the assumed source locations. Since source strengths cannot be measured directly, they are estimated from measurements at so-called indicator locations; i.e., locations in close proximity (within 2–5 cm) to the assumed source locations. Using the volume velocity source, transfer functions between indicator locations and assumed source locations are measured. Next, this matrix of transfer functions is inverted and multiplied by the vector of acoustic responses measured at indicator locations when the machine is in operation. This product yields a vector of estimated source strengths.

In contrast, the decomposition approach separates the sound arriving at the receiver into a number of components according to some criteria based on a reference signal [14]. Once sources are identified and critical transmission paths determined, then noise controls can be developed to reduce noise levels at the receiver; i.e., the operator location.

12.4 Case studies

In this section, three case studies are presented describing the development of noise controls for underground coal mining equipment.

12.4.1 Noise controls for longwall-cutting drums

Longwall systems are sets of machines that work in full synchrony to extract ore from underground mines. Although there are two basic types of longwall systems—shearers and ploughs—in the United States (US), approximately 98% of longwall mines use shearers. These systems are mainly used in coal, but also a few trona mines. As shown in Fig. 12.1, a longwall system comprises the following components: a shearer that traverses back and forth along the face ripping coal; an armored face conveyor (AFC) that runs along the face and transports the ripped coal to the stageloader; powered self-advancing longwall shields that provide temporary roof support for the shearer and the AFC; and the stageloader, which, after crushing the coal, loads it onto a belt conveyor to be taken out of the mine. The shearer measures from 8 to 12 m in length, and by virtue of its ranging arms can perform cuts of 2 to 6 m in height. Each shield measures from 1.5 to 2 m in width, and therefore on a typical 400-m-long face there are over 200 shields providing temporary roof support. Since the AFC runs along the entire length of the face, typical AFCs can measure 400 m in length.

The longwall shearer is the main component of a longwall system. It is usually controlled by two operators who move along with it as it traverses the face. Its function is to rip the coal and push it into the AFC. In order to effectively accomplish these two

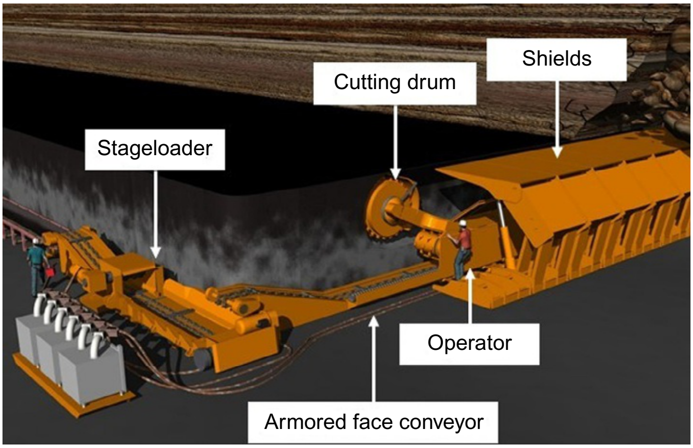

Fig. 12.1 Schematic representation of a longwall mining system showing the location of the shearer operator with respect to the cutting drum.

tasks, the shearer is provided with two rotating cutting drums—the headgate drum and the tailgate drum. These drums are named in reference to their location as being either nearest headgate or tailgate entries, but either can function as the "leading drum." When the shearer travels from headgate to tailgate, the tailgate drum is the leading drum that performs most of the coal cutting, while the headgate drum is mostly in charge of a cleaning operation by pushing coal left by the leading drum into the AFC. When the shearer travels from tailgate to headgate, the drums switch functions, with the headgate drum being the leading drum and the tailgate drum performing the cleaning job.

Noise exposure samples collected from longwall system operators by MSHA between 2002 and 2011 show that approximately 48% of these operators were exposed to noise levels exceeding the permissible exposure level, or PEL [15]. In response to this finding, NIOSH conducted research to develop engineering noise controls for longwall mining systems that would reduce the noise exposure of longwall operators.

Noise assessments conducted by the US Bureau of Mines (USBM) indicated that cutting drums are the dominant sound-radiating components on a longwall shearer [16]. These drums are set into vibration by the excitation forces that arise from the interaction of the cutting bits with the coal and transmitted to the drum through the bit holders. Due to adverse conditions at the face while the drum is in operation, i.e., as the drum is sumped into the coal, vibration measurements are extremely difficult to conduct on an operating drum. In addition, the presence of explosive gases at the face of coal mines, as well as the limited number of instruments approved for underground use in the US, further restricts the ability to perform any type of vibration and/or force measurements. Therefore, in order to reduce the sound radiated by the cutting drums, numerical models of the drums were used to explore the effect of various noise control concepts on the surface vibration and on the acoustic radiation of the drum [17]. Inputs to these models in the form of coal-cutting forces were obtained experimentally using a self-contained, intrinsically safe instrumented bit developed during the course of the project [18].

The longwall shearer-cutting drum examined in this study consists of a cylindrical body with a 0.987-m outside diameter, a 1.067-m height, and a 0.05-m-thick wall. Inside this cylindrical body, there is a circular mounting plate 0.10 m thick having a square opening at the center of the cylinder (refer to Fig. 12.2). The drum is made entirely of steel and weighs 4707 kg. Around the cylindrical body, four helical vanes are welded, starting in the face ring and winding around the cylindrical body toward the discharge side of the drum. The function of the helical vanes is to push the cut coal into the AFC as the drum rotates. The vanes have a 1.91-m outside diameter. On the outermost edge of the vanes, there are 28 bit holders that hold the cutting bits at various angles of attack. There are also 12 bit holders on the outermost edge of the face ring and 4 bit holders in the flange of the face ring, making a total of 44 bit holders. Water is carried through conduits inside the vanes to the bit holders, where the water is sprayed through nozzles to reduce the risk of ignition of mine gases and for dust control purposes.

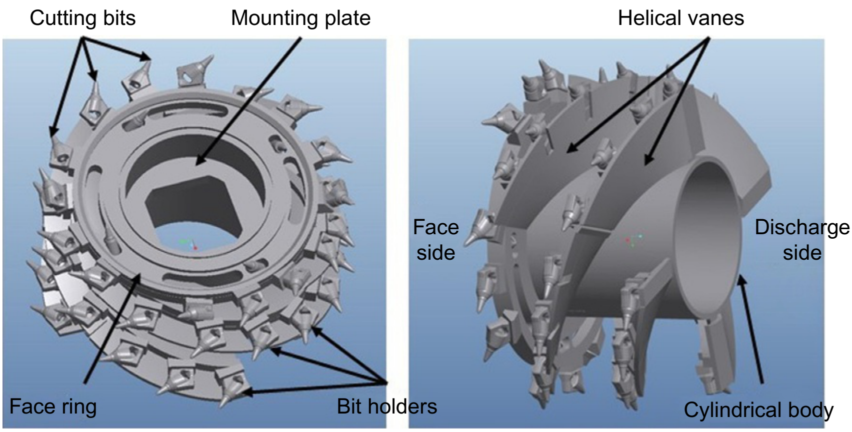

Fig. 12.2 Drawings of a longwall-cutting drum showing its various components.

12.4.1.1 Noise source identification

Although the cutting drums were found to be the dominant sound-radiating components on a longwall shearer, it was not clear what components of the drums were radiating most of the noise. To identify these components, a panel contribution analysis was performed using the boundary element model and dividing it into two parts: the cylindrical shell and the vanes. During this analysis, only one of these parts is assumed flexible while the other is set to be rigid. Analysis results indicated that the four vanes are the critical components of the drum that generate noise of much higher amplitude as compared with the noise generated by the cylindrical shell [19]. However, because of the large dimensions of the vanes, more detailed information was needed.

In order to gain a better understanding of the critical sound-radiating components on the drum, a further panel contribution analysis was performed. In this analysis, the whole drum was divided into three parts: the cylindrical shell face, inner vane segment faces, and outer vane segment faces, as shown in Fig. 12.3.

The excitation was applied in the same manner as in the prior work [20], and the predicted overall sound energy distribution is shown in Fig. 12.4. Similarly, the energy

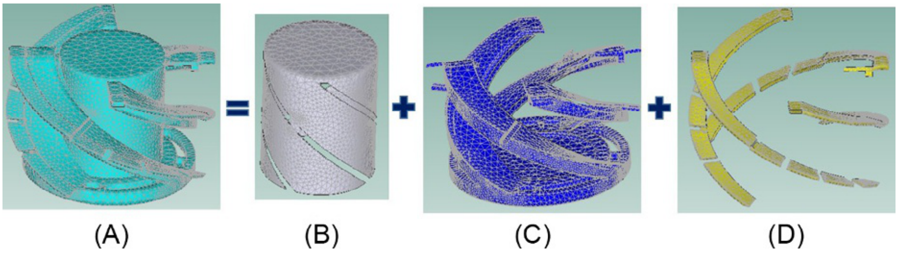

Fig. 12.3 FE-BE faces for: (A) whole drum, (B) cylindrical shell, (C) inner vane segments, and (D) outer vane segments.

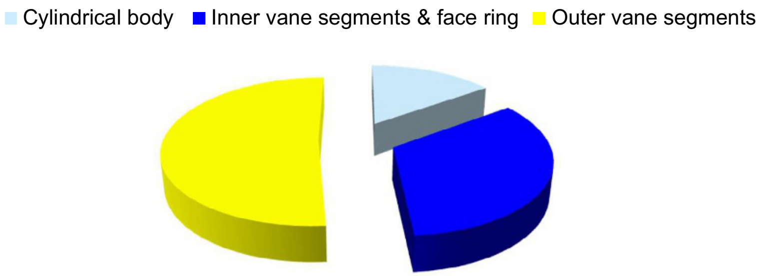

Fig. 12.4 Overall sound energy distribution computed from a panel contribution analysis.

of the noise generated by the vanes, which is the summation of the yellow and dark blue segments, dominates the total noise radiated by the drum. Furthermore, it is observed that the outer vane segments contribute more than the inner vane segments to the total noise radiation [21]. This information suggests potential noise control strategies to reduce the radiated noise.

12.4.1.2 Potential noise control concepts

Validated FE and BEM models of the cutting drum along with operational coal-cutting forces measured with the instrumented bit were used to study three different noise control concepts [21]. At this stage, only the potential of each control concept to reduce the sound radiated by the drum was assessed. This section presents a summary and a brief evaluation of the three noise control concepts that were studied as part of this research.

Force isolation

The force isolation noise control concept aims at reducing the dynamic coal-cutting force being transmitted from the cutting bits to the main drum structure. This noise control concept is schematically shown in Fig. 12.5. In order to isolate the dynamic coal-cutting force, the top layer of the connecting mass block (1-in. or 2.5-cm thickness), shown in Fig. 12.5C, was given the properties of a rubber material. The rest of the connecting mass block shown in Fig. 12.5B, the bit and bit holder system shown in

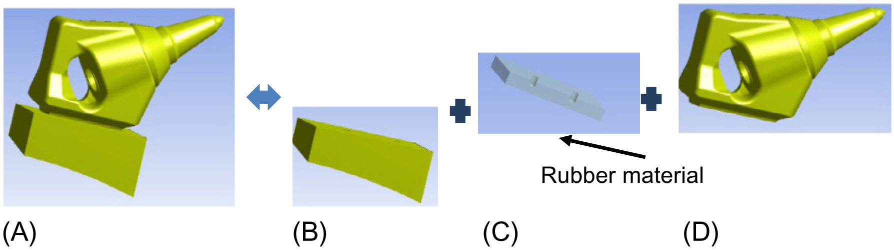

Fig. 12.5 Schematic of the bit isolation concept: (A) bit assembly, (B) connecting mass block, (C) rubber material, and (D) bit and bit holder.

Fig. 12.5D, and the main part of the drum were still given the material properties of steel. For the baseline case, the whole drum was defined as steel.

For the force isolation case, in the frequency range of interest (below 2 kHz), the bit and bit holder vibrate almost as a rigid body with relatively low natural frequencies, due to the flexibility provided by the rubber layer. Meanwhile, the main part of the drum has many flexible modes with relatively high natural frequencies, some of which are significant contributors to the total noise radiation. For frequencies above the highest natural frequency for which the bit and bit holder behave as a rigid body, the force transmitted to the main part of the drum can be significantly reduced, due to the −20 dB/decade slope of the transfer function. However, at frequencies where the bit and bit holder behave as a rigid body, larger forces can be transmitted to the main drum structure due to the resonance.

To reduce the force transmission for all the frequencies, the drum design should be modified so that the highest natural frequency of the bit and bit holder assembly rigid modes is lower than the first flexible mode of the main drum structure. In practical terms, natural frequencies of the bit and bit holder system can be adjusted by using different rubber materials.

In this study, the properties of actual industrial rubber materials were used to evaluate the effect of the bit isolation concept on sound radiation, and significant sound power reduction of up to 25.9 dB(A) was achieved. However, after the authors discussed this concept with cutting-drum design engineers, it was concluded that this concept is not suitable for the cutting drum due to adverse cutting performance and durability issues that the viscoelastic material would pose.

Damping treatment

Experimental modal analysis tests conducted on a newly manufactured drum indicated that the longwall-cutting drum is very lightly damped [17]. A uniform 0.01 loss factor was used for the structure in the structural-acoustic simulation as an approximation of the damping ratio obtained experimentally [20]. Due to the small damping ratio, there are many sharp peaks in the predicted sound power level spectra. Those peaks can be suppressed by increasing the damping ratio of the drum. Therefore, the effect of increasing the damping on the predicted overall sound power level of the noise radiated by the longwall-cutting drum was evaluated using numerical models.

The overall sound power level below 2 kHz, predicted using a uniform 0.01 loss factor, was taken as the baseline. The overall sound power levels for two additional cases—one with a uniform 0.02 loss factor and another with a uniform 0.03 loss factor—were calculated and compared with the baseline prediction. For the 0.02 loss factor and for the 0.03 loss factor, the overall sound power level reductions were 3.3 dB(A) and 5.2 dB(A), respectively.

Despite these reductions, this noise control concept does not constitute a very attractive strategy for the longwall-cutting drum. On the one hand, increasing the damping of the drum would require some type of damping treatments (e.g., attaching a layer of viscoelastic material to the surface of the drum), which, due to the adverse environment, would have durability issues. On the other hand, it would not be practical to treat the whole cutting drum. Performing damping treatments on the outer vane segments, which contribute the most to the total noise radiation, might be much easier and more practical. However, by applying damping treatments to only the outer vane segments, there is a theoretical maximum sound power level reduction, which occurs when the treated components do not radiate any noise. That being the case, the largest noise reduction that could be achieved is approximately 3 dB(A), because the noise generated by the vibration of the outer vane segments accounts for approximately 50% of the total noise radiated by the whole drum, as shown in Fig. 12.4.

Structural modification

The predicted sound power level spectrum has two dominant characteristics that relate to the vibration of the structure, and provide an excellent basis for developing structural modifications for suppressing noise radiation. The first characteristic is that the sound power level has a large amplitude when the direction of the dynamic deformation of the cutting bit either aligns with, or has a large component along, the direction of the excitation force. A straightforward solution to this condition is to minimize the amplitude of the cutting-bit dynamic deformation in the frequency range of interest. This is done by increasing the stiffness of the cutting-bit assemblies. The second characteristic is that the outer vane segment vibration contributes the most to the total noise radiated by the longwall-cutting drum. As a result, reducing the number of outer vane segment modes in the frequency range of interest also reduces the radiated sound.

Helical plates (1×2-in. or 2.5×5-cm cross section) were added to the model to connect bit holders to outer vane segments as shown in Fig. 12.6A. These plates served to stiffen both cutting-bit assemblies and outer vane segments. Stiffeners provide additional support for cutting-bit assemblies, and they also provide T-shaped supports for outer vane segments. Further, these stiffeners connect all bit holder assemblies located on the same vane, which significantly suppresses cutting-bit assembly out-of-phase modes that occur along the vane.

Modal analysis results of the modified cutting drum with stiffeners show that the number of modes in the frequency range of interest (below 2 kHz) was reduced by around 70 from the original 250 modes. For cutting-bit assemblies located on the face ring, there is no vane segment for the bit holder to be connected to. Therefore, an L-shape stiffener, highlighted in Fig. 12.6B, was added for each bit located on the face ring. This approach was taken instead of using continuous plate stiffeners as were used for cutting-bit assemblies located on vanes.

In order to assess the performance of the structural modifications, three different cases with excitations applied at different cutting bits were analyzed. The excitation locations for this analysis are highlighted with yellow circles in Fig. 12.6C–E. For all three cases, the applied excitation consisted of the measured coal-cutting forces [18]. The predicted overall sound power level below 2 kHz was compared with the baseline prediction, and the reduction achieved for each case is given in Fig. 12.7. From the simulation results, it can be seen that a promising sound power level reduction of approximately 3 dB(A) can be achieved by implementing these structural modifications on the longwall-cutting drum.

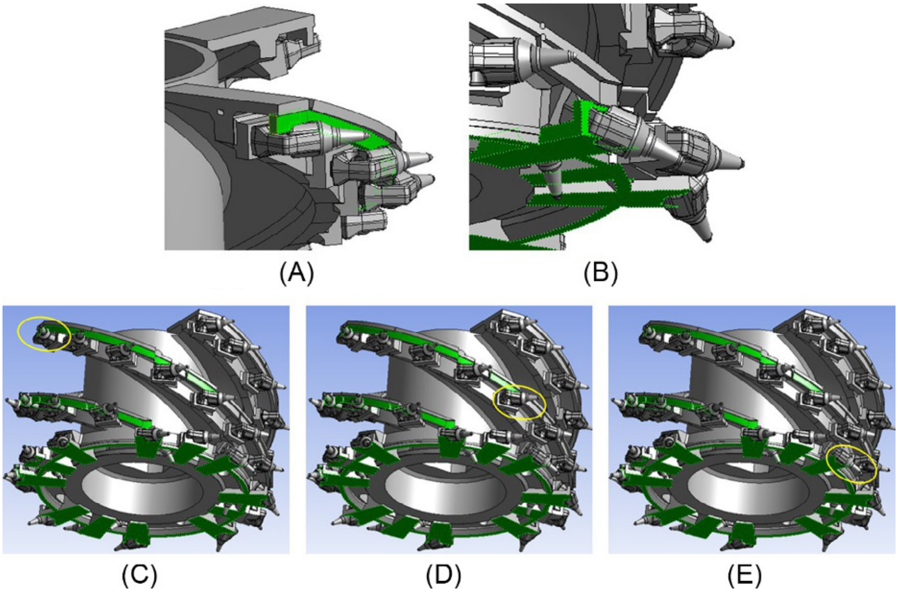

Fig. 12.6 Stiffeners and the location of excitations applied at different cutting bits to estimate the noise reduction yielded by these control concepts: (A) Plate stiffeners on helical vanes, (B) L-shape stiffeners on face ring, (C) excitation near the discharge end, (D) excitation near the center of the cylindrical body, and (E) excitation near the face side of the drum.

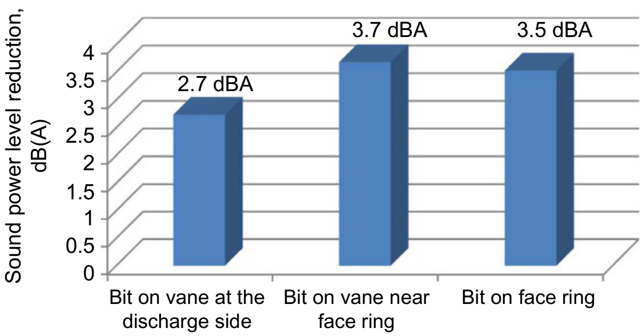

Fig. 12.7 Sound power level reduction for different excitation cases.

12.4.1.3 Constraints

Constraints that had to be observed during the development of noise controls for longwall cutting drums were as follows: First, noise controls should not affect drum-cutting performance. Second, noise controls should not affect drum-loading ability. Both of these constraints are directly related to mine production requirements. In terms of structural modifications, constraints imply restrictions on modification of helical vanes. The third constraint was that noise controls should not compromise the structural integrity of the drum. Changing a drum between scheduled overhauls is not common practice in underground longwall mines. Therefore, most longwall mines do not have spare cutting drums in stock. The fourth and final constraint is that noise controls should be durable enough to withstand rough underground mining conditions.

12.4.1.4 Selected solution

Based on these constraints and upon discussion of the potential noise control concepts with an original equipment manufacturer, it was determined that the most viable solution complying with all constraints was structural modifications to the outermost plates of the helical vanes. However, due to manufacturing and maintenance considerations, this was accomplished by replacing stiffener plates connecting bit holders to outer vane segments with gussets installed behind each bit holder, as shown in Fig. 12.8A. In addition, 1-in. (2.5-cm)-thick steel plates were welded between each gusset and the associated outer helical plate to increase the thickness of these plates, as shown in Fig. 12.8B. A total of eight ribs were used to stiffen the face plate, as seen in Fig. 12.8C.

To evaluate the performance of these controls in terms of noise reduction, a pair of standard drums were built and tested in a laboratory setup to collect baseline data. Then, noise controls were implemented into this pair of drums by making the structural modifications previously described. The modified drums with the implemented noise controls were then tested in a laboratory setup. Comparison of the data collected from the baseline drums and the modified drums confirmed the noise reduction estimated by the numerical models of the cutting drums, and thus validated these models.

The last step of the project involved evaluating the performance of these noise controls in an underground mining environment under actual operating conditions. The objective was to evaluate their effect on the longwall shearer operator's noise exposure with all other noise sources present during normal operation, especially noise

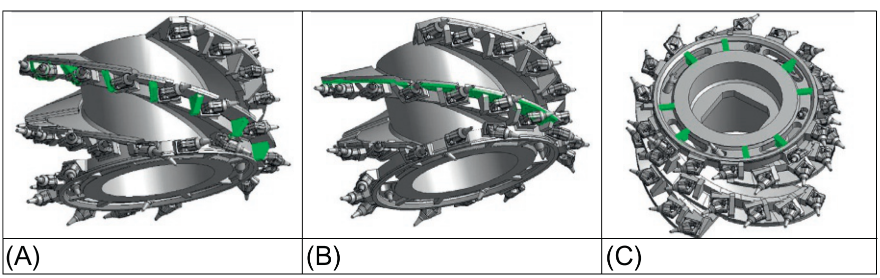

Fig. 12.8 Stiffeners and ribs *(shown in green)* implemented as cutting-drum noise controls. (A) Gussets installed behind each bit holder, (B) 1-in. thick plates between each gusset, and (C) rib stiffeners at face ring.

generated by the interaction of ripped coal with different parts of the shearer and the armored face conveyor, the water spray noise sources, and electric and hydraulic noise sources. This was the second step in validating numerical modeling results for the cutting drum under realistic operating conditions.

Sound levels were recorded using personal dosimeters at the following locations: [1] on the longwall shearer approximately 2 m away from the headgate drum, and [2] on the longwall shearer approximately 2 m away from the tailgate drum. These dosimeters were mounted on magnetic stands installed on the shearer, thus keeping them at a fixed distance from the cutting drums. Additionally, sound pressure data were recorded at the headgate drum operator location using an MSHA-permissible audio recorder at a 44.1-kHz sampling rate, while the shearer was cutting coal from tailgate to headgate. Explosive hazard restrictions prevent the use of sound level meters or more sophisticated instrumentation at the face, which limits the level of detail that can be measured. Comparison of dosimeter data collected in the mine during the use of standard cutting drums, and dosimeter data collected in the mine during the use of modified cutting drums, showed a noise reduction of approximately 2.6 dB (A) at the operator location during an 8-h period [22].

Sound pressure time data were recorded using an MSHA-permissible audio recorder. These data were then converted into the frequency domain using a fast Fourier transform (FFT) algorithm to obtain the sound pressure spectrum. Fig. 12.9 shows the unweighted and A-weighted one-third octave band spectrum of the data recorded at the collaborating mine while operating with the standard (baseline) drums as well as the spectrum of the data recorded while operating with the modified cutting drums. The spectra show a reduction throughout the frequency range. This broadband reduction is a major accomplishment that was not attainable with any other noise control options. In terms of overall sound pressure level, a reduction of approximately 3 dB is observed using the unweighted spectra; however, when these spectra are A-weighted, the overall sound pressure level reduction is approximately 6 dB.

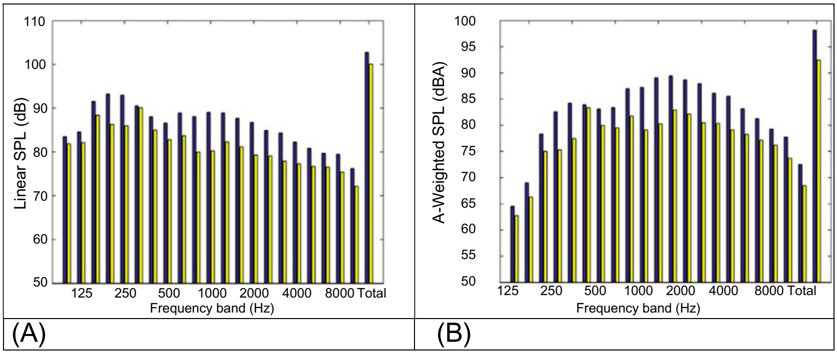

Fig. 12.9 Sound pressure spectra of the standard (baseline) drum in *gray bars* and the modified drum in *yellow bars*, computed from audio recordings at the shearer operator location during normal operation. (A) Linear spectra. (B) A-weighted spectra.

12.4.2 Noise controls for roof-bolting machines

Roof-bolting machines are extensively used in underground coal mines to drill and install bolts for roof support. The majority of these machines involve a manual cycle where the operator inserts a drill steel into the chuck and drills the hole, then inserts an epoxy cartridge into the hole, removes the drill steel from the chuck, and replaces it with a roof bolt tool to install the bolt. These many interactions between operator and machine drilling head require that controls, and thus the operator, be in close proximity to the drill steel.

12.4.2.1 Noise source identification

To identify the location and the frequency content of dominant noise sources during the roof-bolting cycle, a 1.92-m-diameter, 42-channel microphone phased array was used. Testing was conducted in NIOSH's hemi-anechoic chamber with a Fletcher Model HDDR, dual head roof bolter, as shown in Fig. 12.10A. Interior dimensions of this chamber are approximately 17.7 m long by 10.4 m wide by 7.0 m high or approximately 1300 cubic meters. This facility meets ISO 3744 requirements [23] down to approximately 100 Hz. Sound pressure level measurements were also taken at the operator location, as shown in Fig. 12.10B, to determine the overall A-weighted sound level at the operator's ear while drilling.

A large steel support stand comprising rectangular tubes was fabricated by NIOSH to hold the drilling media, shown in Fig. 12.10. To prevent the support stand from radiating significant amounts of sound, sand was used to fill the hollow tubes except for the diagonal tubes and the horizontal tubes along the short direction at the top of the structure. This was done for convenience and to create a vibration impedance mismatch in the structure to reduce vibration transmission.

During the formulation of the test plan, it was decided to use drill bits and drill steels that were representative of industry usage. Therefore, round and hexagonal drill steels were used along with a 34.9-mm drill bit. Granite was chosen as the drilling

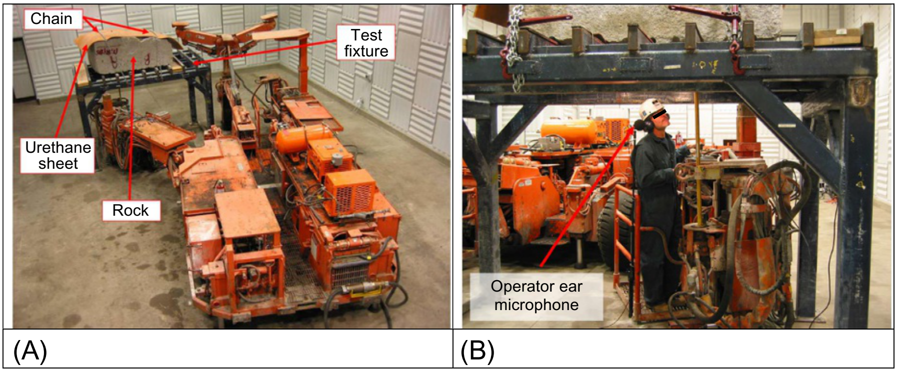

Fig. 12.10 Experimental setup used for drilling tests: (A) roof bolter in the hemi-anechoic chamber, and (B) location of the operator ear microphone.

media because of its high compressive strength. Past research at NIOSH showed that drilling into higher-compressive strength materials generates more noise than drilling into lower-compressive strength materials [24]. Therefore, granite would provide the worst-case scenario for noise emission. Also, a low rotation speed of 200 rpm and a low thrust of 2121 lbs. were used during testing. Previous NIOSH research showed that when drilling into hard materials, lower rotation speeds should be used [25]. The lower thrust was used, so a longer drill time could be obtained.

Fig. 12.11 shows acoustic maps obtained from beamforming measurements for 2500–6300 Hz one-third octave bands. Each figure shows that as the penetration depth increases, the noise source near the top of the drill steel remains in nearly the same location. The source near the bottom of the drill steel travels with the drill head. These results seem to indicate that the mechanisms of noise radiation are independent of drilling depth [26].

Fig. 12.12 shows the one-third octave band spectrum at the position of the roof-bolting machine operator's ear. The overall sound level at the operator's position was 99.7 dB(A). The frequency content of the noise radiated toward the operator is dominated by the 1250–8000-Hz frequency bands.

12.4.2.2 Potential noise control concepts

To reduce the sound level at the operator ear location while drilling, noise controls must target the noise generated at the drill bit and rock interface as well as the drill steel and drill chuck interface. In addition, the controls must address the mid- to high-frequency components of the drilling noise. To reduce the radiated sound at both bit and chuck interface, isolation techniques were used.

Force isolation

A bit isolator was developed to reduce the noise radiated at the bit and rock interface [27]. A chuck isolator was also developed to reduce the noise radiated at the drill steel bit and drill chuck interface [28]. Fig. 12.13 shows the first prototypes built to prove the concept of bit and chuck isolators. Both isolators were designed for noise and

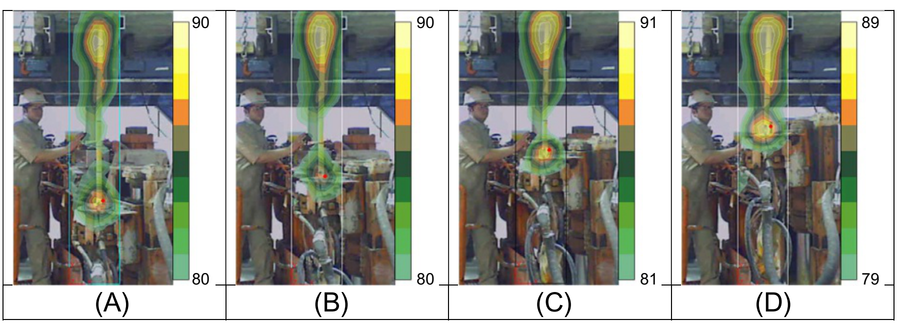

Fig. 12.11 Beamforming results for a 1.2-m hex drill steel at drilling depths of: (A) 0 m (start of hole), (B) 0.15 m, (C) 0.30 m, and (D) 0.45 m.

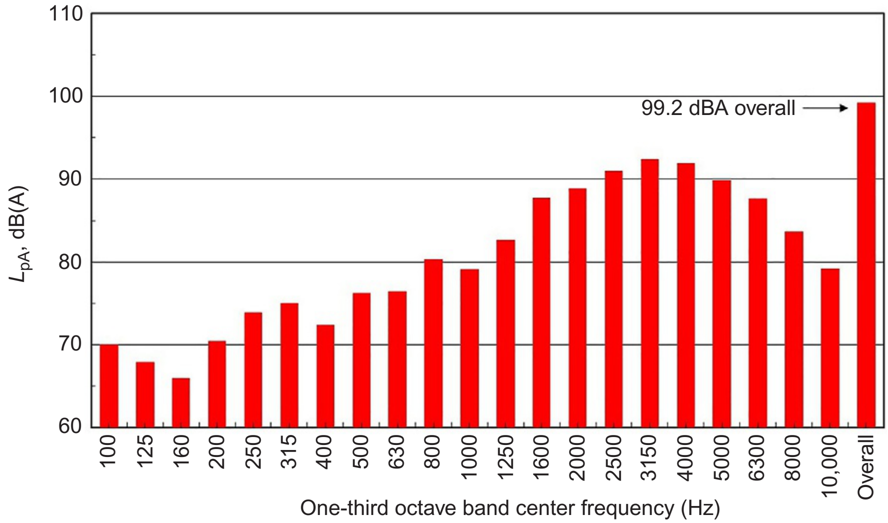

Fig. 12.12 Baseline operator ear sound pressure level with a 1 $^3/_8$-in. (3.5-cm) drill bit.

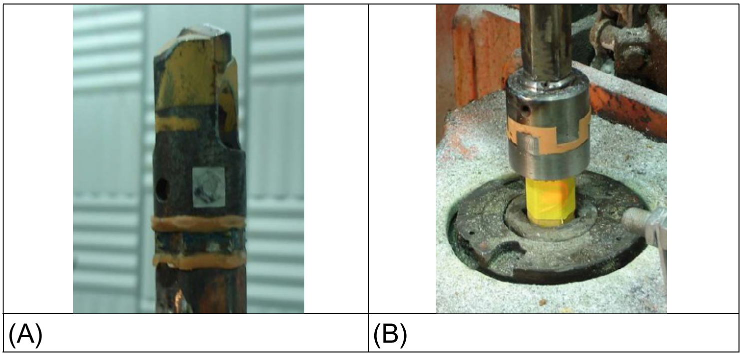

Fig. 12.13 Vibration isolation noise control concepts built to assess potential noise reductions: (A) bit isolator, and (B) chuck isolator.

vibration damping to reduce the noise emitted during drilling by limiting the vibration transmitted down the drill steel from the drill bit/media interface. A urethane material with a durometer of 58 Shore D was chosen for both isolators to reduce the dominant frequency bands between 1250 Hz and 8000 Hz. The chuck isolator used a jaw-type urethane coupler.

Increasing damping

Previous research showed that drill steel/rod vibration is a common source of noise during drilling operations [29,30]. Furthermore, acceleration in a roof-bolting machine drill steel may exceed 500 g, suggesting that this is the cause of significant

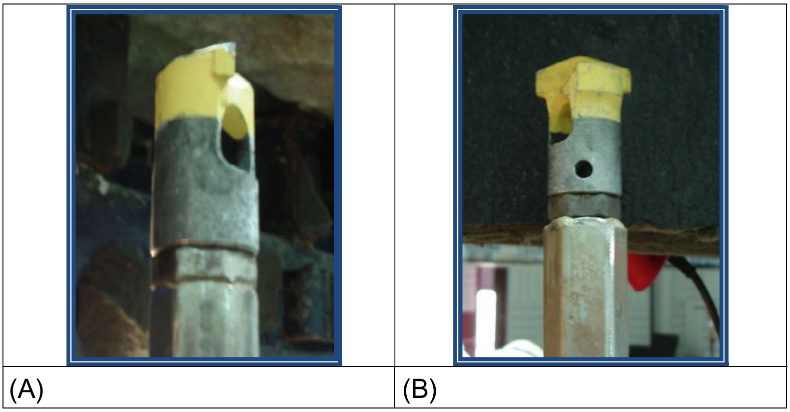

Fig. 12.14 (A) Standard drill steel, and (B) damped drill steel using a constraining layer.

noise during drilling. In this context, a damped drill steel noise control concept was built and tested. To this end, constrained layer damping was used to create a damped drill steel. Fig. 12.14. shows a standard drill steel and a damped drill steel with an outer constraining steel layer. The objective of this damping treatment is to reduce vibration induced by drilling, which, in turn, would reduce noise radiation.

To accommodate for the larger outside diameter resulting from the constrained layer damping treatment on the outer surface of the drill steel, a $1^5/_8$-in. (4.1-cm) bit was used instead of the conventional $1^3/_8$-in. (3.5-cm) bit. At the operator's location, drilling with a standard hexagonal drill steel yielded sound levels on the order of 101 dB(A). Similar data collected while drilling with the damped hexagonal drill steel yielded an average value of 97 dB(A) at the operator position, showing a reduction of approximately 4 dB(A). Fig. 12.15 shows the one-third octave band sound pressure level at the operator location. From this figure, it can be seen that most of the sound is radiated in the 1600–6300-Hz frequency range.

Acoustic enclosure/sound barrier

A third type of noise control concept was investigated using the acoustic enclosure/sound barrier approach. A prototype of a collapsible drill steel enclosure (CDSE) noise control was built to block part of the noise being radiated by the drill steel from reaching the operator. The first CDSE prototype, shown in Fig. 12.16, consists of a round aluminum-coated fiberglass bellows with a spring enclosed inside the bellows. This prototype was mounted on the drill head, near the chuck, and enclosed the drill steel. The purpose of the spring was to keep the bellows raised up when installed vertically on the roof bolter. As the drill head raises to the mine roof, the spring compresses downward, allowing the CDSE to encapsulate the drill steel throughout the drilling process. Fiberglass was chosen because it has good acoustical absorptive properties, has excellent heat resistance, and is incombustible. The fiberglass bellow dimensions were 1.905 cm thick by 19.685 cm outside diameter and an extended length of 1.2192 m.

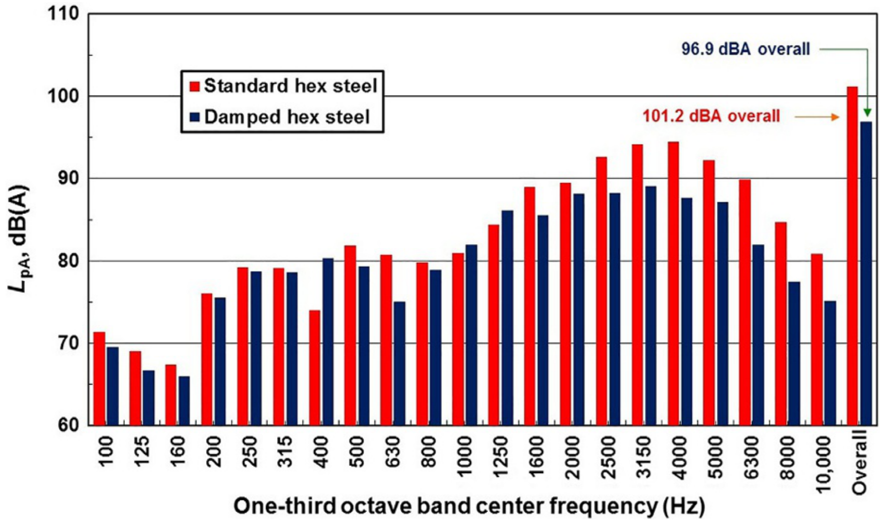

Fig. 12.15 Sound pressure level radiated by standard and damped drill steels with $1^5/_8$-in. (4.1-cm) drill bits.

Sound power levels radiated by a standard drill steel when drilling into granite were compared to the sound power radiated using the CDSE under the same conditions. In addition, tests were conducted with varying gaps between the top of the CDSE and the bottom of the drill media. Varying gaps would determine how tight of a seal was needed between the CDSE and the rock media in reducing sound power levels. Gap lengths of 15.24 cm, 10.16 cm, 7.62 cm, 5.08 cm, and no gap were tested.

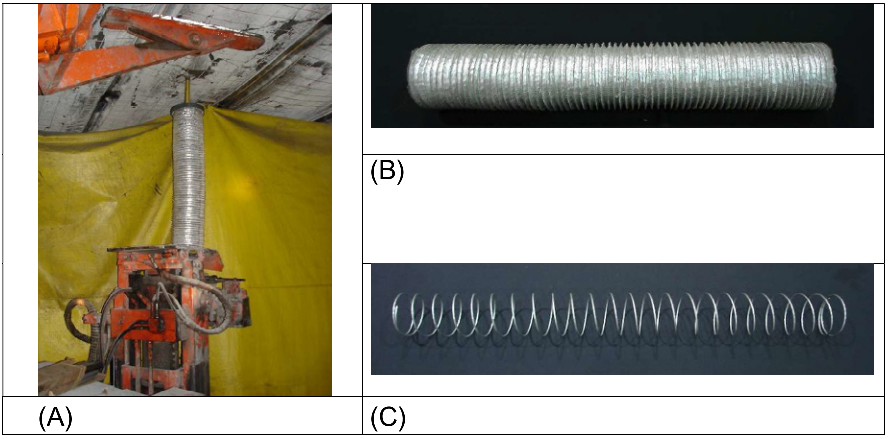

Fig. 12.16 Prototype of the collapsible drill steel enclosure (CDSE) noise control concept: (A) installed on a roof bolter machine, (B) outer layer, and (C) inner spring.

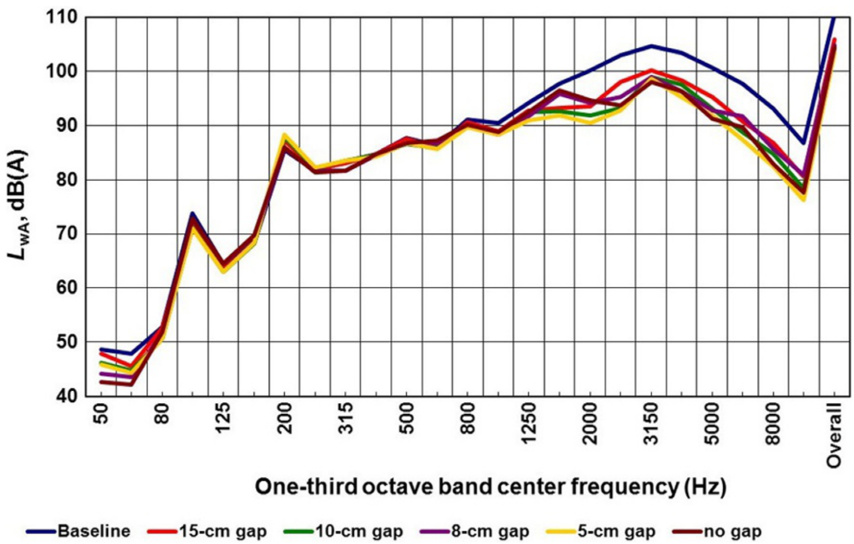

Fig. 12.17 One-third octave band spectrum for all configurations while drilling in granite at 200 rpm and 9.4 k-N thrust.

Fig. 12.17 shows the one-third octave band spectrum for drilling in granite at 200-rpm speed and 9.4 kN thrust for all configurations. From this figure, it can be seen that the CDSE resulted in a sound power level reduction of 6 dB(A) with gaps of 10.16 cm, 7.62 cm, 5.08 cm, and no gap. However, with a 15.24-cm gap, the sound power level was only reduced by 5 dB(A). More generally, Fig. 12.17 shows the effectiveness of the CDSE in reducing the radiated sound power in the frequency range of interest (1250 Hz–8000 Hz).

12.4.2.3 Constraints

The constraints to be observed during the development of noise controls for roof-bolting machines were the following: For the case of the bit isolator, noise controls should have the same outside diameter as that of the drill steel, which is dictated by the bit diameter. The noise controls should also have the same inner diameter as the drill steel in order to allow the flow of dust particles to the dust collector. For the case of the CDSE, noise controls should not block the operator's view of the drill steel. Finally, noise controls should not risk the structural integrity of the drill steel-bit assembly.

12.4.2.4 Most practical solution

The most practical solution complying with all constraints was the drill bit isolator (DBI). After various refining iterations, the final DBI consists of two hollow steel cylinders with a rubber layer between them. A schematic of the device is shown in

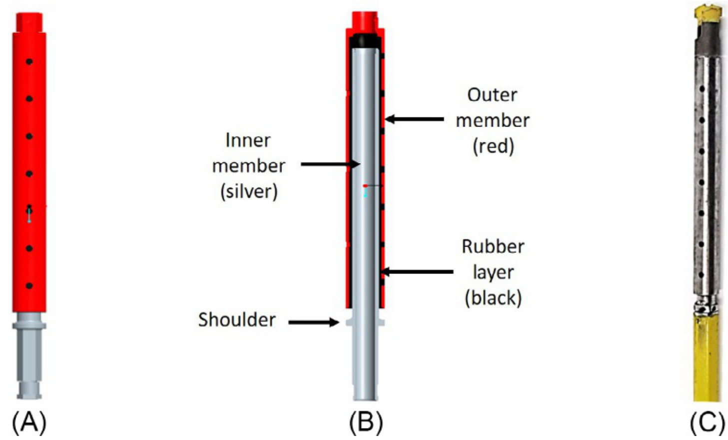

Fig. 12.18 Details of the DBI: (A) diagram of the prototype detailing various components, (B) close-up of inner and outer members showing the location of the isolating rubber layer, and (C) an actual DBI equipped with a 35-mm drill bit and installed on a hexagonal drill steel.

Fig. 12.18. The rubber layer between the inner and outer cylinders isolates vibrations at the drill bit from the drill steel, thereby reducing the noise radiated from the drill steel. This layer is chemically bonded to the steel components to limit torsional travel and produce consistent stiffness. Fig. 12.18B is a close-up view of the inner and outer members and the rubber layer that separates them. The DBI has a drill steel coupling on one end and a bit coupling on the other. These couplings are welded to the ends of the inner and outer cylinders. There is a 0.4-in. (10-mm) gap at the end of the outer cylinder, which is designed to allow for a small amount of relative movement between the layers as axial thrust loads are applied and removed. The gap acts as a safety feature, preventing axial overload by partially closing when thrust is applied and rebounding to the original position when thrust is removed. Fig. 12.18C shows the device installed on a drill steel with a drill bit attached to the end. Minor modifications based on field study results detailed here were incorporated into the final production version of the device.

12.4.3 Noise controls for continuous mining machines

Continuous mining machines (CMMs) are used to extract approximately half of the US underground coal production in room-and-pillar operations, and to develop entries in longwall mines. Unlike longwall operations where the roof collapses after coal is extracted, in room-and-pillar operations coal pillars are left behind for roof support purposes. CMMs are usually operated via a wireless remote control device by a miner who may or may not have a helper. The three main CMM components that radiate noise are the cutting head, the conveyor, and the dust scrubber fan, which are shown in Fig. 12.19. Of these three components, it was determined that conveyor noise is the most important contributor to the total radiated sound [31].

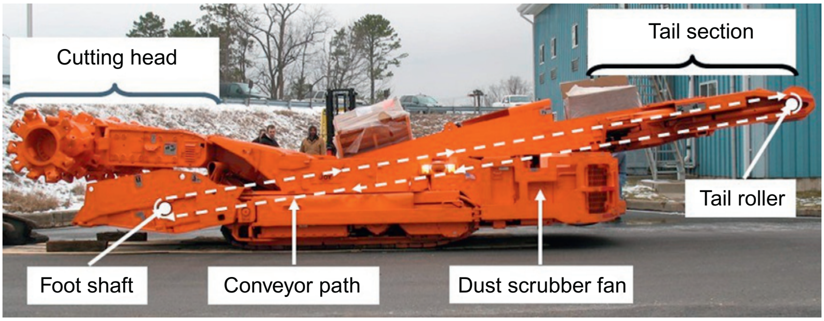

Fig. 12.19 Noise-generating components on a continuous mining machine.

12.4.3.1 Noise source identification

To identify dominant noise sources on a continuous mining machine, acoustic measurements were conducted in a hemi-anechoic chamber at NIOSH. Noise-sensing equipment included a 1.92-m-diameter, 42-element microphone phased array. The CMM under test was a new JOY Model 14CM-15 continuous miner equipped with a 956-mm-wide, 54-flight conveyor chain. NIOSH conducted beamforming testing at the cutting drum and conveyor tail ends of the CMM. The length of the CMM was several times the diameter of the microphone array necessitating a series of data measurements along both sides of the CMM and then above the CMM to evaluate the entire machine. Fig. 12.20 shows the CMM in the hemi-anechoic chamber with the microphone phased array installed above the tail section.

Fig. 12.20 CMM in the NIOSH hemi-anechoic chamber with overhead microphone array.

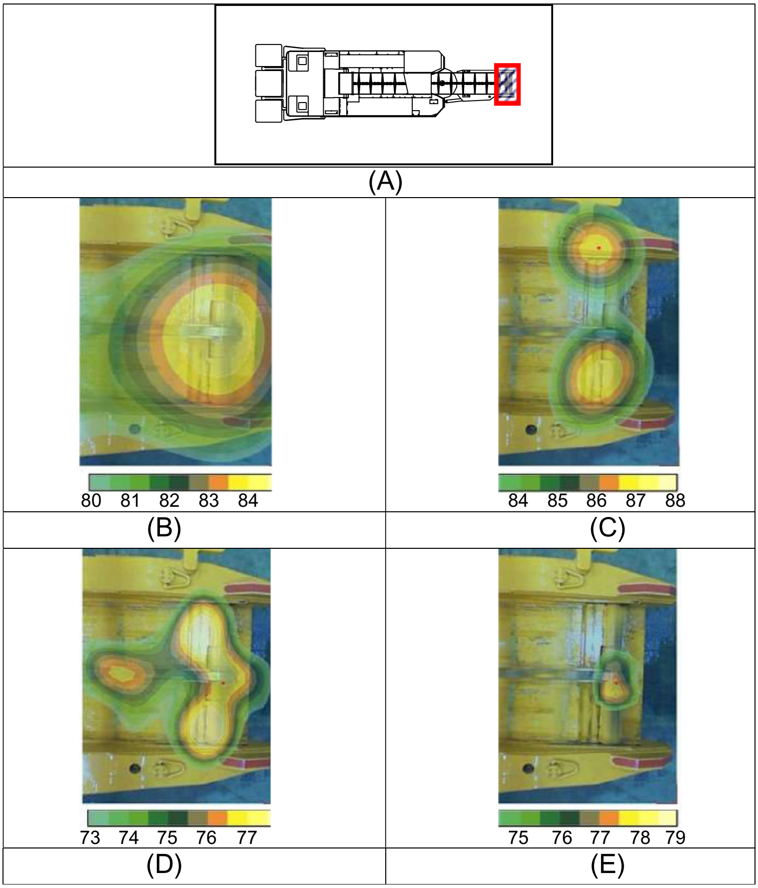

Fig. 12.21 Dominant noise sources at the tail section area (*red box* in (A)) covered by acoustic maps at: (B) 500 Hz, (C) 1000 Hz, (D) 1600 Hz, and (E) 2500 Hz.

Overhead measurements revealed dominant noise sources located at the tail section of the CMM—more specifically in the vicinity of the tail roller. Examination of the acoustic maps suggests three different noise mechanisms: [1] chain-tail roller interaction [2], flight tip-flexplate interaction, and [3] flight-upper deck interaction. Fig. 12.21 shows these sources at four different frequencies. These noise sources were previously suspected but not confirmed. It can be seen that not only do chain links impacting the tail roller (TR) generate noise at the tail section, but that impacts from flight tips on side boards and impacts from chain flights on the upper deck are also significant noise radiators. Given this scenario, an effective noise control should attenuate noise generated in these locations.

Secondary sources were identified at the front end of the left flexplate guide at 1600 Hz and 2000 Hz as shown in Fig. 12.22. Flexplates provide confinement for

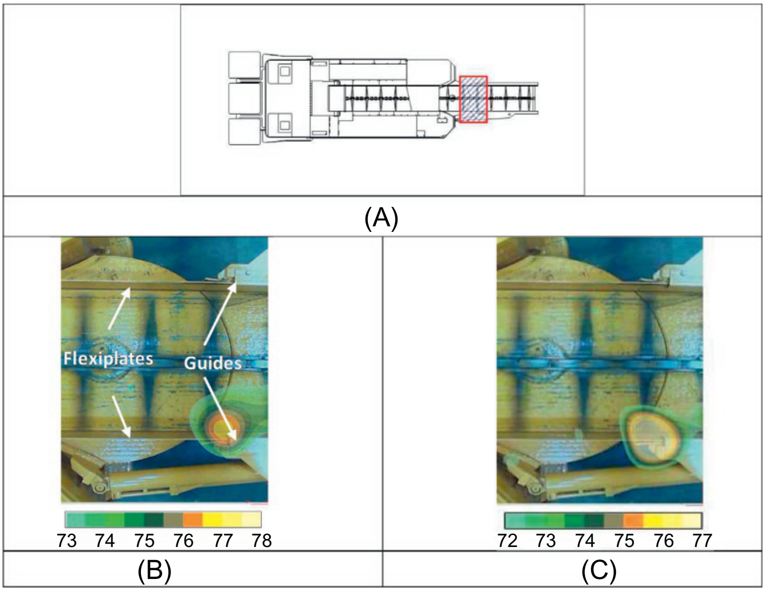

Fig. 12.22 Secondary sources at the flexplate area (*red box* in (A)) covered by acoustic maps at: (B) 1600 Hz and (C) 2000 Hz.

the coal being conveyed as the CMM tail section is swung from side to side during loading of haulage units. It is suspected that chain flight tips impacting flexplates cause them to vibrate and rattle against their guides. Since the clearance between flexplate and flight tips is only approximately 1 cm, a small transverse displacement of the chain causes impacting to shift from side to side. A similar phenomenon was observed in previous studies when the vibration of the left flexplate was greater than that of the right flexplate [32].

Other noise sources were found at the front of the machine; however, since these sources were located farther away from the operator location, they were deemed to be of lesser priority in terms of operator and helper noise exposure reduction [33].

12.4.3.2 Potential noise controls

Conveyor noise has been the subject of previous research [32,34]. From these studies, three noise controls were proposed: [1] a urethane-coated tail roller [2], a jacketed tail roller (i.e., a resilient material between an inner and an outer steel shell), and [3] urethane-coated chain flights. Fig. 12.23 shows prototypes of these noise controls.

Impact forces exerted on the tail shaft by the conveyor chain are transmitted as vibrations through the rest of the structure. Resilient materials have been placed between the chain and roller in an attempt to reduce these forces. Studies conducted

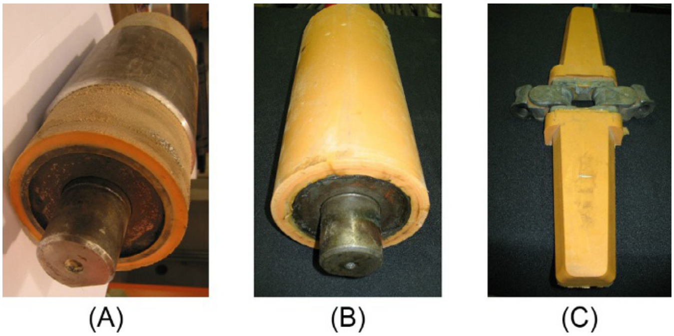

Fig. 12.23 Noise controls for tail section noise: (A) jacketed tail roller, (B) urethane-coated tail roller, and (C) urethane-coated chain flights.

by the Bureau of Mines investigated an isolated tail roller design where an elastomer was bonded to the tail shaft and protected with a steel tube [35,36]. This treatment achieved a 2-dB(A) reduction in sound level at the operator's position, but was never tested underground. Recently, a urethane coating, similar to the coating used for the coated flight bar design, was applied to the outer diameter of the tail roller [37]. In the aforementioned hemi-anechoic chamber, testing with this design resulted in a 2-dB reduction in the operator location sound level. This design failed in underground testing, however, due to high point contact loading from material (e.g., coal or overburden rock) under the chain.

To document the overall noise emission of the CMM during conveying, NIOSH conducted sound power testing in a large reverberation chamber [38]. This testing was conducted using the comparison method of sound power calculation per the requirements of the acoustic standard ISO3743-2 [39]. The same resilient material used for the urethane tail roller tested underground was used to build a prototype for testing in the reverberation chamber. During normal operation of the conveyor (i.e., conveying coal from the cutting end of the CMM to the tail section), the coal has a damping effect on the noise radiated by the conveyor. From these tests, it was determined that overall sound power level of the CMM is 116 dB(A) with a standard tail roller, and is 115 dB(A) with the urethane-jacketed tail roller. Thus, the urethane-jacketed tail roller resulted in a 1-dB reduction.

NIOSH also conducted similar ISO 3743-2 based sound power testing in its reverberation chamber to determine the effect of coating conveyor flight bars with urethane. Here, testing was conducted without loading the conveyor, both for standard tail roller tests and for urethane-coated tail roller tests. The overall sound power level of the CMM with standard chain was 117 dB(A) while the overall sound power level with the urethane-coated flight bars was 112 dB(A). This 5-dB reduction in the sound power emission was considered significant and indicated that the urethane-coated flights bars might be an effective noise control.

12.4.3.3 Constraints

The most important requirement regarding noise controls in this particular application was durability, especially since viscoelastic materials such as urethane were being used. Another constraint, common to all underground mining equipment, is that noise controls should not risk the structural integrity of the component being addressed, which was the conveyor chain in this particular case.

12.4.3.4 Selected noise control

After laboratory and field testing of multiple noise control prototypes, the conveyor chain with urethane-coated flight bars was selected as the most practical noise control. Underground tests showed that this noise control provides an estimated noise reduction of 3 dB in an 8-h time-weighted average using MSHA criteria [34]. Coated flight bars proved to be significantly more durable than the treated tail roller, with in-mine testing suggesting that the life of a coated chain equals or exceeds the expected life of a typical metal-flight chain. This noise control is commercially available and has been implemented in various mines to reduce the noise exposure of CMM operators.

Although the urethane-coated conveyor chain provides significant noise reduction, the overall sound radiated by the CMM must still be further reduced. It is likely that a CMM operator exposed to typical noise levels for an 8-h shift would still be exposed to a time-weighted average sound level exceeding the MSHA PEL. To further reduce the sound level at the operator location, and thus the noise exposure, sound radiated by the dust scrubber system of the machine needs to be addressed. Preliminary work in this area has identified various flow obstructions along the ducting and demister components that result in an off-axis nonuniform flow upstream of the fan. These obstructions create air turbulence thereby decreasing fan efficiencies and increasing noise emissions.

12.5 Conclusions

This chapter has presented an overview of research conducted by NIOSH aimed at reducing occupational noise-induced hearing loss among different mining machine operators. The approach used to achieve this goal was that of reducing sound radiated by various pieces of mining equipment such as longwall mining systems, continuous mining machines, and roof-bolting machines. For each machine, the process involved identifying dominant noise sources and developing engineered noise controls that would attenuate the sound radiated by these sources. Through this process, noise controls were developed and implemented, and in most cases retrofitted, onto machines in operation.

Interestingly, after initial dominant noise sources were attenuated by engineered controls, other noise sources became dominant. For example, drill steel vibration was identified as the dominant noise source for roof-bolting machines. Two noise controls were developed: (1) the drill bit isolator to attenuate vibrations transmitted from bit to drill steel, and (2) the collapsible drill steel enclosure, which acts as a barrier between the drill steel and the operator. However, despite significant noise reductions achieved by these controls (up to 6 dB in sound power level reduction), many roof-bolting machine operators are still overexposed from other sources that have now become dominant. This example suggests the need for continued noise source identification and ranking, followed by development of additional noise attenuation controls.

To eliminate the risk of long-term hearing loss, overall noise exposure for mine equipment operators should be 85 dB(A) (NIOSH recommended exposure limit (REL) for an 8-h shift) or less when possible. Achieving this may require attacking multiple noise sources simultaneously, resulting in a more comprehensive noise control methodology. In this context, the approach for future projects will focus on developing noise controls solutions at the design stage, i.e., quiet-by-design, rather than localized solutions for existing machines.

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Sustainable coal waste disposal practices

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13.1 Introduction and background information

13.1.1 Coal waste defined

Coal is a sedimentary rock that occurs in seams bounded by layers of rock. The generation of waste is unavoidable during coal extraction and beneficiation. Mine waste or "spoils" are materials that are moved from its in situ location during the mining process but are not processed to obtain the final product. Dealing with mine waste is a major part of surface mining methods, where all of the rock above the coal seam (overburden and sometimes interburden) must be removed to expose the coal seam. This is done in a systematic fashion of digging pits with most of the overburden waste being cast from above the coal to be extracted into an adjacent pit from which the coal has already been extracted. Minimizing the handling of overburden waste is one of the keys to economic success in surface coal mining. Various techniques, such as cast blasting, are used to achieve this objective.

If underground mining methods are used, the amount of out-of-seam material handled is much less than in surface mining, and minimizing that amount has multiple economic benefits as discussed in Chapter 11. When large amounts of out-of-seam material have to be removed for underground infrastructure such as ventilation overcasts and undercasts and conveyor belt transfer points, it can be "gobbed" or left underground in untraveled mine openings. However, most of the out-of-seam material extracted in underground mines is mixed with the coal and constitutes part of the run-of-mine (ROM) or "raw coal" product.

Because modern mechanized mining equipment does not distinguish between the coal seam and layers of rock that encapsulate it and because complete or full extraction of a mineable coal seam is generally the objective of any coal-mining operation, there will always be some level of out-of-seam dilution in the ROM product. In most cases, out-of-seam material extracted with the coal must be separated from the coal before shipment to satisfy customer quality requirements. This is accomplished with coal preparation plants that generate a "clean coal" product and a waste material referred to as "coal refuse." Coal preparation plants utilize various mineral processing technologies that, with few exceptions, are slurry-based and involve the use of substantial quantities of water [1]. The efficiency of these processing systems depends on the size of material being treated. Hence, raw coal must be classified into different

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size fractions leading to two coal refuse products on the output side: (1) coarse coal processing waste (CCPW) and (2) fine coal processing waste (FCPW). Generally, CCPW is material larger than 150 μm (100 mesh) in size [2]. CCPW includes reject streams from jigs, heavy media vessels, and heavy media cyclones. FCPW includes reject streams from spirals, flotation columns and cells, desliming cyclones, and effluent streams of filter presses, screenbowl centrifuges, and other dewatering equipment. All FCPW streams are typically concentrated in a thickener whose output is a waste slurry.

Most extraction and beneficiation wastes from coal mining (i.e., mine spoils and coal refuse) are categorized as "special wastes" that are exempted from regulation by hazardous waste rules and laws (e.g., Subtitle C of the US Resource Conservation and Recovery Act). However, coal utilization generates another type of waste known as coal combustion residuals (CCRs), which are regulated to some degree (e.g., Subtitle D of the US Resource Conservation and Recovery Act). CCRs are categorized into four groups based on physical and/or chemical forms that derive from the combustion method and the emission control system used. A brief description of each group follows [3]:

- Fly ash is a very fine, powdery material composed mostly of siliceous mineral matter left over from the burning of finely ground coal in a boiler. It comprises 60% of all CCRs.
- Bottom ash is a coarse, angular, gritty material with similar chemical composition to fly ash. It is too large to be carried up by the smokestack, so it collects in the bottom of the coal furnace. It comprises 12% of all CCRs.
- Boiler slag is molten bottom ash that forms into pellets in the bottom of slag tap and cyclone type furnaces. It has a smooth glassy appearance after it is cooled with water. Boiler slag comprises 4% of all CCRs.
- Flue gas desulfurization (FGD) material is residue from the sulfur dioxide emission scrubbing process. It can be a wet sludge consisting of calcium sulfite or calcium sulfate, or it can be a dry powdery material that is a mixture of sulfites and sulfates. FGD material comprises 24% of all CCRs.

13.1.2 Sustainability defined

Because no other industry has such a direct impact on the environment in which it operates, no other industry is as focused on maintaining its environment than the mining industry. That focus may not have always been there, but as society increasingly recognizes the need for balance between life and lifestyle, the mining community has responded by becoming much better stewards of the land where it conducts business.

Although mining is one of the world's oldest industries, the concept of environmental stewardship as it relates to mining is relatively young [4]. For example, in the United States, the Environmental Protection Agency (EPA) was not formed until 1970. In 1977, the US Congress passed the Surface Mining Control and Reclamation Act, which remains the primary federal law regulating environmental impacts of coal mining. In 1983, the United Nations appointed the World Commission on Environment and Development to unite its member countries in an effort to pursue sustainable development, which it defined as "the advance of human prosperity in a way that does not compromise the potential prosperity and quality of life of future generations" [5]. In other words, coal mining in all of its various facets, including waste disposal, is sustainable if it meets the needs of the present without compromising the ability of future generations to meet their own needs.

Mining epitomizes the challenge of sustainability because, as previously stated, no other industry has such a direct impact on the natural environment yet provides so many essentials to modern society and the global economy. Existing and future mining operations will face some or all of the following challenges [6] when it comes to sustainability:

- Dwindling ore reserves that force operations to move closer to areas that are environmentally or culturally sensitive or are more densely populated.
- Population growth that causes urban areas to move closer to existing mining operations.
- Discovery of new reserves in remote locations that range from pristine wilderness areas to undeveloped or underdeveloped countries.
- Increased environmental awareness among the general public, elected officials, and the media leading to heightened levels of scrutiny on mining operations.

The aim of fostering sustainability in coal mining is to ensure that coal, and more broadly, energy utilization is minimized without having a negative impact on economic growth and standard of living. "Reduce" is one of the three "R" principles of sustainability. The other two are "reuse" and "recycle."

Recycling coal is a hard concept to grasp because most coal is burned to generate heat that is converted to electricity; however, recycling is very applicable when considering the mining industry as a whole [7]. Many of the products produced by mines can be recycled, which reduces the demand for the mine's product. Successful mining companies understand that recycling not only extends the life of the mine but also can create new markets. Recycling requires specific types of equipment that may be different from those required to produce traditional products, but they are made from mining products nonetheless. Successful mining companies work with their customers to improve the efficiency and performance of each and every product.

Reuse is particularly important for CCR waste. CCRs can be beneficially used as a replacement for raw material removed from the earth to manufacture many different products. This not only does conserve natural resources but also reduces CCR disposal costs. CCRs have been found to improve the strength and durability of products over what they are when made with virgin raw material. In addition, reuse of coal ash encapsulates potentially hazardous material preventing it from causing negative environmental impacts if simply disposed of [8].

13.1.3 Current waste disposal practices

13.1.3.1 Overburden waste

Despite the fact that the amount of material moved by surface mining methods is enormous compared with underground mining methods, surface mining achieves a lower cost per ton based on economies of scale achievable with the method and equipment used. Thus, when coal seams are readily accessible from the surface due to shallow depth, surface mining is the preferred option for coal production. Strip or open-cast mining is the most common method for extracting these coal seams. With that method, overburden waste is cleared to expose the underlying coal and placed in an adjacent excavation where the coal has already been excavated.

The overall economic goal in surface mining is to move the least amount of overburden waste necessary to mine the greatest amount of marketable product. Two primary economic parameters are the mining rate, which is used to determine how long the coal reserve will last (i.e., mine life), and the stripping ratio, which is used to define economic limits for surface mining when the coal seam is dipping or the surface topography varies such that overburden depth is increasing.

Overburden waste consists of unconsolidated material (soil) at the surface and consolidated material (rock) overlying the coal seam. Removal is a cyclical process of drilling, blasting, loading, and hauling. Generally, unconsolidated material is easily removed without the need for drilling and blasting. Loading and hauling of unconsolidated material are performed by scrapers, dozers, backhoes, and dump trucks at most operations with bucket wheel excavators used at large mines. Loading and hauling of consolidated material are performed by dozers, front-end loaders, and off-highway trucks at smaller mines and by drag lines and/or truck-and-shovel combinations at larger operations.

Before current reclamation laws were enacted, overburden waste was moved in the most economical fashion and left in "spoil" piles or rows without any grading or separate recovery of top soil to use in restoring land to its premining productiveness. However, since such laws have been enacted, overburden waste must now be carefully managed with grading to original contour and restoration of stratigraphic columns with a top layer of topsoil. In addition to being managed in ongoing reclamation programs, removed overburden can be used for building mine haul roads and constructing stable coal refuse structures. Some overburden waste may have nonmining commercial value and be sold as fill dirt.

13.1.3.2 Coal refuse

As previously stated, full extraction results in out-of-seam material in the ROM product, which usually must be removed to satisfy customer specifications. This requires a complex arrangement of coal storage, coal processing, and waste handling facilities (see Fig. 13.1), which are located on the surface in close proximity to the actual mine. These facilities typically include a closed-loop water management system where on-site water resources are continuously recycled and off-site water discharges are carefully regulated. Such discharges typically occur from sedimentation basins, but only during larger precipitation events when less stringent regulations are applied and/or there is a considerable amount of dilution water available. Coal refuse includes all waste generated from the overall facility but primarily by the cleaning process used in the coal preparation (wash) plant.

The conventional approach to coal refuse disposal is to construct stable embankments with CCPW that are used to contain FCPW as concentrated slurry of ±15% solids or as thickened paste of 50%–70% solids. With increased scrutiny being given

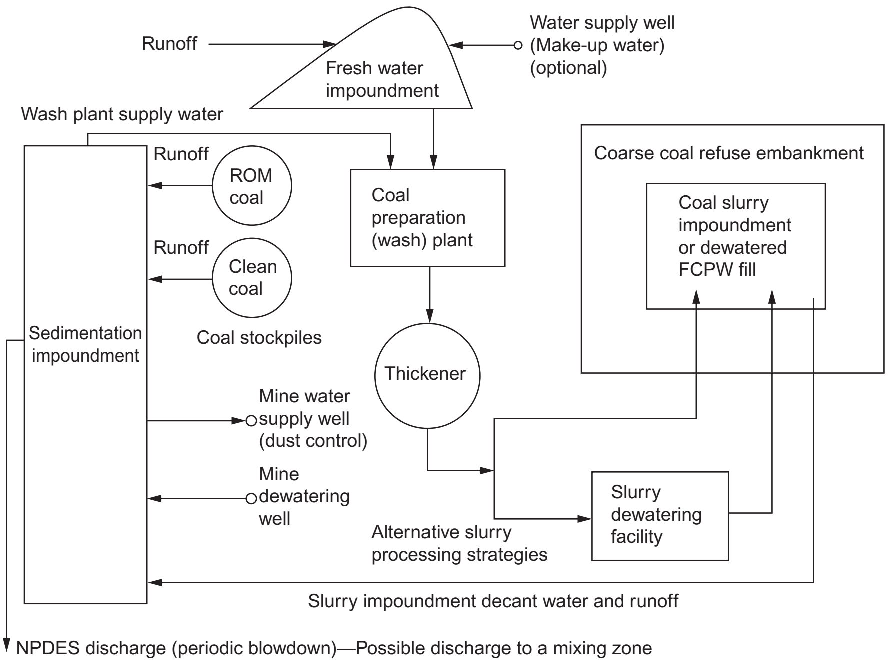

Fig. 13.1 Schematic of typical coal mine surface support facilities [9].

to the process of obtaining permits for constructing slurry impoundments, many mines have turned to the disposing of FCPW in older abandoned mine workings such as underground shafts and tunnels or the final cut of a surface mine. This has led to stability problems (e.g., reduced bearing capacity of clay strata below the coal seam) in underground mines and groundwater contamination problems at both surface and underground mines. Both problems arise from high water content in the FCPW. Addressing that issue has been the focus of research on constructing monofill refuse piles from a blend of CCPW and dewatered FCPW, which will be covered later in this chapter.

The combined trends of mining thinner coal seams with larger, stronger, fully mechanized equipment have increased the percentage of coal refuse to as much as 50% of the ROM product. These trends, coupled with more stringent regulations to control air, water, and land pollution, have complicated the already challenging problem of disposing of coal refuse in a safe and environmentally acceptable manner. As a result, backfilling, long considered cost prohibitive for coal mines, is becoming more economically attractive [10]. Testing has concluded that if improved ground control is the only reason for backfilling, coal refuse alone does not appear to be a suitable backfilling material; however, if rising coal refuse disposal costs can be abated, then it becomes much more attractive [11]. The authors believe that the greatest potential advantage offered by backfilling is increased resource recovery, but this concept still needs to be demonstrated in the field. This will also be discussed later in this chapter.

13.1.3.3 Coal combustion residuals

First-generation coal-fired power plants spewed coal ash and other hazardous pollutants into the atmosphere until environmental regulations began requiring emission control systems. Bag houses, scrubbers, electrostatic precipitators, and other technology were added on at tremendous cost and electricity generating companies then had to deal with where to store CCRs that were now being captured. Storage in lagoons, landfills, or silos has long been the norm for coal ash management in most countries, but that is changing around the world as storage space becomes limited and as environmentally acceptable and competently engineered means of reusing CCRs are proven. For example, according to the European Coal Combustion Products Association, 13.8 of the 48 million tons of coal ash produced in 15 EU countries in 2010 were reused. Countries like the Netherlands and Germany that no longer allow landfill storage saw 100% and 97% reuse, respectively. The UK Quality Ash Association and the American Coal Ash Association (ACAA) reported 50% and 43% reuse, respectively, in 2013 [12]. In China, the recycling rate is about 30% despite a 2005 government challenge to achieve 75% [13]. In the United States, of the 53 million tons of coal ash generated in 2013, 23 million tons were reused with the remainder stored in landfills (36%) and wet storage facilities (21%) [14].

The chief benefit of reusing CCRs is to stabilize or encapsulate components that may be environmentally harmful such as arsenic, lead, mercury, selenium, and dioxins. Common reuses for the previously described CCR groups are as follows [14]:

- Fly ash is most commonly used as a high-performance substitute for Portland cement and in blended cements such as asphaltic concrete. Fly ash also serves as a filler in wood and plastic products, paints, and metal castings. Building material applications range from grouts and masonry products to cellular concrete and roofing tiles. Geotechnical applications include soil stabilization and structural fill for road base and embankments.
- Bottom ash and boiler slag is most commonly used for skid control on icy roads. These materials are also suitable for geotechnical applications such as structural fills and land reclamation. The physical characteristics of bottom ash and boiler slag lend themselves as replacements for aggregate in flowable fill and in concrete masonry products. Boiler slag is also used for roofing granules and as blasting grit.
- FGD material is most commonly used as synthetic gypsum in wallboard and as spray dryer absorbent. FGD gypsum is used in almost 30% of gypsum panel products manufactured in the United States. It is also used in agricultural applications to treat undesirable soil conditions and to improve crop performance.

Despite trends toward reuse, storage remains the primary waste disposal avenue for CCRs largely because it is generally both the easiest and the cheapest option, especially when there is an available disposal site near a power plant. According to the ACAA, if coal ash can be piped to a nearby storage site rather than trucked, costs are \$3–5 per ton; however, when the disposal site is further away and a more complex transport solution is needed due to either higher moisture content or larger volume, the cost could rise to \$20–40 per ton [14].

13.2 Sustainable coal waste disposal practices of the future

13.2.1 Eliminating slurry impoundments with codisposal

Following a 2000 incident in Martin County, Kentucky, the United States, in which an impoundment containing >300 million gallons of fine coal processing waste failed allowing slurry to flow through an underground mine into surrounding creeks and rivers, the US Mine Safety and Health Administration (MSHA) and the Office of Surface Mining recommended that research be conducted to identify and evaluate alternative methods of coal waste disposal [15]. The need for alternative methods is due to multiple hazards associated with conventional coal refuse disposal facilities. Such facilities have a history of geotechnical failures, although the risk for this has diminished significantly with the development of regulatory performance standards (e.g., the US Surface Mine Control and Reclamation Act passed in 1977). Additionally, these facilities have been identified as principal sources of elevated sulfate and chloride discharges resulting from pyrite weathering due to infiltration of oxygen- and ferric iron-bearing water.

In response to the above recommendation and additional concerns regarding efforts to regulate coal mine discharges at the same level as the general use of water quality standard, the authors became involved in a series of studies to develop good management practices restricting pyrite oxidation thereby reducing sulfate levels in coal mine discharges to levels that comply with present and future regulatory standards. The studies culminated in long-term, field-scale column leaching experiments showing that codisposal of coarse and fine coal refuse provides both the geotechnical stability needed to lower refuse facility liabilities and the geochemical environment necessary to minimize sulfate and chloride discharges. The primary objectives of these studies were to reduce water treatment costs during mining and prevent long-term problematic discharges that could hinder reclamation and bond release.

13.2.1.1 Experimental set up

FCPW and CCPW samples were collected from a large coal preparation plant cleaning ROM coal from two underground mines and one surface mine all operating in commonly mined seams in the Illinois Basin. A complete physical and chemical characterization of these samples and some limestone material was performed including particle size distribution, moisture content, acid-base accounting (pH), total and pyritic sulfur content, key trace element concentrations in a high-temperature ash (HTA) product, and X-ray fluorescence (XRF). Potential acidity (PA), neutralization potential (NP), and net neutralization potential (NNP) were calculated.

After this characterization, samples were mixed according to three waste disposal options to be considered: disposal practice (DP) 1 consisting of 100% CCPW (control), DP 2 consisting of codisposal of blended CCPW (90%) and FCPW (10%), and DP 3 consisting of codisposal of blended CCPW (84%) and FCPW (8%) with limestone addition (8%). The geotechnical properties of all three mixtures were characterized including moisture-density relationships developed using standard Proctor tests (ASTM D698-12e1 Method C) [16].

Finally, field-scale kinetic tests were conducted outdoors at the preparation plant complex. This ended up being done in two phases due to a tornado damaging some of the initial columns. During the first phase, three duplicate test column sets (FC-1–FC-6) were constructed using 200 L (55 gal) barrels and monitored for 19 months. In the second phase, three more duplicate test column sets (FC-7–FC-12) were constructed with 380 L (100 gal) polycarbonate livestock watering troughs (see Fig. 13.2). Phase 2 column mixes were 100% CCPW for DP 1; 93.3% CCPW and 6.7% dewatered FCPW by volume for DP 2; and 86.7% CCPW, 6.7% dewatered FCPW, and 6.7% ground limestone by volume for DP 3. Assuming normal porosity of 16%, approximately 300 kg (650 lb) of CCPW were packed into each column. Sand cone tests were performed during construction to determine dry unit weight and moisture content. Phase 2 columns were leached in free-draining mode for ~19 months (16 leach cycles) with pore-water and leachate samples collected once per leach cycle (~35 days). Sample temperature, pH, and dissolved oxygen (DO) were measured in the field. As soon as samples arrived in the laboratory, temperature, pH, and specific conductance (SC) were measured. Alkalinity and ferrous iron were determined within 24 hours after sample collection. All water sample quality results were statistically analyzed using spreadsheet-based models.

13.2.1.2 Results and analysis

Table 13.1 shows particle size distributions for CCPW and limestone samples. FCPW was uniformly <0.074 mm (0.0029 in). Initial moisture content of CCPW was 11.5%, dewatered moisture content of FCPW was 31.9%, and initial moisture content of agricultural limestone was 0.1%. Therefore, blending limestone with CCPW and FCPW in the DP 3 mixture significantly reduced its overall moisture content. Table 13.2 shows

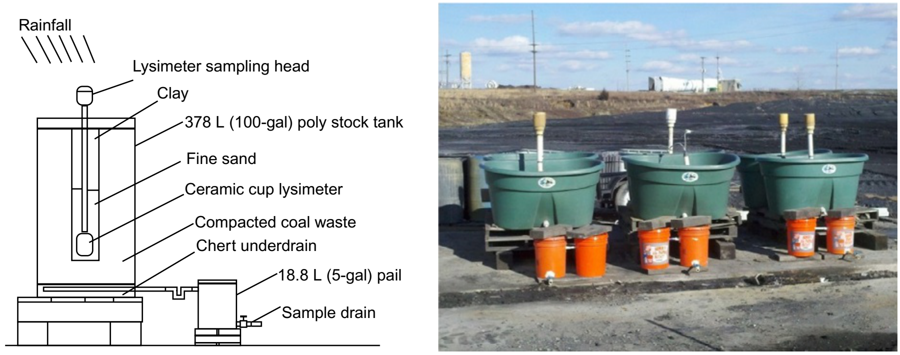

Fig. 13.2 Schematic and photograph of field-scale kinetic test columns [2].

Table 13.1 Particle size distribution for CCPW and ground limestone [2]

Sieve opening (mm)	Percent passing	
	CCPW	Ground limestone
50.8	94.31	100
19.05	62.20	100
4.75	29.79	78.09
1.7	15.09	27.03
0.425	4.44	13.17
0.075	0.53	4.58
<0.075	0.52	3.58

Table 13.2 Particle size distribution for Proctor test materials [2]

Sieve opening (mm)	Percent retained		
	DP 1	DP 2	DP 3
4.76	63	58	54
1.68	20	18	20
0.42	11	10	11
0.75	5	4	4
<0.075	1	10	11

particle size distributions for Proctor test samples. These are based on three tests for DP 1 materials, two tests for DP 2 materials, and two tests for DP 3 materials.

Table 13.3 shows acid-base accounting results. Data from one of the mine permit applications are included for comparison. These data corroborate conclusions reached in earlier studies [17] by one of the authors that FCPW typically represents 10%–15% of total coal refuse and that FCPW sulfur content is generally lower than CCPW sulfur content. Nothing of significance appeared when analyzing concentrations of main elements and key trace element.

Proctor test results for all disposal options are summarized in Fig. 13.3. DP 1 moisture content varied from 2% to 17% with maximum dry unit weight of 18.5 kN/m^3 (118 lb/ft^3) at 6.0% moisture; DP 2 moisture content varied from 2% to 9.5% with maximum dry unit weight of 18.0 kN/m^3 (115 lb/ft^3) at 5.5% moisture; DP 3 moisture content varied from 2.5% to 11.5% with maximum dry unit weight of 18.52 kN/m^3 (118 lb/ft^3) at 7.5% moisture. The addition of ground limestone advantageously increased moisture content at which maximum density was achieved by about 2%.

Table 13.4 shows median values for pH, oxidation reduction potential (ORP), alkalinity and acidity, and concentrations of major anions and cations as well as total

Table 13.3 Geochemical properties for CCPW, FCPW, and ground limestone samples [2]

Coal waste fraction	Mean sulfur content (%)		Median paste pH	Tons $CaCO_3$ per 1000 tons of waste		Net neutralization potential (NNP)
	Total	Pyritic		Potential acidity (PA)	Neutralization potential (NP)	
Mine permit (coarse)	5.70 (n[a] = 2)	3.41 (n = 47)	7.12 (n = 47)	106.4 (n = 47)	23.8 (n = 47)	−84.5 (n = 47)
CCPW	4.84 (n = 2)	3.90 (n = 1)	5.92 (n = 1)	121.88 (n = 1)	1.51 (n = 1)	−120.37 (n = 1)
FCPW	2.56 (n = 2)	2.13 (n = 1)	6.31 (n = 1)	66.53 (n = 1)	2.65 (n = 1)	−63.91 (n = 1)
Limestone	NT[b]	0.17 (n = 1)	8.35 (n = 1)	5.31 (n = 1)	58.17 (n = 1)	52.86 (n = 1)

[a] n, number of samples.
[b] NT, not tested.

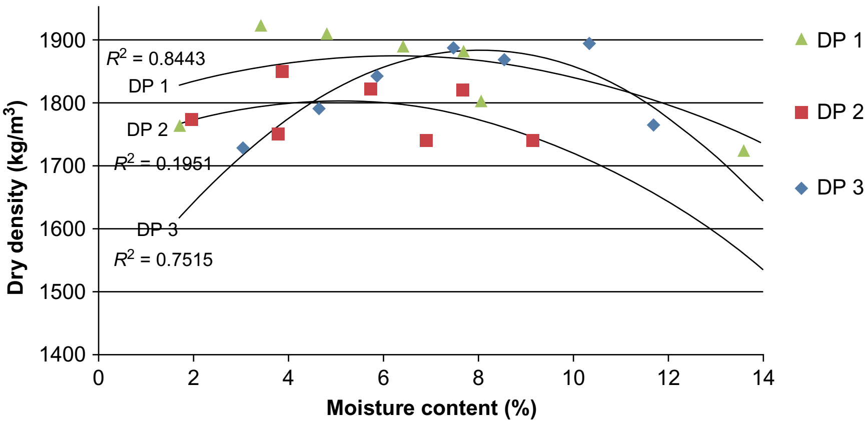

Fig. 13.3 Composite Proctor test results [2].

dissolved solids (TDS) of leachate collected from all columns. Values are reported separately for the initial leaching period (≤7 months) and the latter leaching period (>7 months). Regarding pH, during the initial leaching period, pH values were well above the carbonate buffer pH level of 6.37; however, they declined sharply after that. Acidity is seen to increase sharply from the initial period to the latter period for all three mixes; however, the increase is much less dramatic for DP 3. As would be expected, alkalinity exhibits the opposite effect, and again, it is much less dramatic for DP 3. Regarding the major anions monitored, HCO_3^- and Cl^- were most easily leached. HCO_3^- (not shown in Table 13.4) is desired because it provides alkalinity needed to buffer pH, but concentrations of it declined after 7 months of sampling [9]. Similarly, concentrations of Cl^-, a highly mobile anion, declined dramatically after the initial leaching period suggesting that Cl^- discharges are minimally affected by alternate disposal practices. Conversely, SO_4^{2-} concentrations were high from the start and rose significantly after the initial leaching period; however, concentrations in blended columns were considerably lower, and the percentage increase was significantly less for DP 3. Similar results were obtained for several elemental cations (Ca^{2+}, Mg^{2+}, Na^+, K^+, Ni^{2+}, Zn^{2+}, Cd^{2+}, Sr^{2+}, and Pb^{2+} not shown). Metals commonly associated with pyrite weathering (Fe, Mn, Al, Ni, Zn, Cu, and Cd) increased in the latter leaching period but were again lower for DP 2 and substantially lower for DP 3.

Table 13.5 shows similar results for pore-water samples collected from all columns. Overall, the volume of pore water collected was much lower than the leachate volume with the average pore-water sample collected being only 30.8 mL. After 7 months of kinetic testing, all but two lysimeter ceramic sample cups ceased producing pore water because of Fe-rich precipitates, and those two ceased producing after 9 months. There was limited analysis of pore-water chemistry during the latter leaching period; however, even with limited sampling, it was observed that pore water exhibited the same effects as leachate.

Table 13.4 **Average concentration[a] of selected mine drainage parameters in column leachate [2]**

Refuse type	Interval	pH	ORP	SO_4	Cl	TDS	Alkalinity	Acidity	Fe	Mn	Al
DP 1	≤7 months	8.02	0.132	3437	198.0	3865	236.6	9.0	0.76	0.89	1.22
	>7 months	2.50	0.769	5449	10.8	10,445	0.0	4909	1258	78.8	140.6
DP 2	≤7 months	8.32	0.077	2490	197.2	2968	266.2	2.7	0.23	0.62	0.66
	>7 months	3.56	0.621	4502	12.7	5253	18.2	1008	204.6	23.1	44.00
DP 3	≤7 months	7.83	0.133	3093	126.2	3698	203.7	1.5	0.08	0.66	0.01
	>7 months	5.82	0.454	3456	11.4	2549	35.9	100.2	12.43	7.10	5.14

[a]Data in mg/L except pH (median value), ORP (volts), and alkalinity and acidity (mg/L CCE).

Table 13.5 **Average concentration[a] of selected mine drainage parameters in column pore water [2]**

Refuse type	pH	SO_4^{2-}	Cl^-	F^-	Acidity	Fe	Mn	Al	Ni
DP 1	7.65	2844	849.5	4.92	1.07	0.787	0.054	0.025	0.212
DP 2	7.66	3022	356.4	4.59	1.69	0.516	1.952	<0.001	0.378
DP 3	7.76	1995	515.6	4.41	3.02	1.555	1.182	<0.001	0.260

[a]Data in mg/L except pH (median value) and acidity (mg/L CCE).

The decline in HCO_3^- concentrations was much greater than for SO_4^{2-} concentrations in both codisposal columns suggesting that carbonate weathering rates are faster than pyrite weathering rates in the coal refuse environment or that alkalinity-producing minerals are being coated with mineral precipitates that are limiting dissolution. SO_4^{2-} mobility was significantly lower in codisposal columns, especially with limestone addition. The higher extraction of S compared with Ca suggests that the formation of calcium SO_4^{2-} such as gypsum or anhydrite was relatively small. Elements associated with alkalinity-producing minerals (e.g., calcite and dolomite) such as Ca, Mg, and Sr were leached to a greater extent than heavy metals typically associated with pyrite such as Mn, N, and Zn and lithophile elements such as Al and K.

Data for elemental constituents were converted to a mass (loading) basis by multiplying concentration values and leachate volume allowing the determination of cumulative elemental extraction. Mass data were plotted as a function of time, with time represented by leach cycles. The complete kinetic testing program consisted of 16 leach cycles comprising 568 days with an average of 19,813 mL of leachate collected from all of the columns for each cycle. Leachate volume was then compared with estimated pore volume of 54,501 mL to yield an initial liquid-to-solid (L/S) ratio of 0.19. As a result, the average rate of pore volume flushing is approximately 0.36 volumes per leach cycle with 5.82 pore volumes leached over the course of the study. Cumulative extraction versus time for S and Cl are shown in Figs. 13.4 and 12.5, respectively. Although the Cl extraction percentage was greater than the S extraction percentage during the initial leaching period, over the entire 19-month test period, the amount of S extracted was higher, especially for DP 1 columns.

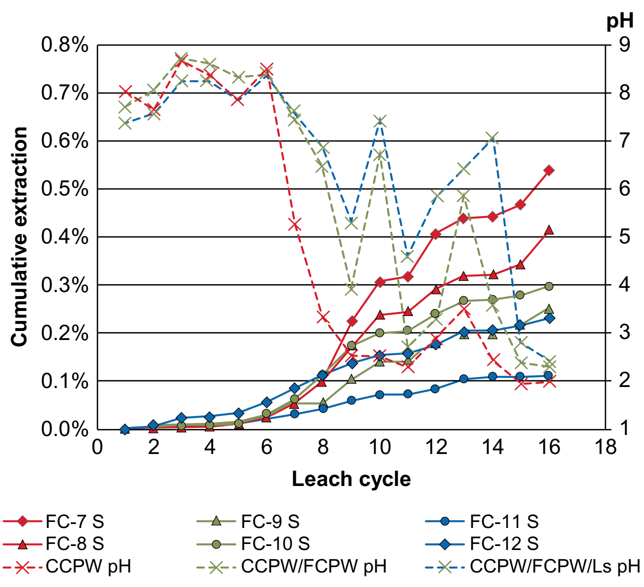

Fig. 13.4 Cumulative S extraction and measured pH with time for the complete test period [2].

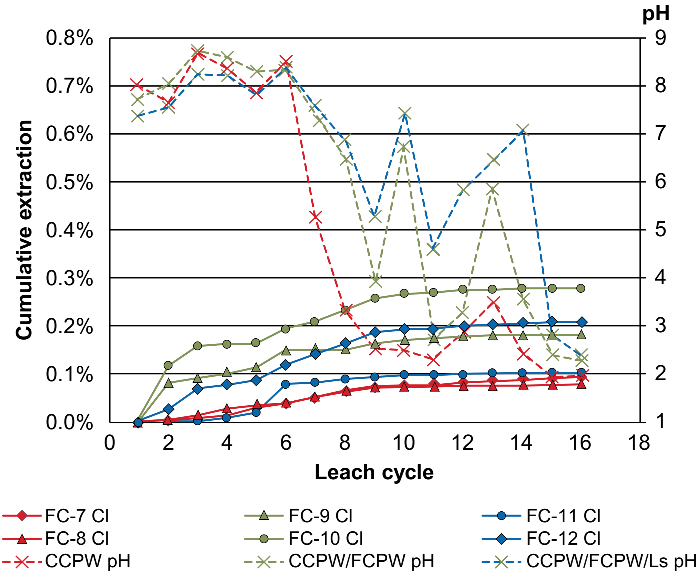

Fig. 13.5 Cumulative Cl extraction and measured pH with time for the complete test period [2].

S and Cl are the major anions, and Na and K are the major cations in high total dissolved solid (TDS) discharges. Fig. 13.6 shows the percentage extraction of these and a few other selected elements during the 19-month kinetic test. Mobility of Ca, Mg, and Sr elements commonly associated with calcareous elements was relatively low throughout the test period compared with Mn, Ni, and Zn trace elements that are commonly associated with pyrite. Mn, Ni, and Zn also had higher extraction rates than Fe suggesting considerable precipitation of Fe phases within the test columns. Fe extraction from DP 1 columns and to a much later and limited extent from DP 2 and DP 3 columns increased as pH dropped.

After collecting and analyzing all of the data, geochemical modeling was performed to predict reaction pathways and evaluate reaction kinetics. Models were used to better understand geochemical conditions such as pore water and leachate composition in column materials. All of the test data and modeling indicate that CCPW and FCPW codisposal practices with or without limestone addition are a significant improvement over the current practice of disposing CCPW by itself. Codisposal had a minimal effect on Cl^- release as most of the Cl^- discharge occur soon after placement and must be dealt with using water management practices of retention ponds, dilution, and discharge during major precipitation events.

13.2.2 Dewatering fine coal processing waste

Codisposal will require robust technologies for dewatering coal refuse. Larger coal and refuse particles are easily dewatered using screens and basket centrifuges, two common types of mineral processing equipment that are used in many applications;

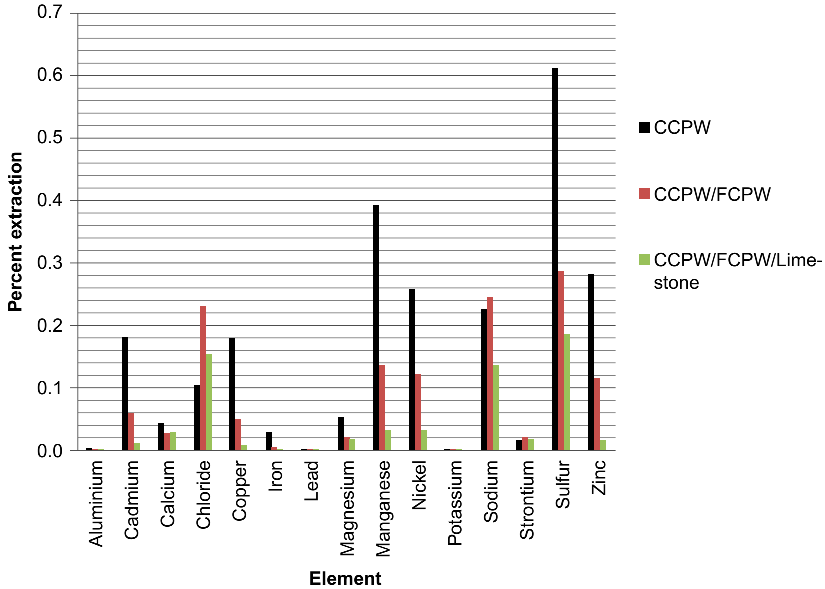

Fig. 13.6 Elemental extraction throughout entire 19-month kinetic test period [2].

however, developing similar technologies for fine coal and refuse particles has been a "holy grail" topic in the mineral processing industry for decades. Thanks to the relentless pursuit of that quest, several options are available. They can be broadly classified as thermal drying, sedimentation, and filtration [1]. Thermal drying is the application of heat to evaporate water. Although very effective, it is rarely used due to the high cost of supplying the thermal energy involved and the safety hazards associated with heating carbonaceous material. Sedimentation is the rapid settling of solid particles in a liquid that produces a clarified liquid and a thickened slurry. It is most efficient when there is a large density difference between liquid and solid; for example, water (SG = 1.00) and pyrite (SG = 4.95). Particle settling in sedimentation may be aided by centrifugation or the use of chemicals that cause particles to adhere to each other. Filtration uses a porous medium to retain solid particles while allowing liquid to pass through. Filtration is often aided by mechanical pressure or vacuum suction.

Of the many options now available for dewatering FCPW, the authors have chosen to highlight two. One, the deep cone paste thickener, is a commercially available advanced sedimentation process; the other, osmotic dehydration, is an advanced filtration process that is still being developed.

13.2.2.1 Paste thickening

As coal mining has become more mechanized, the percentage of fine material in ROM product has increased leading to various technologies being developed for separating the fine fraction into product and waste streams. With few exceptions, these technologies use massive quantities of water requiring additional dewatering technology, without which efforts to recover the fine coal fraction are essentially wasted [18]. Dewatering coal is a simpler problem than dewatering FCPW due to the angular characteristics of fine coal particles. Furthermore, dewatered coal generates revenue while dewatering FCPW only adds to costs. Hence, FCPW is often looked at as the red-headed stepchild of coal preparation.

As previously described, in a typical coal preparation plant, FCPW consists of reject streams from spirals, flotation columns or cells, desliming cyclones, and effluent streams from filter presses, screenbowl centrifuges, and other dewatering equipment, all of which report to a thickener where they are concentrated into a waste slurry. While concentrated, the underflow slurry from a conventional thickener is still too low in percent solids to be able to handle it like CCPW. Addressing this issue, conventional thickener technology has been modified to create the paste thickener, which has proved effective at increasing thickener underflow solid content to ~50% solids. At this concentration, FCPW material has the consistency of paste with rheological properties that allow surface stacking.

In one paste thickening study [19], thickener underflow slurry from a central Appalachia coal preparation plant was tested in a laboratory-scale T-Floc apparatus to optimize flocculant dosages for obtaining maximum settling flux and underflow solid concentration. A maximum solid concentration of 35% by weight was achieved. Pilot-scale tests were then conducted using a Dorr-Oliver Eimco Deepcone thickener, which concentrated the same thickener underflow slurry solids from 10% to 50% by weight. Thickened paste had a yield stress of about 165 Pa, which is sufficiently low to allow transport to a disposal area using a conventional positive displacement pump. Clarity of the paste thickener overflow stream was similar to that currently achieved with a conventional thickener.

In a comparison of thickener and centrifuge technologies [20], it was shown that the most obvious benefit of paste thickening technology is a reduction in surface impoundment area. Other benefits include increased recovery and recycling of plant process water, less potential for groundwater contamination, and easier site reclamation. When the product from a paste thickener is mixed with appropriate chemical agents, it can be pumped back into the mine as backfill, an opportunity that will be examined later in this chapter. Some disadvantages of paste thickening are that it does not replace the existing thickener, but requires installation of a second thickener and the need for additional chemicals. Furthermore, transporting the paste over any distance beyond what can be achieved with free gravity flow requires expensive positive displacement pumps and wear resistant piping, which increase capital and operating expenses significantly.

13.2.2.2 Osmotic dehydration [21]

Virtually all filtration-type devices used in FCPW dewatering can be considered active approaches in that they force water through the porous filter medium. However, there is a passive technology used in municipal water systems that has been investigated for its applicability to dewatering FCPW. This technology utilizes the

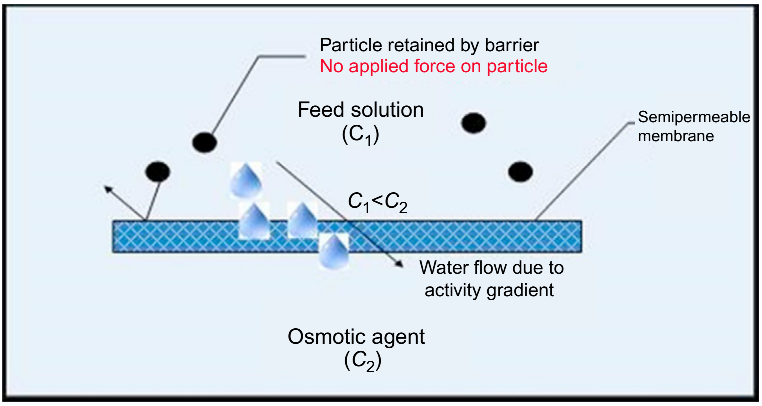

Fig. 13.7 Principle of osmotic dewatering technology [21].

differences in *activity* of an osmotic agent and FCPW slurry to instigate self-directed transport of water from the coal slurry to the osmotic agent. It is, in essence, a passive dewatering technology. The principle of the technology is illustrated in Fig. 13.7, which shows a feed solution having a relatively low concentration (c_1) of ionic or nonionic solute separated from the osmotic agent or draw solution containing an elevated concentration (c_2) of the ionic or nonionic solute by a semipermeable membrane barrier. Since the *activity* of water in the feed solution is higher than that in the osmotic agent, water flows from the feed solution to the osmotic agent effectively dewatering the feed solution.

This technology has two key components—the semipermeable membrane and the osmotic agent. Development of highly selective, high-flux, semipermeable cellulosic membranes in the 1960s led to the development of forward osmosis or extracting potable water from seawater and reverse osmosis (RO) filtration for household water systems. The primary purpose of the semipermeable membrane is to separate the material being dewatered from the osmotic agent. This allows selective flow of water without the mixing of the two solutions.

The choice of osmotic agent is equally important as it determines regenerability, which is a critical characteristic if the process is to be efficient. Requirements of a good osmotic agent include the ability to lower water activity at low concentrations; thus, low-molecular-weight solutes are more efficient than high-molecular-weight solutes. Common osmotic agents in use are sodium chloride (readily available at mines with high-chloride discharges), magnesium sulfate, glycerol, and sucrose. The osmotic agent should be inexpensive, easily removed from solution to enable recycling, and preferably noncorrosive, nonflammable, and nontoxic.

Operational parameters of osmotic dehydration include the concentration of the osmotic agent, which depends upon the salinity of the coal slurry. The higher the salinity of the feed solution, the higher the concentration of the osmotic agent; however, the maximum concentration of the osmotic agent solution is limited by its solubility. The minimum concentration is fixed by the desired extent of feed solution dewatering.

For example, if water activity is 0.9 at the targeted level of dewatering for the coal slurry, the corresponding osmotic agent solution activity must be ≤ 0.9. Another factor that influences osmotic agent concentration is the ability to be removed from solution using energy efficient methods such as RO. This limits osmotic agent concentrations to a maximum of 6%–7% sodium chloride or equivalent; however, using other techniques such as membrane distillation allows relaxation of these limits. The other important parameter is membrane area. Since the amount of membrane used directly affects costs, it is desirable to use the minimum possible membrane area. This requires a maximum activity difference between feed solution and draw solution containing the osmotic agent.

Osmotic dehydration offers two distinct advantages over conventional pressurized membranes or other filtration processes. First, there are no externally applied driving forces on slurry particles, only gravitational body forces and drag forces. This helps ensure that particles are not forced toward the filter barrier, which lowers the potential for the plugging of the porous media. Second, while osmotic forces create pressure-type forces equivalent to several megapascals (hundreds of pounds per square inch), the equipment itself is unpressurized leading to simple processes and ease of equipment construction and operation. For example, components for the pilot-scale system were fabricated using a 3D printer.

Laboratory-scale testing of an osmotic dehydration system dewatering FCPW feeds achieved 70%–80% solid content (see Fig. 13.8). Average flux is achieved, while dewatering as-received coal refuse slurry at ~25% starting solid content to a final solid content of ~75% was approximately 1.25 L/m^2/h (LMH). When processing washed coal refuse slurry (i.e., lowering the osmotic pressure of the slurry and increasing driving forces), average flux achieved for dewatering feed slurry with starting solid content of 25% to a final solid content of ~80% was approximately 3 LMH. Amazingly, this degree of dewatering was achieved without the application of any

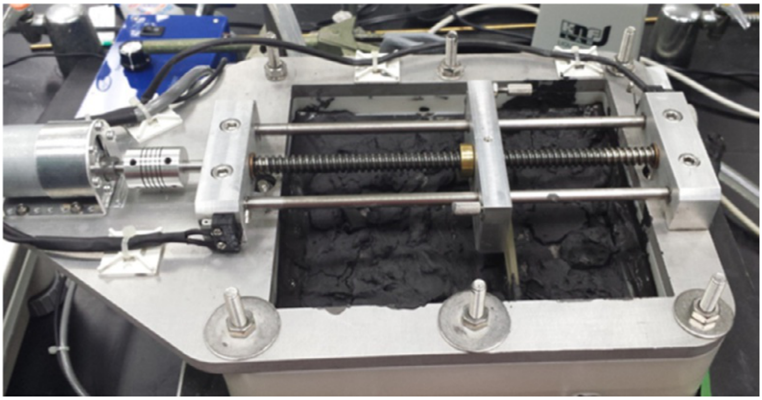

Fig. 13.8 Laboratory-scale osmotic dehydration test cell including automated membrane scraper [21].

mechanical pressure. The system used in these experiments was a batch configuration with freestanding, handmade, cylindrical membrane bags filled with feed slurry. They were immersed in a reservoir of draw solution causing the sides of bags to collapse as water was removed, thereby creating a moving wall effect. While this batch of configuration was effective in proving the concept, it would be challenging to scale up. A continuous forward osmosis (FO) system using membrane disks was also tested, but it was not as successful in achieving deep dewatering due to impermeability of cake layer that developed on the membrane. This led to creating the automated scraping device seen in Fig. 13.8, which was designed to mix the feed solution during dewatering in an attempt to maintain intimate contact between water in the feed slurry and the membrane by provisioning channels for water flow.

Scale-up testing of the laboratory devices focused on membrane robustness, which was tested by using the same membrane coupon repeatedly for over 30 trials with no deterioration in performance observed. Throughout these tests, the membrane was easily cleaned with a simple water rinse. As part of Rajagopalan's research, the continuous system FO cell was scaled by a factor of 20 in terms of membrane area, and the amount processed was scaled by a factor of 60. The performance achieved with larger cells was comparable with that obtained with smaller cells validating the feasibility of scale-up of continuous dewatering configuration.

The cost of dewatering a ton of solids is dependent on starting slurry concentration, target slurry concentration, flux (measured in LMH), membrane life, and mode of draw solution regeneration. An estimate of the cost to dewater a ton of solids given the following parameter values, average flux of ~2 LMH, membrane cost of $20/m^2$, 3-year membrane life, and a target solid content of 75%, arrived at around $3.50. This estimate assumed the use of RO for osmotic agent regeneration. This estimate compares favorably with costs for alternative technologies that generally achieve inferior dewatering success (measured in percent product solid content), such as deep cone thickeners (45%–50%), belt presses (25%–45%), and hydraulic filter presses (20%–80%) [22].

In summary, osmotic dehydration of fine coal refuse and ultrafine clean coal has been shown to be technically feasible, scalable, and potentially economic. Further research is focusing on lowering membrane replacement costs through productivity enhancements such as decreasing cake resistance and maximizing driving forces (i.e., lowering the osmotic pressure of the feed solution). The latter is achieved through management of total dissolved solids (TDS) and water washing. For example, TDS in the feed solution used for laboratory-scale testing were ~8500 mg/L (0.07 lb/gal), which is more than double reported values for several Illinois Basin coal mines. Processing slurries with lower TDS is expected to result in higher average flux. Cake resistance can be lowered by increasing particle size within slurries or by increasing the sphericity of materials in the cake bed. A doubling of the productivity reported by Rajagopalan is considered achievable with adjustments to slurry and cake properties and membrane material. Finally, further development of the technology should focus on continuous process systems with integrated solids handling within membrane cells.

13.2.3 Backfilling

Following the 2008 coal fly ash slurry spill at the Tennessee Valley Authority's Kingston Fossil Plant located in Harriman, TN, the United States, the US EPA initiated efforts to classify all coal waste as hazardous. Facing the possibility of significant increases in waste disposal costs, coal mine operators began exploring less environmentally intrusive waste disposal methods that could be implemented cost effectively. Backfilling or underground disposal was to some a natural solution. Internationally, there have been several mines employing the practice over a long period of time; however, in the United States, the MSHA has been very opposed to underground placement of any type of waste that could contain combustible material, especially in the wake of the Upper Big Branch mine explosion that killed 29 miners. The US EPA also expressed concern that underground placement would make it difficult to detect and monitor for ground water contamination. Therefore, research on backfilling has focused on developing materials that set up quickly, minimizing the risk of spontaneous combustion or propagation of an explosion due to air passing through unconsolidated material that contains combustible matter. Such material would have little or no permeability, which also minimizes the potential for ground water contamination after placement.

In addition to addressing the waste disposal issue, the authors believe that backfilling offers other significant advantages that derive from backfill material providing supplemental ground support in mine openings. For example, strategic placement of backfilling material has the potential to reduce or eliminate surface subsidence caused by mining, increase extraction ratios so that a greater percentage of reserves are recovered, and eliminate the need for costly explosion proof seals in mined-out areas.

Accurately determining and properly manipulating the rheology of backfill material is undoubtedly the key to designing a successful underground placement program. Two studies involving the authors have shown that there are multiple possibilities for creating backfill mixtures that can be pumped and placed underground as backfill. In one study [23], a mixture of 67% CCRs and 33% FCPW was used to make a paste backfill that was injected through boreholes into underground mine openings. Roughly 14,000 tons (15,300 US tons) of material was placed at an underground depth of 100 m (325 ft) with a maximum flow distance of 100 m (325 ft). In the second study [24], coal spiral waste was tested (pumped) with and without admixtures in 5-cm (2-in) and 7.6-cm (3-in) pipe loops. Results showed that paste at a specific gravity of ~1.65 could be pumped at flow rates approaching 30 m^3/h with maximum pressure of 1000 kPa. Rheology is not a one-size-fits-all characteristic, and each potential fill material must be tested to obtain its rheology before designing backfilling systems and layouts; however, these projects demonstrated that backfill material can be produced that meets requirements for a high-density material that gels following placement to achieve low permeability.

Engineering an economical backfilling program must examine several important parameters. Among these are the cost and layout of pump and piping systems to achieve maximum coverage (horizontally and vertically) in mined-out panels and the effect of backfill material on mine floors and coal pillars. Regarding the former, in a follow-up to the 2010 Spearing study, an economic analysis indicated that employing paste backfilling in a typical Illinois Basin room-and-pillar coal mine would increase current operating costs by \$0.86 (about 2.5%–3.0%) per clean ton mined over the conventional practice of FCPW impoundments [10]. This analysis ignored potential productivity increases due to increased extraction from secondary mining, which may be significant. Regarding the latter, modeling performed in the same follow-up study indicated that a paste backfill regimen utilizing a minimum backfill strength of 150 kPa (22 psi) at a minimum fill height of 50% of pillar height could significantly increase long-term stability of coal pillars in the Illinois Basin [10].

13.2.4 Reduce, reuse, and recycle

The first priority in developing sustainable coal waste disposal practices of the future has to be reducing the amount of waste produced. This strategy is covered in detail in Chapter 11. Some reuse and recycling strategies were previously mentioned in this chapter. Because of fly ash's chemical properties, cement plants have been recycling it for decades, using it as a substitute for Portland cement in concrete, a practice that is coming into increasing favor as "green" concrete. Every ton of Portland cement made the conventional way (heating limestone and clay to thousands of degrees) creates 0.8 tons of CO_2. Thus, every ton of fly ash used in concrete eliminates almost an equivalent amount of the greenhouse gas. However, there are some barriers standing in the way of that happening on a broader scale. The chief one is transportation costs. Like aggregate, transporting fly ash over long distances destroys the economics of using it. Cement plants need a reliable, steady supply that is local meaning that green concrete will only be produced in close proximity to a power plant. The same can be said for synthetic gypsum wallboard, clay bricks, and many of the other products previously mentioned that are made from recycled fly ash.

However, fly ash has a number of other qualities that have been shown to be ideal for high-value, specialty applications, which might change the transportation component fly ash recycling economics. For example, hollow fly ash particles are strong enough to be used as a lightweight additive in many metals. Researchers have developed and tested aluminum metal-matrix composites (MMCs) made with up to 50% fly ash [25]. MMCs made with 20–30 wt% fly ash were found to be as strong as metal only but much lighter leading the US Office of Naval Research to explore whether they can be used to make lighter armored vehicles or ships [13].

Coal ash particles also have a chemical structure that can easily be manipulated to absorb oil. Researchers funded by the US National Science Foundation patented "functionalized" fly ash particles that absorb oil but repel water [26]. The idea is to place them in booms on the surface of an oil spill where they become oil-saturated, then to recycle them back into a power plant to use the energy of the spilled oil. Testing has shown that this process prevents leaching of toxins from the fly ash [13].

13.3 Summary

Facing the concurrent trends of increased environmental awareness among the nonmining public and decreasing margins of profitability in the coal-mining business, mine operators and all those who provide technical support to them must examine the sustainability of current practices in all facets of their operation but especially their waste handling operations because, while they may not be the most glamorous side of the business, they are surely the most noticeable to the general public. Because coal reserves are finite resources, addressing sustainability in coal-mining ventures may seem counterintuitive; however, "an enterprise that contributes to sustainable development enhances its own sustainability as a business" [27]. If existing coal mines are to continue operating and new coal mines are to be developed, all involved must incorporate sustainability concepts into their everyday thinking and practice. As one Australian scientist noted, "The value of attention to sustainability is not so much in what it stops us from doing but in what it encourages us to do differently" [28].

In that spirit, this chapter has called attention to various strategies that can be adopted separately or in concert with one another. The first codisposal of coarse and fine processing waste with the objective of eliminating liabilities is associated with fine waste slurry impoundments. Taking this course of action requires investing in newer, state-of-the-art dewatering technology. Two such technologies were briefly discussed: one that is already commercially available and one that is still under development. The authors are hopeful that work will continue to develop osmotic dehydration and other as yet undiscovered methods for meeting the challenge of dewatering fine coal waste so that it may become a usable raw material, either for mining applications such as backfilling or perhaps in some revolutionary concept that is currently only a figment of imagination in the mind of a future great scientist. Development and implementation of these and other innovative concepts are proposed in an effort to guide the formulation of policies and practices that are both environmentally acceptable and economically sustainable and will satisfy requirements of stricter standards for coal waste disposal practices that are sure to be faced by coal mine operators in the future.

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Sustainable reclamation and water management practices

14

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14.1 Introduction

Contemporary mining practices can create environments that differ from preexisting conditions, yet those preexisting conditions are expected to be largely re-established during reclamation. As public awareness of mining impacts increases, regulatory and public expectations of the mining industry also advance. As a consequence, environmental laws that regulate the mining industry have become more complex and challenging. Miners are expected to produce reclaimed mine lands that will support economically viable postmining land uses while protecting environmental quality and, for some uses, to re-establish ecosystem structures and processes that often resemble premining conditions.

Given advances in public awareness and policies, it is in the interest of mining firms to achieve reclamation and environmental restoration goals. Improved environmental protection and restoration practices and outcomes will result in greater public acceptance of mining that in turn influence regulatory policies and actions. It is clear that effective reclamation must be fully integrated with mining operations. Successful reclamation demands engineering, design, and careful reconstruction of the full mining disturbance; and mitigation of mining effects on the land surface, air, and water. A capability to produce lands that can support viable postmining use and sustainable ecosystem services, as needed to satisfy regulatory requirements, is crucial to the long-term success and profitability of the mining industry.

This chapter provides an overview of mine reclamation methods that can be utilized to restore land use capability and environmental quality. Because the experience of most of the authors is primarily in the Appalachian region of the United States of America (USA), reclamation practices of that region are emphasized. In accord with the book's theme, special consideration is given to reclamation and water management practices that can address emerging regulatory and environmental management challenges faced by the global mining industry.

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14.2 Legal and regulatory context

Coal mining is a multifaceted and complex landscape disturbance. As such, it is regulated by a variety of laws and policies, and is subject to numerous reclamation requirements.

14.2.1 Surface mining control and reclamation act

In 1977, the Surface Mining Control and Reclamation Act (SMCRA) was enacted in the USA. The Act stipulates that in order to mine coal legally, a mining firm must apply for and obtain an SMCRA permit. Almost every activity of any coal-mining operation is affected by this law's provisions.

In particular, the SMCRA governs land reclamation operations of surface coal mines. In order to obtain an SMCRA permit, mine operators are required to prepare a reclamation plan that describes lands targeted for mining operations, and plans for reclamation operations that will restore those lands to a condition that will comply with SMCRA requirements. Important provisions include the requirement to return the land to its "approximate original contour" (AOC) unless a variance from that requirement is obtained. The mine operator is also required to "restore the land affected to a condition capable of supporting those uses which it was capable of supporting prior to any mining, or higher or better uses of which there is reasonable likelihood …," which is often stated as a requirement that the reclaimed mine land use should support an "equal or better" land-use capability than that which was present prior to mining.

In addition, the law requires the mine operator to "effectively control erosion and attendant air and water pollution," and, if an unmanaged postmining land use is planned, to establish "a diverse, effective, and permanent vegetative cover of the same seasonal variety native to the area of land to be affected and capable of self-regeneration and plant succession."

The SMCRA also includes numerous other requirements that concern documentation of premining conditions, the mining operation's influence on the "hydrologic balance," disposal of excess spoil that is generated by the disturbed materials' physical expansion, and numerous other mining impacts. These requirements are reviewed in greater detail by Skousen and Zipper [1].

14.2.2 Clean water act

The Federal Water Pollution Control Act of 1948 was amended in 1972 and became commonly known as the Clean Water Act (CWA). As per promulgations of the CWA, coal mine operators must also comply with requirements of the CWA that regulate the quality of surface waters. The CWA's stated purpose is to "restore and maintain the chemical, physical, and biological integrity of the nation's waters." The CWA's permitting requirements are based on the concept of "designated use," meaning that surface water bodies are expected to remain suitable for beneficial uses, including but not limited to support of aquatic life, recreational activities such as swimming and fishing, and maintaining water supply provisions. The CWA is interpreted to require that all regulated water bodies maintain balanced, indigenous populations of aquatic flora and fauna.

Because SMCRA requires that mine operators monitor discharged waters so as to assure compliance with "applicable Federal and State law in the receiving stream," all waters discharged from the mine site must comply with CWA requirements. Hence, mine operators must apply for and receive CWA permits for all water discharges. These permits place limits, called "effluent limitations," on concentrations or quantities of pollutants that may be discharged. Effluent limitations are established with the intent of limiting discharged pollutants to levels that will enable the receiving stream to continue to support aquatic life and other designated uses following mining activities [1].

Historically, primary water-quality concerns for eastern USA mining operations have included sediments, acidity, and metals such as aluminum (Al), iron (Fe), and manganese (Mn) that are often released in association with acidic waters. These releases are regulated under the CWA by effluent limitations described by federal regulation 40 CFR 434 (Table 14.1). Other acid-soluble metals such as copper (Cu), nickel (Ni), and zinc (Zn) can also be released at elevated concentrations in acidic discharges. In-stream concentrations of these elements are regulated by state water-quality standards, established under the CWA, that are either similar to or exceed the USA Environmental Protection Agency's water-quality criteria guidance [2].

In recent years, eastern USA coal-mining operations have become more keenly alerted to two new water-quality concerns: total dissolved solids (TDS) and selenium (Se). TDS applies to nonvolatile elements dissolved in water, most of which are of

Table 14.1 Technology-based effluent limitations applied to active coal-mining operations and postmining reclamation areas by Clean Water Act regulations (40 CFR 434)[a]

Pollutant, or discharge property	Maximum for any one day	Maximum average daily values—30 consecutive days
Iron, total		
New sources (post 5/84)	6.0 mg/L	3.0 mg/L
Other sources (pre 5/84)	7.0 mg/L	3.5 mg/L
Manganese, total	4.0 mg/L	2.0 mg/L
Total suspended solids[b]	70.0 mg/L	35.0 mg/L
Settleable solids[b]	0.5 mL/L	n/a
pH	[c]	[c]

[a]Not all standards are applied under all conditions; alternate effluent limitations may be established and/or applied under conditions defined by Clean Water Act.
[b]Applies only to postmining areas, "not to be exceeded."
[c]Within the range 6.0–9.0 at all times.

mineral origin. Total dissolved solids can be measured by evaporating a water sample at 180°C and comparing the mass of the evaporative residue to that of the original water sample [3]. TDS concentration can be estimated by summing concentrations for all dissolved constituents. The water's electrical conductivity (EC) is often used as a proxy for TDS because it corresponds closely to TDS [4], and because it measures the concentration of solutes in water and is much easier to measure. Because both water's dissolved ions and its temperature affect its EC, those EC measurements are usually converted to specific conductance (SC), a measure of the water's capability to conduct electric current at a temperature of 25°C. Both TDS and SC have become mining industry concerns because numerous studies have documented depressed aquatic life in mining-influenced streams with elevated SC [5]. As of this writing, no federal regulatory standards are in place for TDS or SC, although recent studies by USA federal agency scientists have suggested that SC levels $<300\,\mu S\,cm^{-1}$ and $<500\,\mu S\,cm^{-1}$ can protect aquatic communities and sensitive aquatic taxa, respectively [6,7].

Selenium (Se) can also be problematic in mine discharges. Selenium is a naturally occurring element that can be released at elevated concentrations by environmental waters' interactions with some mining-disturbed strata. Selenium is an environmental concern because of its tendency to bioaccumulate in aquatic systems, causing reproductive abnormalities in higher-trophic level organisms such as piscivorous fish and birds [8]. US EPA's recommended water-quality criterion for Se is $3.1\,\mu g\,l^{-1}$ or a demonstrated absence of dangerous bioaccumulation in flowing waters [9].

In addition to obtaining water discharge permits, most mining operations must comply with other CWA requirements. For example, mining impacts to streams are regulated under the CWA's Section 404, a program that oversees the discharge of dredged or fill material into waters of the USA. For example, reductions in stream distance can occur with mining operations as ephemeral, intermittent, or flowing streams are covered by excess spoil disposal structures such as valley fills. Such losses must be approved through permitting. The law requires that mining operations be designed and operated to avoid stream losses when possible, to minimize such losses when avoidance is not possible, and to mitigate whatever stream losses do occur. The "compensatory mitigation" that is required in order to obtain a CWA Section 404 permit is commonly achieved by improving the watershed and habitat conditions of other streams in the vicinity of the mine and by reconstructing the impacted stream on the reclaimed mine site, when possible.

14.2.3 Other environmental laws

Depending on the mining situation and reclamation plan, other environmental statutes can come into play. For example, the Endangered Species Act (ESA) protects threatened and endangered plant and animal species. If the mining operation has potential to affect habitats for any such species, the ESA's requirements must be complied with. If disposal of coal combustion products such as fly ash from coal-fired power plants is planned for the mining area, Resource Conservation and Recovery Act (RCRA) requirements must be considered when developing the mining permit.

14.3 Reclamation practices

At contemporary mines, reclamation strategies are often fully integrated with mining plans and operations because management of geologic materials is essential to achieving environmental goals. Overburden, interburden, and/or in-seam partings are all waste materials that are being handled and managed at every step of the mining process. These materials vary widely in their chemical and physical properties and hence, in their potential to influence postmining revegetation and water management. In this chapter, the term overburden covers all noncoal waste material that must be dealt with in the reclamation process.

14.3.1 Overburden analysis

Overburden characterization is essential to mine planning and environmental concerns. Traditionally, overburden characterization has focused on identification of highly acidic "toxic" materials. For reasons discussed earlier, overburden analysis also addresses potentials for release of TDS and Se.

Both chemical and physical properties of overburden are important considerations for material management. Subsurface drainage structures require hard, durable rock. Topsoil substitutes, if permitted, should have both chemical and physical properties that are suited to the postmining land use. Any overburden materials intended for placement in locations exposed to environmental waters should be selected to minimize release of acids, TDS, and Se.

The amount and distribution of acid and alkaline strata (the mineralogy of rock types) disturbed by the mining operation, along with various interactions, ultimately controls the acidity or alkalinity of the drainage [10,11]. Therefore, premining analyses of soils, overburden, and materials immediately underlying the coal are required by law to ascertain the physical and chemical characteristics of the strata above and below the coal bed. Overburden characterization provides important information about rock strata that are acid forming, neutral, or alkaline producing, as well as strata that may promote drainage waters high in TDS, Se, or other problem elements. A number of tests are available for overburden characterization and they can be broadly defined into static (whole rock) or leaching techniques ([12,13]a). Each has specific uses in premining water-quality prediction. Since SMCRA requires coal operators and regulators to predict whether water-quality concerns may occur at a potential mine site, the analysis is critical to provide a quick and accurate prediction and to limit the long-term liability and expense of treating a polluted water discharge.

The most common static test for overburden analysis is acid-base accounting. It characterizes physical and chemical properties of rock strata in terms of its acid or alkaline production potential. In the method, each rock strata is identified by rock type, color, and thickness. Ground samples of each rock strata are subjected to sulfur analysis to determine their acid-generating potential, and to carbonate analysis to determine their alkaline-producing potential. These potentials are then combined to evaluate the likelihood for acid or alkaline drainage [10,11,14,15]. Acid-base accounting procedures have been refined over time as more stringent effluent limits have been implemented [16].

Material characterization is essential to development of mining practices that will reduce pollution in waters discharged from disturbed areas. Where neutralization potential is sufficient to overwhelm any acids that will be produced, then no special handling plan is needed for the purpose of controlling acidity. In such situations, the entire overburden profile can be blasted and mixed together when constructing the mine backfill if the primary goal is minimizing discharge acidity. However, if the acid-producing potential is greater than the amount of neutralizers, then special handling plans or alkaline addition is required to control pollution [17]. Skousen et al. [14] found that overburden analysis by acid-base accounting was >90% accurate in predicting postmining drainage quality from 56 surface mines in West Virginia. However, it can be expected that the common practice of blending net acid-producing spoils with alkaline spoils or reagents, so as to attain a net-alkaline discharge, in hydrologically active locations would accelerate the release of TDS compared to the alternative procedure of isolating acid-forming materials [18]. Concerns with other pollutants such as Se can also be evaluated by analysis of whole rocks where solids are extracted with a strong acid and the concentrations of Se in the rock are measured. Thresholds can be established where rock materials with certain contents (such as $\geq 20\,\mathrm{mg\ Se\,kg^{-1}}$) are specially handled so as to reduce their contact with environmental waters for leaching.

Leaching tests are an alternative method for overburden characterization. Tests are conducted by breaking or grinding rocks into smaller pieces, subjecting those rock fragments to repeated leaches with defined water volumes, and analyzing leachates for chemical composition. The most common leaching tests are humidity cells, soxhlet extractors, and column-leaching procedures [12,19,20]. Leaching tests provide information in addition to that provided by acid-base accounting [14,21].

Orndorff et al. [22] and Daniels et al. [18] conducted leaching tests to characterize mine overburdens for TDS release. They showed that unweathered rock materials typically release waters with higher SCs than do weathered rock materials and that fine-textured sedimentary rocks (siltstones and shales) typically release waters with higher SCs than do more coarsely textured sedimentary rocks (sandstones). In this context, the term "weathered" refers to those rock materials that occur in the upper portion of the overburden column, typically extending for depths of 10–50 ft (3–15 m) directly beneath the soil. Weathered rock materials have been affected by percolating rainwater, and therefore differ both visually and chemically from unweathered rocks below them.

Orndorff et al. [22] and Daniels et al. [18] also showed that tested geologic materials typically release waters with the highest SCs early in the leaching process, with subsequent leaching producing lower SCs. Daniels et al. [18] found that common laboratory procedures can be conducted with less time and effort than leaching studies, such as short-term saturated-paste or static SC measurements of water:spoil mixtures that correspond with peak SC levels in leaching waters; and SC levels in waters produced by column leaching tests that correspond with field observations.

14.3.2 Landform design

Postmining land use and SMCRA's AOC requirements are essential considerations in landform design. Geologic materials expand upon disturbance; hence, in the eastern USA where coal seams are thin relative to overburden strata, postdisturbance spoil volumes typically exceed the volume of material prior to mining. Overburden expansion or "swell" factors can range from 15% to 40%, depending on rock type and degree of fracture. Historically, mine operators disposed of excess spoils in "valley fill" structures placed in topographic depressions below the mining operations. In recent years, however, valley fill construction has been discouraged by regulatory policies due to water-quality concerns, as valley fills often give rise to waters with TDS concentrations that are elevated relative to natural background levels [23,24]. In response, mine operators have been developing alternative excess-spoil disposal strategies, including placement in abandoned pre-SMCRA mine sites for use in mitigating long-standing environmental problems.

In the western USA, coal seams are often thick relative to overburden. Therefore, even though overburden swells during disturbance, postmining landforms often occupy smaller volumes than the undisturbed landforms present before coal removal and postmining elevations can be much lower than premining elevations.

Prior to mining, the process of designing landforms, and methods that will be used to construct such landforms, may consider the existence of problematic spoils, if such are expected to be disturbed by the mining operation. Historically, spoils have been considered as problematic if containing significant concentrations ($\geq$0.5%) of sulfur (S) and lacking alkaline neutralizers sufficient to offset the S-oxidation-generated acids. Under current environmental policies, spoils are considered problematic if they have high TDS generation potential, and/or a high potential to release elevated Se. In the Appalachians, these problems can coexist with spoil materials exhibiting high concentrations of S (i.e., potential highly acidic spoils) and thus the capacity to generate high levels of TDS [21], because S oxidation generates TDS as well as acidity. Given the tendency of Se to substitute for S in certain mineral forms, such spoils may also be prone to release of high quantities of Se. Premining overburden testing procedures can identify problematic spoils in advance of mining, such that the mining plan can be developed to manage those spoils in a manner that is both cost effective and protective of the environment. Some of these methods are described in the remainder of this section.

14.3.3 Geomorphic reclamation and hydrologic concerns

The term "geomorphic reclamation" describes reconstruction of mined landscapes that resemble premining landscape forms and restore water drainage patterns similar to those present before mining. The outcome of geomorphic reclamation is a more natural look and more intricate drainage patterns than typical engineered structures with linear features. Similarly, geomorphic reclamation landscapes are intended to remain more stable over the long term than nongeomorphic land configurations produced by conventional reclamation and characterized by linear features. That enhanced stability, if successfully achieved, will minimize movement of sediments into water sources as the postmining landform ages over multidecade time periods. An aerial survey of a West Virginia mining complex, reclaimed using conventional methods, found that slope distributions of postmining areas differed substantially from those of premining landscapes [25].

Design of geomorphic-reclamation landforms entails estimation of landscape features such as drainage-basin area, weighted mean slope, and drainage density of nonmined areas; and construction of postmining landforms that mimic those features [26]. But replication of premining slope distributions is characteristic of the geomorphic reclamation method. Construction of slopes with sigmoid profiles (convex-traight-concave) as an alternative to linear slope designs can enhance land surface stability [27]. Breaking long slopes with terraces and construction of stream channels with meanders resembling natural landscape drainages can reduce down-slope water-movement velocities and sediment-transport potentials.

The flow of water from highlands to oceans is an essential landscape process, and natural stream channels are often excavated and/or filled by mining operations. Hence, stream channel reconstruction is also an essential practice when conducting geomorphic reclamation. Dimensions, patterns, and profiles of reconstructed stream systems are determined by assessing essential features of natural, stable streams of the region [28]. Construction of stable, natural-looking channels that mimic natural stream forms on mined landscapes often requires strategic placement of stable rocks to establish a channel base and produce pools and riffles. Restoration of premining stream "functions," such as organic matter processing, is often seen as a desirable stream reconstruction outcome [29]. Recent research has found that establishment of woody vegetation in the riparian areas of constructed streams, and creating soil conditions that enable the woody vegetation's survival and growth, can aid functional restoration in streams that are constructed at mine sites [30].

14.3.4 Mine-site preparation and initial excavation

As a first step of mine-site preparation, sediment control structures are established. Water channels and ponds are constructed to manage water runoff from the disturbed area. Sediment retention ponds are essential features, given that mines will expose unvegetated soils and rock fragments that become subject to movement by rainfall runoff. Retention times and hence volumes available for water retention are regulated based on area and climate (e.g., in West Virginia the regulated retention capacity is 0.125 acre-foot of volume per acre of disturbed land). It is essential that ponds have sufficient capacity to store runoff waters as needed and capture sediments so that discharged waters will satisfy permit requirements.

Once sediment control structures are in place, timber (if present) is harvested. Soils, vegetation, and posttimber harvest vegetative debris (if present) are removed to initiate the mining excavation. If the intended postmining land use involves natural ecosystem restoration, the soil and associated organic materials should be salvaged for use in reclamation. When possible, salvaged soil and organic materials should be moved to and placed in reclamation areas immediately; otherwise, they should be stockpiled for later reclamation use. Prompt placement of soil and associated organic materials can help to ensure that viable seeds, live roots, and soil animals (such as earthworms, soil microbes, etc.) contained in the soil remain alive and able to assist the reclamation process. Soil is essential for rapid restoration of natural ecosystem structures and processes after disturbance [31,32].

After soil removal, bulldozers construct benches for drill rigs to bore holes for blasting reagents. Blasting activities are carefully controlled and monitored, so no off-site damage occurs, especially damage to nearby roads, buildings and other structures. Once the overburden has been fragmented and fractured by blasting, surface mining equipment such as draglines, front-end loaders and trucks, and bulldozers remove the overburden to reach the coal. The blasting sequence and pattern are essential to environmental controls, as successful reclamation requires that geologic materials be managed in accord with their physical and chemical properties. The blasting sequence and pattern will influence the ease with which adjacent geologic strata can be separated from one another at the loading site, when such separation is required for effective environmental control. The blasting pattern can also influence rock-fragment size distributions as needed for reconstruction of the mined landform.

14.3.5 Backfill construction to isolate problematic materials

Once coal is removed, filling the pit or backfilling the highwall begins with material from adjacent pits. Overburden materials with suitable chemical characteristics are used to fill the pit. If problematic spoils have been identified, special provisions are made to ensure that they are disposed of in a manner that achieves "hydrologic isolation," meaning that they are placed such that they are not subject to in situ leaching by environmental waters. Such materials can be segregated during overburden excavation and placed above the water table in a 3- to 5-m layer of problematic material that is solidly based (i.e., compacted) on the undisturbed strata below the lowest coal seam ("pavement"). This layer is then covered with nonproblematic material that is also compacted and has a pitched surface to shed water. Although these procedures were developed for isolation of highly acidic strata [10,11], they can be applied to isolate high-TDS and/or high-Se strata as well.

In some cases, acid material can be managed by mixing with nonacidic material containing sufficient alkalinity to neutralize the acidity, but the acid-neutralization reaction tends to generate high TDS concentrations in water. Therefore, more emphasis is being placed on isolating acidic materials. Materials traditionally used for enveloping and sealing acid-producing materials in the backfill, such as fly ash or kiln dust, must now be scrutinized for use since they may cause elevated TDS concentrations.

The hydrology of mine backfills is complex and is under study [33]. Fine-grained materials such as siltstones and shales, when compacted, tend to have lower infiltration rates than more permeable sandstones. The quality of water emerging from a backfill is affected by all rock types found in flow paths, and constituent concentrations are influenced by contact time and geochemistry. Therefore, mine water chemistry can be directly influenced by directing water through rock materials with minimal chemical reactivity, such as low-TDS durable-rock sandstones [34]. Subsurface drains constructed of low-TDS durable-rock sandstones can be placed within and at the base of backfills that receive groundwater influx for the purpose of draining that water to daylight along a low-TDS drainage pathway.

14.3.6 Surface and mine-soil construction

A “mine soil” is a soil-like material that is placed on the reclaimed mine surface and that often develops form and properties similar to natural soil over time [35]. Mine-soil construction involves selection of disturbed materials, and placement of those materials on the reclaimed land surface. Ideally, materials are selected with the intent of establishing a mine soil that is suitable for the proposed land use. Disturbed materials can vary widely in physical and chemical properties; and hence, in their suitability for soil construction [31].

Soil contains organic matter and is often the most favorable material if natural ecosystems are to be restored. Moving soil materials directly from the excavation area to the reclamation area is a recommended practice that can maintain beneficial microorganisms, and the viability of seeds and propagules from plant species that were present prior to mining disturbance. Soils are recommended for salvage and use in mine-soil construction in Appalachia [32] and in other world regions [36–42]. The term “soil” (as used in the Appalachian Region) refers to all surface soil material to a depth of broken bedrock that can be removed with a dozer.

Weathered spoils are the rock materials that occur directly below the soil [32]. Generally, weathered spoils will have more favorable chemical properties for native vegetation and will break down into soil-like materials more easily than unweathered parent rock. Weathered spoils will generally have less capacity to generate TDS than unweathered rock materials [22]. However, weathered spoil will lack organic matter and plant nutrients that are present in salvaged soil.

Unweathered spoils occur below the zone of environmental weathering (generally deeper than 10 m below the premining surface). Unweathered spoils can vary widely in physical and chemical properties. For example, Roberts et al. [43] describe siltstone-derived mine soils that declined in pH from 7.5 to 6.3 within 2 years. Conversely, Emerson et al. [44] describe soils constructed from gray unweathered sandstones that retained soil pH levels >8 over a similar time period. Some unweathered spoils break down easily into soil-like fragments and maintain soil pHs that are favorable for pasture grasses and legumes [45]. If unweathered spoils are to be used for soil construction, it is necessary to select certain strata that have both physical and chemical properties that will be favorable for the postmining land use as initially placed and as they weather. When re-establishing natural ecosystems, use of natural soils and weathered spoils for mine-soil construction is recommended when available [31,32]. If such materials are not available, unweathered spoils should be selected based on capacity to achieve pHs that are similar to those of natural soils (e.g., moderately acidic in Appalachia) and relatively low-soluble salt levels.

Respreading of native topsoil is important when establishing unmanaged postmining land uses for many reasons. First, viable seeds and propagules contained in the soil (called a seed bank) enable restoration of native species. Second, organic matter in the native soil contains soil nutrients, such as nitrogen and phosphorus, not readily available to plants in mine overburden materials, but essential for plant growth. Third, use of native soils, with organic materials salvaged from the premining landscape, promotes favorable hydrologic properties. Finally, soil-dwelling animals and microorganisms in the native soil aid in creating channels for air and water movement that is essential to productive vegetation.

Salvage and reuse of native soil is also important when conducting reclamation for a postmining use such as cropland. For example, soil consists of visually and texturally distinct layers that have been described by soil scientists as follows from top to bottom:

Horizon A: The top layer is often referred to as topsoil. It consists primarily of darker decomposing organic materials called humus. It is the horizon in which the most biological activity occurs.
Horizon B: Commonly referred to as subsoil, this horizon consists primarily of concentrated mineral layers, such as clay or oxides of iron or aluminum. It may contain some organic matter. Both organic and mineral matter are typically moved to this horizon by leaching.
Horizon C: The bottom layer is weathered bedrock. It may contain large boulders or shelves or parent rock that is still undergoing the weathering process.

These horizons are sometimes divided into subhorizons to account for certain special characteristics. Occasionally, these subhorizons are identified separately. For example, Horizon O (for organic) is used to identify a top layer of organic plant residue in relatively undecomposed form; Horizon E (for eluviated) is used to identify top soil that has had most of its mineral and/or organic material leached out. Research has shown that the salvage and respreading of individual soil horizons to a depth of 1.3 m (4 ft) can greatly aid in restoring the agricultural productivity of prime farmland (further details are provided later).

Soil compaction is necessary at mine sites that are being prepared for developed postmining land uses such airports, shopping centers, and industrial parks. At most mine sites, however, soil compaction is not desirable and should be avoided. Living plants extend roots into the soil to obtain water and nutrients, but compacted soils will hinder development and activity of functional rooting systems. Dense soils will physically impede root extension, water movement, and movement of air between the rooting zone and the surface. Minimizing compaction during soil spreading operations is always desirable when the postmining land use is dependent on the productivity of living plants.

Hay land, pasture, and other agricultural postmining land uses often require smooth surfaces. Experience has shown that pasture grasses and legumes are able to survive and grow on mine-soil surfaces that have been graded smoothly if soils are graded when dry, are not excessively compacted, and have favorable chemical properties. In contrast, natural ecosystem restoration typically requires loose soil conditions and does not require smooth surfaces. On such lands, even minor amounts of compaction can influence rooting by inhibiting native plant species which have less-aggressive rooting systems than commonly used reclamation grasses. Smooth soil surfaces, even when slightly compacted can inhibit water infiltration, increasing water runoff and erosion; while less intensive grading leaves loosened, and often rougher surfaces, which enhances soil moisture infiltration and retention. In addition, several studies have found that surface roughness can aid in establishment of unseeded volunteer plants that arrive as live seed [46,47].

Smaller dozers will exert less compactive effort than larger dozers during grading and should be used if practical and available. Surface materials should be spread and graded only when in a relatively dry condition, as soils are more vulnerable to compaction when in a wet condition. Once a mine-soil material intended for plant growth has been placed and graded, it is essential that mining equipment be kept off of those surfaces.

14.3.7 Reclamation for specific post-mining land uses

14.3.7.1 Prime farmland—Agriculture

Coal occurs beneath some of the most productive agricultural soils in the USA. Hence, SMCRA establishes reclamation success standards that are area specific. On prime farmlands, reclamation success is achieved if the restored soil is shown to be capable of producing equal or greater crop yields compared to what was previously grown on the site or on adjacent areas under the same levels of management.

Most prime farmland reclamation includes separate removal and handling of A/E, B, and C soil horizons. When soils are reconstructed, those horizons are replaced in the correct order and with similar depths. Research has shown that avoidance of soil compaction during soil reconstruction is essential if premining yields are to be restored. Otherwise, compacted soils should be loosened using physical means [48]. Lime, fertilizer, and mulch should be applied as needed, and vegetation should be established as quickly as possible.

14.3.7.2 Hay land pasture

Under management, many mined lands in the USA can support highly productive and sustained forage for livestock grazing if suitably reclaimed [49–53]. Forage-based agronomic systems, even on natural soils, require periodic lime and fertilizer applications. These treatments are also necessary to maintain productive and high-quality forage crops on mined lands [53].

When livestock pasture is the planned postmining land use, reclamation procedures should construct mine soils that are relatively fine textured and moderate to neutral in pH, ideally in the 6.0–7.0 pH range. More acidic materials (pH <5.5) can be used if sufficient lime is applied to raise the pH to 6.0–7.0. Legume species such as clovers are essential to nutritious pasture vegetation; such species are sensitive to soil acidity and do not persist if adequate soil pH is not maintained. Soils for hay lands and pastures should be graded smoothly, ideally with small equipment and under dry conditions so as to avoid excessive compaction. Such areas can be seeded with a mixture of pasture grasses and legumes suited to the local area (Table 14.2).

14.3.7.3 Forest

Surface mining has converted formerly forested ecosystems to nonforested conditions throughout significant areas of the Appalachian coalfield [56,57]. In Appalachia, forested ecosystems are valued because they produce saleable wood products, serve as

Table 14.2 **Common hay and pasture species for use on eastern US surface mine sites.[a] For information on seeding rates, see Ditsch et al. [54] and Skousen and Zipper [55]**

Common name	Scientific name	Type	Soil pH range	Wet soil[b]	Comment
Dactylis glomerata	Orchardgrass	Grass	4.5–7.5	P	Develops rapidly and long lived
Festuca arundinacea	Tall fescue	Grass	5.0–8.0	G	Common species does well on most mine soils. Use endophyte-resistant varieties for pasture and hay
Lolium perenne	Perennial ryegrass	Grass	5.0–7.5	P	Short lived
Lotus corniculatus	Birdsfoot trefoil	Legume	5.0–7.5	G	Grows well in mixtures, tolerates acidic soils
Medicago sativa	Alfalfa	Legume	6.5–7.5	P	Requires deep soil with good drainage, pH > 6, adequate soil P
Melilotus officinalis	Yellow sweetclover	Legume	5.5–7.0	F	More drought tolerant and competitive than white sweetclover
Panicum virgatum	Switchgrass	Grass	4.1–7.6	E	Slow to establish, productive in warm season
Phleum pratense	Timothy	Grass	4.5–8.0	P	Good-quality hay and pasture but does not tolerate heavy grazing. Fertility demanding
Trifolium hybridum	Alsike clover	Legume	5.0–7.5	F	More tolerant of moist, acidic soils than other clovers
Trifolium pratense	Red clover	Legume	5.5–7.0	F	Requires maintenance of soil P
Trifolium repens	White clover, Ladino clover	Legume	6.0–7.0	P	Sod former. Used in pastures for erosion control, soil improvement, wildlife. P and Ca are critical

[a]Fertilizers should also be applied: at least 112 kg N and 120 kg per ha P (equivalent to 275 kg per ha of P_2O_5) are recommended. If soil pH is <6, liming is also recommended.
[b]Tolerance of wet soil conditions: *E*, excellent; *G*, good; *F*, fair; *P*, poor.

habitat for both game and nongame wildlife, protect water quality in this mountainous region, provide natural beauty, and produce other ecosystem services that are valued by local residents. Conventional reclamation practices (use of unweathered mine spoils for soil construction; "smooth grading" that compacts soils; revegetation with agricultural grasses and heavy fertilization) generally produce land that is not conducive to re-establishment of forested ecosystems [58]. These lands often become occupied by plant communities that include dominance by nonnative species [59].

To improve reforestation success in Appalachia, a new reclamation method called the Forestry Reclamation Approach (FRA) has been developed [60–62]. The FRA is being widely applied by operating mines in Appalachia for the purpose of re-establishing forest vegetation. Although the FRA is intended for use in the eastern USA, similar methods are being developed for use in other mining regions [63]. The FRA comprises five steps, each of which must be executed in order to ensure successful reforestation.

FRA Step 1: *Create a suitable rooting medium for good tree growth that is no less than four feet deep and comprises topsoil, weathered sandstone, and/or the best available material.*

The surface growth medium will influence survival and growth of planted trees and colonization by native vegetation. When available, native soil should be salvaged and respread and include organic materials from the premining area such as roots, stumps, and other organic debris. When possible, the soil should be excavated, hauled to reclamation areas, and respread immediately. If soils are not available, weathered non-pyritic overburden will generally be more favorable for trees than unweathered spoils. Most Appalachian native hardwoods prefer soils with pHs in the 5.0–6.5 range. Some tree species can grow with pHs in the 4–5 range, but soil pH >7.0 limits the establishment and growth of many native Appalachian trees [31,62]. If soil and/or unweathered spoils are available, but not in adequate quantities, they may be mixed with unweathered spoils that have moderately acidic pH and soluble salts to produce a growth medium that will be more favorable to tree establishment.

FRA Step 2: *Loosely grade the topsoil or topsoil substitutes established in Step 1 to create a noncompacted soil growth medium.*

The rough soil surface left by loose grading aids forest re-establishment by allowing water infiltration, offering little resistance to root growth, and aiding volunteer establishment by providing features to hold seeds carried by wind and wildlife to the mine site. Conversely, dense soils produced by conventional smooth-grading practices impede root growth and hinder water infiltration and soil-air exchange. Small equipment may be used to grade soils only as needed to achieve the desired surface configuration. Even then, rough soil condition should be left and grading operations should be conducted only during dry soil conditions. This will produce mine soils with physical properties that are favorable for trees [64]. Soils with loose, rough, uneven surfaces, prepared using these methods, will enable more rapid water infiltration, and thus will produce less runoff and will be more resistant to erosion than conventional mine soils with smooth surfaces, even when groundcover revegetation develops slowly. Once soil construction is complete, vehicles should not enter reclamation areas.

Table 14.3 Example of a seeding application[a] for FRA reclamation on mine sites where soil conditions are favorable for forest vegetation

Scientific name	Common name	Rate (lbs/acre)	Rate (kg/ha)
Perennial Grasses			
Lolium perenne	Perennial Ryegrass	10	11
Dactylis glomerata	Orchardgrass	5[b]	6[b]
Phleum pretense	Timothy	5	6
Annual Grasses			
Lolium multiflorum or	Annual Ryegrass, or	25[c]	28[c]
Setaria italica	Foxtail Millet	30[c]	34[c]
Legumes (with inoculant)			
Lotus corniculatus	Birdsfoot Trefoil	5	6
Trifolium repens	Ladino or White Clover	3	3

[a]Fertilizer should also be applied at a rate of 50–75 pounds N and 80–100 pounds P (180–230 pounds P_2O_5) per acre (56–84 kg N and 90–112 kg P per hectare).
[b]Seed orchard grass on steep slopes only.
[c]Substitute foxtail millet for annual rye in spring plantings.
(Reproduced with permission from Burger J, Davis V, Franklin J, Zipper C, Skousen J, Barton C, Angel P. Tree-compatible groundcovers for reforestation and erosion control. US Office of Surface Mining, Appalachian Regional Reforestation Initiative, Forest Reclamation Advisory Number 6;2009. http://arri.osmre.gov/.)

FRA Step 3: Use less competitive ground covers that are compatible with growing trees.

"Tree-compatible" herbaceous vegetation mixes of species that are low in stature and are low in water and nutrient demand are recommended for seeding when reforesting mine sites (Table 14.3). Such seeding can produce vegetation plant cover sufficient for erosion control, while minimizing competition with planted tree seedlings for soil water, soil nutrients, and light [65]. These seed mixes are typically applied with lower nitrogen (N) fertilizer rates than would be used to re-establish pasture; as higher N fertilization will encourage more rapid groundcover growth and greater competition with planted seedlings. Excessive herbaceous competition will impair survival and growth of planted trees, and should be avoided when re-establishing forest on mine sites. Trees should be planted before or shortly after herbaceous seeding, so as to enable trees to get a good growing start while herbaceous groundcover is developing.

FRA Step 4: Plant two types of trees—early successional species for wildlife and soil stability, and commercially valuable crop trees.

Crop trees are long-lived species that produce saleable timber products when mature and are characteristic of the region's mature forests. Early successional trees can establish and grow quickly, but will have limited long-term growth and will not form saleable timber. Early successional trees will, however, attract seed-carrying wildlife to stimulate more diverse nonplanted vegetation. Ideally, some of the wildlife tree species will be capable of transforming atmospheric nitrogen to plant available forms (i.e., atmospheric nitrogen fixation). Generally, 500–600 crop trees per acre (1236–1483 per hectare) distributed among 4–6 or more species; and 50–100 nurse trees per acre (124–247 per hectare), also of several species, should be planted [66] (Table 14.3). Preparing the reforestation site using FRA Steps 1, 2, and 3 will enable a wide range of tree species to survive and grow on reclaimed surface coal mine lands [67], and will enable invasion by additional unplanted species that may enter the reclamation area as live seed carried by wind and wildlife. Current recommendations are to plant only native tree species [68].

FRA Step 5: Use proper tree-planting techniques.

Proper tree-planting techniques are described and illustrated by Davis et al. [68a]. Essential to successful tree planting is to prepare a planting hole of adequate size for a seedling's root system. If FRA Steps 1 and 2 have been used to prepare a loose mine soil, it will be easier for tree planters to open a hole sufficient to plant trees properly.

It is not feasible under current economic conditions to re-establish the full forest community by seeding and planting. Hence, the FRA seeks to establish forests of species that are prominent in native species, including those that are slow to disperse across landscapes naturally; to establish soil conditions that are favorable to those trees and to native vegetation and to natural successional processes; and to rely on successional processes for a more complete plant community [69]. Methods for re-establishing native plant communities, including forest trees, on mine sites in other world regions rely on similar approaches [38,40,46,70–72].

Re-establishment of essential soil properties and processes is necessary for forest restoration, and planted trees can act as catalysts in natural succession. The FRA is intent on establishing site conditions suitable for survival and growth of planted trees while also enabling colonization by native vegetation whose seeds are carried by fauna and wind (Table 14.4).

14.3.7.4 Wildlife habitat

Postmining wildlife habitat is rehabilitated similar to forestry reclamation sites, but with integration of water resources. Since woody vegetation is a primary component of wildlife habitat, FRA techniques are often applied. Use of salvaged soils, including roots, stumps, and organic debris, for mine-soil construction is advised when restoring wildlife habitat. In addition to the advantages described here for forest reclamation, these materials can provide habitat for ground-dwelling insects and other animals. Such materials are generally not available on mine soils constructed from rock spoils. As when establishing forest as a postmining land use, a mix of crop trees and wildlife trees can be established. When establishing wildlife habitat, however, the planting mix would include more wildlife-favored trees and shrubs, and more species that will supply shelter as well as sources of food [68]. For example, dogwood, redbud, and species like hawthorn, apple, and cherry might be planted for wildlife food; while crop trees that produce seed that is edible by wildlife, including (but not limited to) oaks,

Table 14.4 Tree species that are commonly planted on coal surface mines in the eastern USA. For further details and species, see Davis et al. [66]

Scientific name	Common name	Type	Comment
Acer saccharum	Sugar maple	Crop	For moist sites: eastern and northern slopes, lower slopes
Betula nigra	River birch	Crop	For riparian plantings
Carya ovata	Shagbark hickory	Wildlife	Provides habitat for Indiana Bat, an endangered species throughout eastern USA coalfields. Can mature into saleable timber
Castanea dentata	American chestnut	Wildlife	Use only varieties that have been genetically improved to resist chestnut blight (*Cryphonectria parasitica*)
Cercis canadensis	Eastern redbud	Wildlife	Understory species; produces food for wildlife
Cornus sp.	Dogwood	Wildlife	Understory species; produce food for wildlife
Liquidambar styraciflua	Sweetgum	Crop	For riparian plantings
Liriodendron tulipifera	Yellow poplar	Crop	For fine-textured soils in lower-slope positions; will often invade naturally if present in natural forests close to the mine site
Malia sp.	Crab apple	Wildlife	Produces food for wildlife
Morus rubra	Red mulberry	Wildlife	Understory species; produces food for wildlife
Pinus strobus	Eastern white pine	Wildlife	Provides winter cover and shelter for wildlife
Platanus occidentalis	American sycamore	Crop	For wet soils and riparian plantings
Prunus serotina	Black cherry	Crop	For cool climates: northern states, or high elevations elsewhere
Quercus alba	White oak	Crop	For a range of site types
Quercus coccinea	Scarlet oak	Crop	For drier sites: ridge tops, western and southern slopes
Quercus prinus	Chestnut oak	Crop	For drier sites: ridge tops, western and southern slopes
Quercus rubra	Northern red oak	Crop	For moist sites: eastern and northern slopes, lower slopes
Quercus velutina	Black oak	Crop	For drier sites: ridge tops, western and southern slopes
Robinia hispida	Bristly locust	Wildlife	Nitrogen fixer, less competitive than black locust; for moist sites and riparian plantings
Robinia pseudoacacia	Black locust	Wildlife	Fast-growing nitrogen fixer; use planting rates $<$30/acre ($<$75/ha) to limit excessive proliferation and competition with crop trees
Salix nigra	Black willow	Crop	For riparian plantings

hickory, and maple can be included in the planting mix. It is important to include some mid- and high-canopy trees in the planting mix for birds and mammals. In Appalachia, a coniferous species, such as eastern white pine, may be included in the planting mix to provide winter cover and shelter.

Reclamation practices that can be used to establish terrestrial wildlife habitat on mine sites are described by Carrozzino et al. [73] and by Wood et al. [74], while tree and shrub species for use in reclamation are described by Davis et al. [66,67] and Monteleone et al. [68]. Such habitat is best established in association with surface water sources, such as ponds, wetlands, or flowing streams, given that proximity to water is a habitat requirement of many terrestrial species.

Although nonnative plant species, such as autumn olive (*Elaeagnus umbellate*) in the Appalachian region, have been used in the past for wildlife habitat plantings, many such species have become problematic because of their tendencies to spread from areas where planted to other areas where they interfere with both managed land uses and natural ecosystem processes. Current best practice is to plant only native species for purposes of creating wildlife habitat.

14.3.7.5 Biofuel crops

Many perennial herbaceous plants have been evaluated as sources for cellulosic materials to be converted to biofuel. For example, switchgrass (*Panicum virgatum*), a warm season perennial grass native to North America, has been investigated as a biofuel feedstock [75]. Due to its high biomass production, adaptability, tolerance to adverse growing conditions, and low input requirements, switchgrass has been the most selected and planted cellulosic biofuel feedstock [76]. *Miscanthus giganteus* is another high-yielding plant with potential to produce large quantities of biomass [77].

Growing switchgrass and other biofuel crops on marginal lands offers a unique opportunity to serve two purposes: using land not planted with food crops and conservation of soil resources. Surface mined lands with good soil properties could be seeded with grasses such as switchgrass and *Miscanthus* that have significant biofuel production potential. In addition, reclaimed lands with existing cool-season grasses and legumes, such as fescue and lespedeza, could be converted to production of biofuel feedstocks. Growing switchgrass as a biofuel feedstock could be a more efficient and economical use of mined land that has not been placed in managed uses.

Studies conducted on switchgrass on mined land have shown biofuel production is achievable. On mine soils in Pennsylvania, Dere et al. [78] found that increased compost rates increased switchgrass growth compared to unamended soil, and manure combined with paper mill sludge further increased switchgrass growth. Switchgrass performance was >5 Mg/ha after the second year on a fertile mine soil in West Virginia [79], and >7 Mg/ha after 3 years [80]. *Miscanthus* achieved 20 Mg/ha on a reclaimed mine site in West Virginia after 2 years of growth [77,81]. These results demonstrate the potential opportunity of high-yielding biomass crops as feed stocks for transportation fuels on mined lands. Mined lands can be prepared for establishment of herbaceous biofuel crops using procedures described for hay land and pasture establishment, as stated earlier in this chapter.

It is also possible to grow woody biofuel crops on reclaimed mine sites. Species that have demonstrated good performance on Appalachian mined areas include willow, hybrid poplar, American sycamore, and black locust [82–84]. Although hybrid poplar grows well on some mine sites, it requires good soil drainage and repeated fertilization; and its performance is hindered where those site conditions are not achieved. Zipper et al. [59,62,84] reported rapid early growth of black locust on a Virginia mine site, but subsequent performance of those trees has been hindered by locust leaf miners (*Odontota dorsalis*), an insect pest that commonly infests this species throughout the area. One advantage of woody biofuel crops, relative to herbaceous crops such as switchgrass or *Miscanthus*, is that woody crops can be grown on slopes that would hinder operation of the agricultural equipment. Mined lands can be prepared for woody biofuel crops using procedures described for forest establishment, also previously covered. However, in contrast to native hardwoods, hybrid poplars prefer soils in the range of 6.0–7.0 pH.

14.3.7.6 Developed land uses (industrial, commercial, residential)

Reclaimed mines are widely discussed as potential development sites, but modern reclamation rarely prepares mined areas for building-support purposes. Surface stability is a critical factor affecting suitability of reclaimed mines for industrial, commercial, and residential development. Other important factors include the reclaimed mine site's access to water, utilities, and waste disposal.

Surface stability for the building development area can be achieved by using common-sense procedures that are well supported by engineering practices. A building development area (building support "pad") should be located over flat benches when possible, in an effort to avoid "differential settlement"—where the depth of settlement varies under different parts of a structure—which can damage buildings through structural distortion. The entire building support pad should be placed in a well-drained location, as water saturation can stimulate settlement of even well-compacted spoils, and its location should be surveyed so it can be located precisely after reclamation is complete.

The pad should be constructed by placing spoil in lifts of controlled thickness and composition, and should extend beyond the building's perimeter by at least 3 m in all directions. Each lift should be constructed using a relatively uniform spoil material that is compacted in place; the maximum size of rock fragments allowable in the building pad should be determined through engineering calculations. Lift thicknesses and degree of compaction should be determined based on engineering specifications considering the nature of the spoil materials and the degree of stability required by the postmining land use. For further details, see Zipper and Winter [85].

14.3.8 Revegetation and soil amendments

Once suitable materials have been placed on the surface and graded as needed for the postmining land use, the soil can be prepared for seeding and revegetation. As stated previously, the best material for use in mine-soil construction will usually include the native soil because of the important physical, chemical, and biological properties it possesses; properties that are not characteristic of mine spoil derived from rocks. Soils contain organic matter and essential nutrients, such as nitrogen and phosphorus, which are essential to plant growth. Salvaging and respreading the native soil, where available, will supply a mine soil that contains a viable seed when spread fresh, and will be favorable to native plant establishment and natural ecosystem development through the process of ecological succession.

14.3.8.1 Liming, fertilizer, and mulch

Liming materials neutralize soil acidity and add calcium, a macronutrient essential to plant growth. A soil pH between 6.0 and 7.0 is recommended since this is the range where plant nutrients are most available and toxic elements are less available. However, liming is not always essential especially when reforestation is the land use goal. Trees will grow well in soils of 4.5–5.5 pH, the pH of native forest soils.

Fertilizers containing nitrogen, phosphorus, and potassium (N, P, and K) are generally applied to the soil when reclaiming mine sites because most mine soils have relatively low amounts of organic matter, which is the primary reservoir for N and P in soils. Fertilizer recommendations for pasture grasses require large amounts of N and P for establishment and growth. Hence, a common fertilizer prescription for hay and pasture postmining land use would be 672 kg/ha (600 pounds/acre) of 10-20-10 fertilizer. This amount of fertilizer will provide 67 kg/ha of N, 134 kg/ha of P_2O_5 (equivalent to 59 kg of elemental P), and 67 kg/ha of K_2O (equivalent to 59 kg/ha of elemental K).

Forestry postmining land uses require a different fertilizer prescription. Generally, forested postmining land uses require less N than hay and pasture uses. Larger amounts of P, however, are often recommended to support the planted trees' longer-term growth requirements. Burger et al. [65] recommend that 30 kg/ha (40 pounds/acre) of N and 40 kg/ha (52 pounds/acre) of elemental P should be applied as fertilizers when establishing forests. This can be achieved by applying 180 kg/ha (400 pounds/acre) of 18-46-0 (diammonium phosphate); by applying a blend of 90 kg/ha (200 pounds/acre) of 0-60-0 with 135 kg/ha (300 pounds/acre) of 19-19-19; or with other fertilizer mixes.

Mulch improves forage and tree establishment on mined land by reducing water loss through evaporation and increasing infiltration, augmenting surface soil temperature extremes, protecting the soil from erosion through absorbing rainfall impact energy and holding soil particles together, and minimizing soil crusting [86]. Materials used as mulch include organic mulches (sawdust, bark, wood chips, animal manure, hay and straw, compost, and other litter materials); inorganic mulches (plastic, rocks, and other durable cover materials); chemical soil stabilizers (which form a protective film or cover to bind particles); and preparatory and vegetative or nurse crops (annual grains). The use of organic mulches is especially desirable because they have the potential to add organic matter to the soil, which helps to provide a reservoir of plant nutrients and aids in water-holding capacity. Application rates depend on soil properties and the slope of the land, but generally 1.5–2 tons per acre (3.4–4.5 t per hectare) of material are required by law to be applied to the surface after seeding. In many instances, a product called "hydromulch," comprising wood-fiber and paper-fiber materials, is applied as a component of a hydroseeding mixture containing lime, fertilizer, and seed. As well as serving as a traditional mulch, the hydromulch is also easily visible on the land surface, which aids in ensuring that the hydroseed mix is applied uniformly over all areas.

14.3.8.2 Plant species for reclamation seeding

The selection of plant species for seeding on disturbed sites in based on postmining land use decisions. For pasture and hay land uses in the eastern USA, a combination of cool season grasses and legumes are commonly seeded [49,55] (see Table 14.2 above). For grasslands in the western USA, different species adapted to more arid climates are generally seeded. Such species include grasses such as wheatgrass (*Agropyron* spp.), orchardgrass (*Dactylis glomeratus* L.), smooth brome (*Bromus inermis* Leyss.), wildrye (*Elymus* spp.); legumes such as sweetclover and vetch; along with shrub species such as sagebrush (*Artemisia* spp.), rabbitbrush (*Chrysothamnus* spp.), or saltbushes (*Atriplex* and *Sarcobatus*) [51,87]. Recommended herbaceous groundcover species for forest reclamation in the eastern USA are slower growing and less competitive than those used for hay land and pasture (Table 14.3), which are typically called "tree compatible." Tree seedlings are usually planted immediately after hydroseeding the ground cover. When restoring unmanaged forest as a postmining land use in the eastern US, tree seedlings of commercially valuable species that are characteristic of the region's mature forests along with some midcanopy and early successional tree species, including those that can produce fruits and seeds for wildlife within a few years, are planted [66,67] (Table 14.3).

14.4 Water management during and after mining

14.4.1 Surface water management

Water resources management has become a critical component of coal mine permitting and during the mining operation. Effective water management on the mine site is essential for maintaining environmental quality in rivers and streams that receive waters draining the mining area. Failure to manage water in a manner that minimizes pollution impacts can result in impaired aquatic biological community assemblages in streams both inside and outside of the mining area. Such impacts are often easily observed by the public, and depending on regulatory requirements, may result in litigation.

Therefore, practices are employed to reduce the contact of environmental waters with disturbed materials. Diverting surface water that would otherwise drain into a mining disturbance from above a mined site is a recommended and often-practiced method to decrease the volumes of requiring management within mined area. Surface drains or diversions are often constructed to move surface water quickly from the mining disturbance. Such drains and diversions are generally placed on spoil materials that have been compacted to become impervious or placed in areas where infiltration will not cause water-quality problems. Such drainage networks can also include upland retention ponds, to slow runoff and capture sediments, and are often developed at low points near the permit boundary to control waters that are leaving the mine site.

Water management may also include structures to move groundwater out of mine backfills quickly and in a manner that reduces water-spoil contacts. Water controlled by such structures may include groundwater entering the mining disturbance from adjacent unmined strata and those originating from rainfall infiltration on the mine site. Groundwater management structures may include French drains or other permeable channels within mine backfills to move water out of the fill and to reduce its contact time with disturbed materials. Blanket or bottom drains are constructed of coarse permeable rock to also promote transmission of water from mine backfills. Among essential water management practices are those employed to reduce water contact with spoil materials identified during overburden analysis as problematic due to the potential to release elevated concentrations of acids, acid-soluble metals, TDS, and/or Se.

On some mine sites, water management practices also involve treatment to remove chemical contaminants so as to minimize off-site pollution and to satisfy regulatory requirements. Ideally, effective problematic-spoil identification and management would enable mining to occur without need for water treatment; but disturbance of acidic strata, if present in the overburden column, can be expected to mobilize acids and, perhaps, acid-soluble metals such as Al, Fe, and Mn. Both active and passive water treatment methods are commonly applied to treat acidic discharges so as to mitigate both acidity and such metals. Active AMD treatment relies on constant addition of industrial chemicals, and requires frequent resupply and maintenance [34]. Passive AMD treatment relies on natural biological, chemical, and physical processes to neutralize acidity, and to oxidize and precipitate metal contaminants [88]. With proper premining overburden characterization and spoil handling, any water treatment that proves necessary should be temporary. If a proposed mine's overburden contains volumes of acidic material that are so large that hydrologic isolation is not possible and acidic drainages requiring treatment over long terms would be expected, it is often prudent to avoid mining in such areas.

Active water treatment methods for TDS and Se are available; however, the primary method employed to date, reverse osmosis, is very costly and is typically not employed by mining operations in the absence of strong legal incentives. Methods for passive treatment of TDS are not known. Experience has demonstrated that passive treatment systems employing sulfate reduction processes [88] can be effective in removing Se from discharge waters. However, passive treatment for Se is an emerging technology and, to the best of the authors' knowledge, has not been described in published literature.

14.4.2 Ground water management

Mining activities can impact the quantity, quality, and usability of groundwater supplies. Underground mining for coal by longwall or room-and-pillar mining methods often interrupts and depletes groundwater and can alter its quality. Surface mining can enhance the introduction of surface water with dissolved solids into shallow and then deeper groundwater systems through fractures or other conduits. The type and nature of the mining activity, the disturbed geologic strata, and alteration of surface and subsurface materials will determine how groundwater supplies will be impacted.

Because mining activities can result in impacted ground waters, enforcement of regulations is needed to minimize and/or eliminate potential problems. The SMCRA identifies policies and practices for mining and reclamation to minimize water-quality impacts. It requires that specific actions be taken to protect the quantity and quality of both on- and off-site ground waters. All mines are required to meet either state or federal groundwater guidelines, which are generally related to priority pollutant standards described in the CWA and Safe Drinking Water Act (SDWA). As water comes into contact and interacts with disturbed geologic materials, constituents such as salts, metals, trace elements, and organic compounds become mobilized [89]. Dissolved substances can leach into deep aquifers and cause groundwater-quality impacts [90]. In addition to concerns related to naturally occurring contaminants from disturbance activities, mining operations may also contribute to groundwater pollution from leaking underground storage tanks, improper disposal of lubricants and solvents, and contaminant spills. Blasting and hydraulic fracking activities can provide additional connection to surface water inputs, and underground injection of wastes can also occur during these operations.

The chemistry of ground waters in mined lands and potential levels of naturally occurring contaminants are related to the following: (i) groundwater hydrologic conditions; (ii) mineralogy of the mined and locally impacted geological materials; (iii) mining operations (e.g., extent of disturbed materials and its exposure to atmospheric conditions); and (iv) time. Movement of metal contaminants in ground waters varies depending on the chemical of concern. Solubility considerations include metals such as cobalt, copper, nickel, and zinc being more mobile than silver and lead, and gold and tin being even less mobile [89]. As conditions such as pH, redox, and ionic strength change over time, dissolved constituents in ground waters may decrease owing to adsorption, precipitation, and chemical speciation reactions and transformations.

Acid mine drainage (AMD) is one of the most prevalent groundwater-quality concerns at active and abandoned underground mine sites. If geologic strata containing reduced S minerals [e.g., pyrite (FeS2)] are exposed to weathering conditions, high concentrations of sulfuric acid can develop and form acid waters with pH levels below 2. Neutralization of some or all of the acidity produced during the oxidation of reduced S compounds can occur when carbonate minerals in proximity to the acid-producing materials dissolve [34]. Neutralization can also occur when silicate minerals dissolve, but sometimes, high levels of potentially toxic metals such as Al, Cu, Cd, Fe, Mn, Ni, Pb, and Zn may be released. For example, coal mining in the Toms Run area of northwestern Pennsylvania resulted in groundwater contamination by AMD containing high concentrations of Fe and sulfate that leached into the underlying aquifer through joints, fractures, and abandoned oil and gas wells [91].

Extensive underground mining has taken place in West Virginia since the late 1800s, and Bennett [92] estimated an area of about 610,000 ha with underground mining beneath the surface in West Virginia alone. This legacy of mining has changed groundwater quality and quantity due to intercepting and changing underground water flow paths [93]. In areas of northern Appalachia where high sulfur coal exists and no limestone layers are present for neutralization, the greatest environmental impact from underground mines has been on surface water quality from AMD [94]. Treatment by chemicals reduces the acidity and removes the metals, but active treatment is expensive and must be continued for decades [95]. Passive methods for treating AMD are also available and work well when appropriately designed for specific water conditions [88].

Water quality changes over time in underground mines [96,97]. Demchak et al. [98] observed that changes in water chemistry over time differ between below-drainage (flooded) and above-drainage (not flooded) underground mines, with flooded mines rebounding to much better water quality within a decade and unflooded mines remaining acid for much longer. Lambert and Dzombak [99] found that flooded underground mines in Pennsylvania change from very acid water to neutral or net alkaline water shortly after complete flooding [100,101]. Borch [102] found similar results in the flooded Meigs mine in Ohio and suggested the following reasons for the dramatic water-quality improvement within a few years after flooding at Meigs: (i) Pyrite oxidation ceased in flooded sections; (ii) after the initial flush, there was less readily available iron sulfate salts to dissolve; (iii) alkaline strata in the roof rock of the mine pool provided some neutralization; (iv) dilution and influx of alkalinity occurred from groundwater inflows; (v) the groundwater flow path exhibited some short circuiting, so areas of rapid transport or flow exhibited better water quality than areas of restricted water movement; and (vi) geochemical reactions, such as sulfate reduction and cation exchange, occurred along the underground water flow path, thus improving the quality before discharge.

Above-drainage mines did not show the same dramatic improvement as below-drainage mines; they tended to improve slightly in water quality but remained acidic [98,99,103]. Some sections or voids of abandoned above-drainage mines are flooded or partially flooded, which virtually removes those pyrite reaction surfaces from contributing acid products. Many other areas within the mine remain open to oxygen and water exchange and are susceptible to reaction. These exposed pyrite surfaces produce less acidity over time due to: (i) weathering products forming an iron hydroxy sulfate coating, which reduces air and water contact and release of acid products [104], and (ii) the more morphologically reactive pyrite (framboidal) is depleted first, thereby leaving the less reactive pyrite (massive) for subsequent oxidation. Therefore, changes in pyrite reaction rate and availability of surfaces in these areas can result in drainage quality improvement. Only during roof or pillar collapse are fresh pyrite surfaces exposed to the mine atmosphere and water. Once mines are closed, ventilation systems cease, which greatly reduces the availability of oxygen for pyrite oxidation. Land surfaces over underground mines can be compacted or altered to reduce the amount of infiltration, or surface cracks can be clogged, thereby inhibiting direct inflow of surface water into the mine. Roof or pillar collapses within the mine can change flow paths or create pools of water in the mine. Although all of these factors presumably decrease acidity with time, most are difficult or impossible to validate. Researchers are therefore left with empirical predictions of decline based on long-term data sets.

Demchak et al. [98] calculated a 2.2% decrease in acidity per year for 40 mines between 1968 and 2000. Wood et al. [96] calculated a slightly higher 3.3% acidity decrease per year in coal mine discharge chemistry over time in Scotland. Mack et al. [103] found a 2% to 4% decrease per year in acidity from 40 underground mines in West Virginia and modeled the decline with first-order decay rates.

Within the Coeur d'Alene District of Idaho at the Bunker Hill Superfund site, groundwater samples were found to contain high concentrations of Zn, Pb, and Cd [105]. The contamination originated from the leaching of old mine tailings deposited on a sand-and-gravel aquifer. When settling ponds were constructed to catch the runoff from the tailings, water from the ponds infiltrated into the aquifer and caused an increase in metal concentration in the local groundwater system.

Gold (Au)-mining operations have used cyanide as a leaching agent to solubilize Au from ores, which often contain arsenopyrite (As, Fe, and S) and pyrite. Unfortunately, cyanide, in addition to being toxic on its own, is a powerful nonselective solvent that solubilizes numerous substances that can be environmental contaminants. These ore waste materials are often stored in tailing ponds and, depending on the local geology and climate, the cyanide present in the tailings can exist as free cyanide (CN^-, HCN); inorganic compounds containing cyanide (NaCN, $HgCN_2$); metal-cyanide complexes with Cu, Fe, Ni, and Zn; and/or the compound CNS. Because cyanide species are mobile and persistent under certain conditions, a large potential for trace element and cyanide migration into ground waters exists. For example, a tailings dam failure resulted in cyanide contamination of groundwater at a gold-mining operation in British Columbia, Canada [89].

14.5 Conclusion

Reclamation is a mining activity that is intended to produce land and water conditions that meet human needs. Mined landscapes meet human needs when they support viable economic enterprises and other forms of community development. Reclamation practices have been described that will enable mined land to support agricultural production (including food crops, hay land, pasture, and bioenergy crops), forest production that can grow valued wood products, and developed land uses for building sites.

In order for mined landscapes to support human needs, it is also essential that reclamation processes restore environmental quality. The relatively undisturbed ecosystems that occupy many mine sites prior to mining produce environmental benefits that are valued by human society. Reclamation practices have also been described that can be applied to establish ecosystem structure, process, and function on postmining lands that have some resemblance to premining conditions.

Today's regulatory policies expect that coal mine operations will produce reclaimed lands that will either support economically viable postmining land uses while protecting environmental quality; and, for some uses, to establish ecosystem structure and processes that resemble those present prior to mining. An ability to execute reclamation practices in a manner that satisfies regulatory and public expectations cost effectively is essential to any mining business.

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The role of research in the coal-mining industry: Moving forward using lessons from the past

15

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15.1 Introduction and background

Coal mining has a fascinating history that dates back several millennia. For most of this time, coal was used as a domestic fuel to provide heat, but it became the world's primary energy source during the industrial revolution that began in Great Britain in the 1700s and swept from continent to continent over the course of the next two centuries. Today, coal is used mostly by industry with electricity generation being the primary market.

This history includes amazing advances in mining methods and startling fatality statistics. As with the industrial revolution that it fueled, coal-mining methods were transformed from an extremely labor-intensive pick-and-shovel manual operation to one that employs highly mechanized machines and relatively few people. This transformation has resulted in huge productivity gains for the industry. Even with these technological advances, mining has always been considered a hazardous occupation primarily due to the release of methane gas, whose explosive nature is compounded by the presence of coal dust; and to dangers associated with falling ground; two concerns which are ever-present in underground mines. To address these issues, organizations such as the United States (US) Bureau of Mines, formed in 1910, have been conducting ongoing health and safety research that has resulted in huge reductions in the number of coal miner fatalities and the size and scope of mine disasters.

As one who came into the coal industry during the robust growth years of the late 1970s, the author has witnessed multiple shifts between boom and bust cycles in the demand for coal and the need for skilled labor. The 1970s was a period of retrenchment. In the United States, this period brought the Environmental Protection Agency, the Mine Safety and Health Administration, the Office of Surface Mining Reclamation and Enforcement, and the Department of Energy, all of which have had major impacts on the coal industry. On the world stage, oil embargos brought about a sharp increase in energy prices opening up markets especially for Australian coal. The 1980s was a period of great transformation, especially in the US coal industry where the

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highly unionized labor force lost their foothold, largely due to sizable nonunion mines opening in the Powder River Basin with repercussions that spread throughout the largest coal companies. Longwall mining methods also became more widely employed. Globally, Europe was eclipsed by Asia as the leading coal producer, and Australia surpassed the United States as the leading coal exporter. The robust economy of the 1990s led to tremendous expansion in the coal industry with annual coal production reaching and exceeding 1 billion tons in the United States and Asia. In the United States, production from surface mines surged past production from underground mines as economies of scale were realized in the Powder River Basin. In the first decade of the 21st century, the talk of global warming throttled the previous decade's expansion in the United States leading to a wave of mergers and acquisitions. Globally, however, economic growth in China led to a surge in Chinese coal production and imports with much of the latter coming from Indonesia. This decade ended with a global recession sparked by the US financial crisis that spilled into the next decade. In the current decade, coal markets have been significantly impacted by the availability of cheap natural gas and tremendous growth in renewable energy sources. What the future holds remains to be seen, but it is clear that the industrialized world requires energy and coal is still one of the most efficient and economical energy sources.

During the last 50 years, one can see in the brief summary just provided that at times, local trends ran counter to national or global trends. For example, while US coal production grew to surpass the billion-ton threshold in the 1990s, in the author's home state of Illinois, coal production fell by one-half due to clean air legislation. Twenty years later, while US coal production was declining significantly, Illinois Basin coal producers were experiencing robust growth including a surge in coal exports.

Coal industry employment has experienced similar booms and busts although employment trends do not always track with production trends due to increased mechanization and automation. For more than a decade, there have been ongoing discussions regarding an impending skilled labor shortage in the coal-mining industry. This shortage will affect all sectors of the industry including production workers and managers, engineers and geologists, mine inspectors and regulators, and even professors and researchers.

15.2 Why research is important

Lifestyle or standard of living is based on wealth creation. In the book *Unlimited Wealth* [1], one economist defines wealth as.

$$W = P \times T^n$$

where W is wealth, P is physical resources, and T is technology. Technology has an exponential multiplier effect denoted by n. Assuming that more wealth is better than less, improved lifestyle is a function of acquiring more physical resources or developing newer, better technology.

Applying this theory to advancing productive, safe, and responsible coal mining, it is assumed that physical coal resources are finite. Companies compete with each other for ownership of those resources, but for the industry as a whole, they are a constant. That leaves technology as the most important determinant of industry growth or advancement. In fact, technology can define or redefine the physical resource component of wealth. The reason why research is so important is that research is what drives technology development.

There are many ideas about what constitutes research. For the purposes of this discussion, research is considered to be the systematic investigation of theories and concepts OR the creative exploration of ideas that, when engaged in, increases the body of knowledge (what we know) concerning a particular subject matter. Research not only formulates new knowledge, but it also develops ways to apply that knowledge. Hence, research and development are often used together. Without research, the technology needed to advance the coal-mining industry would not exist. And without the development of new or improved technology, it is difficult, if not impossible, to advance productive, safe, and responsible coal mining.

15.3 Major accomplishments in coal research

15.3.1 Coal mining has become more productive

Mechanization has been at the center of research and development efforts to improve mine productivity. Other than pack animals employed to haul loaded coal out of mines, coal mining prior to early in the 20th century consisted entirely of backbreaking manual labor. Inventors such as Henry Harnischfeger, Joseph Joy, and Joannes Montabert led the way in pursuing technology to make coal mining more productive. Harnischfeger built large earthmoving machines such as shovels and draglines used in surface mines. Joy's inventions included the shuttle car, the continuous miner, and the longwall shearer used in underground mines. Montabert developed pneumatic drilling and rock-breaking equipment used in both surface and underground mines. Using these machines, one miner could do the work of 100 miners without them.

Rather than conducting a variety of laboratory experiments, research for these men and the companies they founded consisted of fabricating prototypes based on ideas they had for different types of equipment. Prototype machines may have been checked out for operability, but the only place for real, true testing was in the mine. This was a tedious and expensive process; however, when they came up with something that worked, the benefits in terms of increased productivity were enormous. Development of faster, stronger, easier-to-operate equipment is ongoing within research and development groups of equipment manufacturing companies.

The mining process consists of repetitiously performing a number of tasks. Each task is performed in a certain sequence or cycle, the collection of which makes up the overall production cycle. In some cases, mechanization led to one machine accomplishing several tasks simultaneously, thereby eliminating one or more tasks or cycles. For example, the continuous miner performs cutting and loading functions previously requiring three machines (cutter, drill, and loader) to accomplish. Other machines completely alter the production process eliminating certain tasks and introducing new cycles in their place. For example, longwall mining eliminates the roof-bolting task and replaces it with moving and setting shields.

Mine engineers have been interested in using computers to build models that simulate these mining sequences ever since the computer was introduced to the industry in the 1960s [2]. The primary reason for this interest is that computer models imitate real-life systems in such a way that operational scenarios can be tested and evaluated without the need for actual field experimentation, which is always a difficulty given the challenging variability of the mine environment. Contemporary productivity research takes advantage of the advances in computing technology to construct simulation models of various mining processes. Using these models, each step of the mining process can be analyzed to determine optimal operating techniques and sequences. Such models are at the forefront of current efforts to develop autonomous haulage systems for both surface and underground mines.

15.3.2 Coal mining has become safer

According to one industry expert, there are a number of factors contributing to a safer mining industry [3] including removing the mine worker away from the working face and utilizing automation to reduce or eliminate human-machine interactions. Once the Bureau of Mines had achieved a measure of success in reducing the frequency and scope of mine explosions, attention turned to falls of ground, which took the place of explosions as the leading cause of fatalities in coal mines. Research into ground control methods and systems led to scientifically designed roof control systems such as truss bolting and pumpable cribs. Additionally, researchers at the National Institute for Occupational Safety and Health (NIOSH) developed a suite of ground control software packages that include the Coal Mine Roof Rating (CMRR), Analysis of Longwall Pillar Stability (ALPS), and Analysis of Roof Bolt Systems (ARBS). These tools are easy to understand and use and have gained wide acceptance throughout the underground coal-mining industry.

Another factor contributing to a safer mining industry is improved education and training for miners. Regulations limiting exposure to dust have been in place in the United States for a half century, but methods used for monitoring exposure took days and sometimes weeks to produce results. The personal dust monitor (PDM), a state-of-the-art monitoring device developed by NIOSH working in concert with a private company that specialized in scientific monitoring equipment, now provides dust exposure data in real time allowing the wearer to adjust positions or behaviors to reduce exposure before it can reach hazardous levels.

A third factor contributing to a safer mining industry is the development and utilization of health and safety management systems, techniques, and concepts. Some of these systems are very broad-based. For example, CORESafety is a self-proclaimed "industry-wide partnership" that focuses on developing leadership and building safety cultures throughout all of mining [4]. Other systems have a much narrower focus that deals with only one safety issue. Research on developing advanced spray systems for coal mine dust control is an example of the latter [5, 6]. Between these broad and limited approaches are systems such as the remote monitoring of atmospheric conditions, which utilize modern communication technology to track concentrations of dangerous gases and provide alerts when hazardous levels are reached. The value of these systems has been proved on many occasions, and as a result, they have become the norm in today's coal-mining industry.

15.3.3 Coal mining has become more responsible

As the modern world moved into the age of environmentalism, the coal industry seized the opportunity to clean up their image. Most reclamation research deals with not only restoring disturbed lands, but also improving them. Reclamation efforts have dealt with strata restoration, reclaiming to original contour, restoring flora and fauna habitat, controlling mine and storm runoff, and revitalizing wet lands. Although the thought may seem counterintuitive, the casual observer of the mining industry's reclamation efforts may come to the conclusion that they embody true environmentalism.

A subset of the overall reclamation topic in mining is the waste disposal issue. Only in rare instances does Mother Nature provide minerals in pure enough form that processing is not required to separate waste material from the desired product. Recognizing that the past practice of just tossing it aside is no longer acceptable, the mining industry has examined multiple strategies for effective and responsible waste disposal. For the most part, these strategies tend to be patterned after waste-handling practices in other industries that typically focus on reducing waste generation by utilizing more selective mining methods, finding beneficial ways to use the waste material, or recycling waste material to recover secondary minerals.

15.3.4 Coal utilization has become cleaner

Perhaps on no other issue has the coal industry been saddled with an undeserved bad reputation than on the matter of pollution or emissions from coal utilization. As human civilization became increasingly environmentally conscious, the coal industry was blamed for causing acid rain and responded by installing flue gas desulfurization scrubbers. Then, the coal industry dealt with urban smog, an issue that really belonged more to the transportation industry, by installing low-NO_x burners with selective catalytic reduction. The problem of particulate matter was addressed with electrostatic precipitators and baghouses. The amazing thing is that all of these problems were addressed with great success in ways other than reducing the amount of coal being burned. In fact, coal utilization soared during the same period of time that emission reductions were declining dramatically (see Fig. 15.1). Thus, thanks to the ingenuity of the coal industry, not only did utilization become cleaner, but access to electricity expanded substantially (see Fig. 15.2).

The biggest issue of them all when it comes to emissions is carbon dioxide (CO_2), a greenhouse gas tied to climate change and global warming. This is a tough one for coal because coal is made up mostly of carbon. Sulfur, ash, mercury, and other hazardous elements are impurities in coal, but even if pure 100% carbon coal could be obtained,

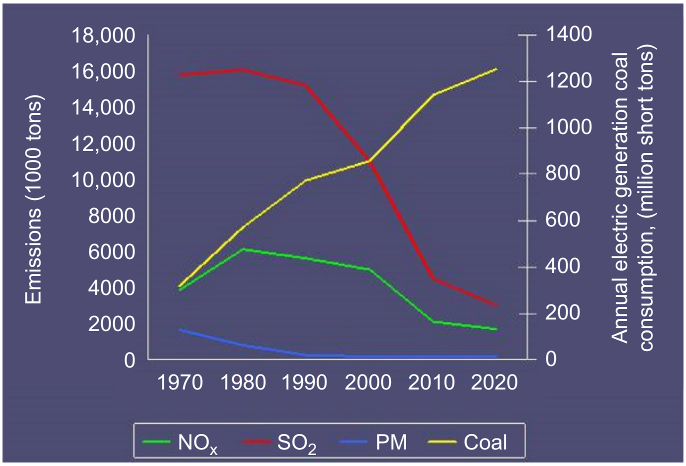

Fig. 15.1 Trends in emissions from coal utilization versus coal consumption.

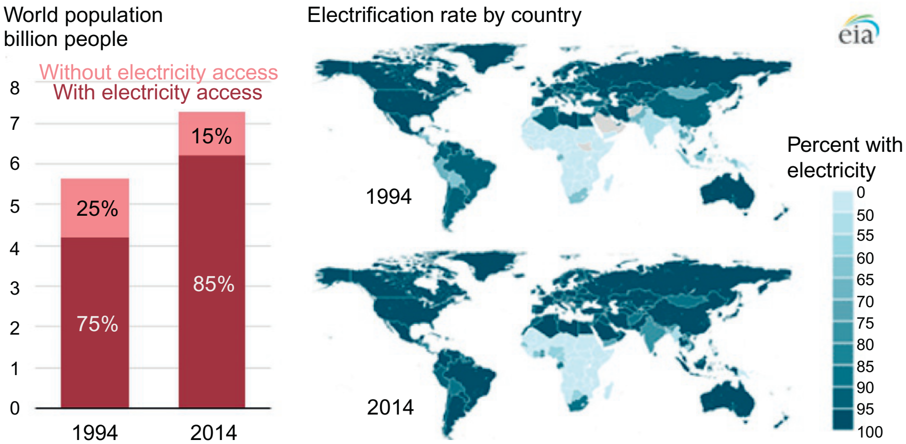

Fig. 15.2 Changes in the availability of electricity from 1994 to 2014 [7].

burning it would produce CO_2. Here again, the industry has not shied away, but faced the problem head on, leading efforts to develop technologies for carbon capture and sequestration (CCS), enhanced oil recovery (EOR), and coal bed methane (CBM) production. While the most viable solution is not yet proven, the author believes that given the chance, the coal industry will find it. For that to happen, politically biased policies that would divert critically important research funding away from technology development research must be avoided. Those policies seek to equitably divide a fixed amount of wealth and ignore the multiplying effect of the technology component of Pilzer's equation.

15.4 Research logistics and funding

15.4.1 Academic research

When the author decided to pursue a career in mining engineering, he investigated how many schools offered such a degree and was surprised at both the number of available options as well as some of the schools that had active research programs, such as Columbia University and University of California at Berkley in the United States. Unfortunately, many of these programs have been eliminated or are no longer active. Today, there are fewer than 15 US universities that offer a degree in mining engineering, and only about half of those have research programs with a coal-related emphasis.

Also, on the decline is the number of university faculty with experience in coal-related research. Whether the reason is retirement, very attractive offers from the private mining sector, or highly lucrative opportunities with other industries all together, the "brain drain" in mining engineering is affecting not only the quantity and quality of research being done, but also enrollment numbers. Recognizing the criticality of this situation, organizations like the Society for Mining, Metallurgy, and Exploration (SME) have stepped up and begun to offer funding to graduate students interested in pursuing an academic career in a mining-related field. Additionally, research grants are being awarded to junior faculty to incentivize their pursuit of tenure.

15.4.2 Government-funded research

For more than 80 years, the US Bureau of Mines was the hotbed for coal mining and processing research, particularly on health and safety topics. When budget cuts eliminated the bureau, politicians from coal states lobbied heavily to preserve at least some research facilities and personnel. They were successful to a degree in keeping the Pittsburgh (PA, USA) and Spokane (WA, USA) research groups alive. These groups became part of NIOSH in the US Department of Health and Human Services' Center for Disease Control and still include "mining research" in their official names. The principal objectives of the NIOSH mining research program are to "eliminate mining fatalities, injuries, and illnesses through relevant research and impactful solutions" [8]. NIOSH research focuses on reducing overexposures to hazardous dust and diesel contaminants, noise, unsafe ground, and work environments that are high-risk when it comes to causing musculoskeletal injuries.

Another major sponsor of coal-related research has been the US Department of Energy (DOE). Born out of the 1970s oil embargos, the DOE has focused on energy development and regulation. Of course, this covers more than coal, but coal-fired electricity generation has been the leading source of energy both domestically and internationally since the DOE's origin, so it is natural that the DOE would be involved in coal research. Like NIOSH, the DOE coal research program includes a number of national laboratories where research is conducted; however, the DOE program also provides significant financial support to external organizations. Research emphasized with this external funding has evolved as "the Department has sought to transform the nation's energy systems and secure leadership in clean energy technologies, pursue world-class science and engineering as a cornerstone of economic prosperity, and enhance nuclear security through defense, nonproliferation, and environmental efforts" [9].

At other levels of government, many states and even some local governments have recognized the need to provide financial support for technology development. The author is most intimately familiar with the Illinois Clean Coal Institute (ICCI), a state agency whose budget was supplied by a small "energy fuel tax" levied on every utility customer in the state, regardless of whether their power came from a coal-fired electric generator or a nuclear power plant. The ICCI was strictly a funding agency that provided financial support to outside contractors to conduct research designed to "promote the development and application of new and/or improved technologies that contribute to the economic and environmentally sound use of Illinois coal" [10]. Sadly, budget cuts brought an end to this program and all others like it.

15.4.3 Private industry research

At the beginning of the author's career in the private sector, the company he worked for had a corporate engineering group that focused on research and development related to improving efficiencies at the mine. Several projects were done as joint ventures between corporate and local mine engineers. Others were collaborations with local universities where expertise was readily available. It was not unusual at that time for coal-mining companies, especially the larger ones, to have their own research and development programs. Unfortunately, in the coal-mining industry, these corporate programs have all but disappeared as engineering teams have been required to focus their time and attention on complying with regulations requiring permits for every step of the mining process from exploration through mining and processing to reclamation and waste disposal. Independent contractors and consultants have stepped in to partially fill the void created as corporate R&D groups folded, but even so, they have become caught up in producing the massive volume of paperwork required to obtain permits.

One bright spot in coal research is the Australian Coal Association Research Program (ACARP), established in 1992. ACARP's research covers a wide range of topics from production and utilization to health, safety, and the environment [11]. The research program uses a very effective model that matches the financial resources of coal producers (obtained through a severance tax) with the technical expertise available at academic institutions, all with oversight from governmental policy makers, a model that is self-proclaimed as "the most successful coal research program in the world" [12].

15.4.4 Proactive versus reactive research

The ACARP model is a shining example of industry, academia, and government forming a synergistic partnership. One of the keys to their success is that coal producers are not viewed as just program financiers. Rather, executives and engineers from coal-producing companies play a vital role in determining the research agenda, thereby keeping the program relevant and proactive. When a coal company executive is seated at the table determining where his or her company's money is going to be spent, they will seek out the highest quality project that can be completed at the most economical cost. On the flip side, when research programs are financed primarily by taxpayer dollars, they become reactionary programs, negotiated by pork-barrel politicians trying to assuage the clamoring of the general populace who have strong opinions, but no skin in the game.

15.5 Coal's challenge: Mission impossible or a bright future

In recent years, political rhetoric has spawned the term "war on coal" to describe the impact environmental, health, and safety regulations have had on the coal industry. Taking the position that coal is under attack from those desiring a cleaner environment, healthier air and water, and safer working conditions presupposes that those ideals are not as desirable as the counter positions of preserving jobs and keeping electricity prices low. Such arguments will always come out on the losing end in this battle because the battle is really a moral dilemma. If the coal industry is to succeed in today's society, it has to accept and endorse all moral efforts that are beneficial to human life [13].

The high ground in any moral battle is won by solving problems, and there are plenty of opportunities for that in coal mining. In solving problems, the correct and/or best course of action will always be hard. Those who solve problems do so because they are willing to do hard things. Those who solve problems do hard things not because they have to, but because they are the right thing to do. Solving problems requires action rather than reaction. Solving problems requires passion, loyalty, determination, and discipline.

Coal mining is a very cyclic process, and it can be easy to get stuck in the rut of a routine that breeds complacency. A coal miner satisfied with the status quo is like a comfortable frog in a pot of water that has just been put on the stove. There is no doubt that the heat is on for the coal miner of the future, but that heat is opening up opportunities for those willing to seize them. Those opportunities are and will be available to the next generation of coal miners, reclamation specialists, safety engineers, mine inspectors, computer modelers, laboratory scientists, researchers, professors, and all others who continue to work in the coal industry, because it is going to be around for many years to come.

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