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A Dynamic Programming Approach to Identifying Optimal Mining Sequences for Continuous Miner Coal Production Systems

Joseph Christian Hirschi
Southern Illinois University Carbondale, jhirschi77@hotmail.com

A DYNAMIC PROGRAMMING APPROACH TO IDENTIFYING OPTIMAL MINING SEQUENCES FOR CONTINUOUS MINER COAL PRODUCTION SYSTEMS

by

Joseph Christian Hirschi

M.S. in Mining Engineering, M.B.A.

University of Utah, 1985

B.S. in Mining Engineering

University of Utah, 1983

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Approved by:

Dr. Yoginder P. Chugh, Chair

Dr. Satya Harpalani

Dr. Henri U. Schurz

Dr. Stanley C. Suboleski

Dr. Gregory P. White

Graduate School

Southern Illinois University Carbondale

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MAJOR PROFESSOR: Dr. Yoginder P. Chugh

Underground mines are the source of 33% of US coal production and 60% of worldwide coal production. Room-and-pillar mining with continuous miners has been the most common production system used in these mines since the 1960s. The introduction of continuous miners mechanized the underground coal mining industry triggering a period of sustained growth in mine productivity; however, productivity peaked at the turn of the century and has been in decline for a decade. Research on productivity in underground coal mines began at Southern Illinois University Carbondale in 2000 and led to development of a deterministic spreadsheet model for evaluating continuous miner production systems. As with other production models, this model uses a heuristic approach to define the fundamental input parameter known as a cut sequence. This dissertation presents a dynamic programming algorithm to supplant that trial-and-error practice of determining and evaluating room-and-pillar mining sequences. Dynamic programming has been used in mining to optimize multi-stage processes where production parameters are stage-specific; however, this is the inaugural attempt at considering parameters that are specific to paths between stages. The objective of the algorithm is to maximize continuous miner utilization for true production when coal is actually being loaded into haulage units. This is accomplished with an optimal value function designed to minimize cut-cycle time. In addition to loading time, cut-cycle time also includes change-out and place change times. The reasonableness of the methodology was validated by modeling an actual mining sequence and comparing results with time study and production report data collected from a cooperating mine over a two-week time period in which more than 300 cuts were mined. The validation effort also inspired some fine-tuning adjustments to the algorithm. In a case study application of the dynamic programming algorithm, a seven-day "optimal mining sequence" was identified for three crosscuts of advance on an eleven-entry super-section developing a main entry system for a new mine in southern Illinois. Productivity improvements attributable to the optimal sequence were marginal but the case study application reconfirmed the reasonableness of the methodology.

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CHAPTER 1

INTRODUCTION

1.1 Background

Importance of underground coal mines. Since the early days of the Industrial Revolution two and one-half centuries ago, coal has been a leading source of world energy production. In the United States (US), according to statistics compiled by the Energy Information Administration (EIA), coal was the leading source of energy production from the Civil War through World War II, a position that it regained following two oil crises in the 1970s, and held until 2010 when natural gas claimed the top spot (EIA, 2011a). Even then, in that year coal accounted for one-fifth of total energy consumption in the US with the electric power sector responsible for 94% of coal consumption (Freme, 2010). Coal's staying power at the forefront of the nation's energy mix for more than a century is the result of low-cost price stability, driven in part by a major shift in production from numerous underground mines in relatively thin coal seams east of the Mississippi River to a few surface mines in the massively thick coal seams of the Powder River Basin (PRB) in Wyoming.

In the US, despite PRB coal's economies-of-scale and other negative influences, such as political efforts to regulate carbon emissions and detrimental impacts of mine disasters, underground coal mining still contributes 31% of coal production, and in Illinois that contribution is 85% (EIA, 2011b). Worldwide, underground mining accounts for 67% of coal production (DOE, 2009). Although underground mines typically produce bituminous coal, which has a higher energy value than sub-bituminous PRB coal, the real resilience of underground coal mining is due to significant productivity gains made possible in the last half century by an "*extensive application of complex continuous systems possessing remarkably high production rates*" (Pavlovic, 1989). The two obvious "complex continuous systems" are the continuous miner and the longwall.

Continuous miners and longwalls are the centerpiece of mechanized mining systems. As shown in Figure 1.1, this mechanization of the industry increased US underground coal mine productivity (in tons per man-hour) from 0.68 in 1949 to 4.15 in 2000 (EIA, 2011a). Of present concern is the steady decline over the past decade, which may be attributable in part to increasing regulatory requirements and depleting coal reserves. The MINER Act of 2006 may be contributing to a downturn similar to that of the 1970s caused by Federal Mine Safety and Health Acts of 1969 and 1977. US coal reserves remain abundant but reserves with the best conditions continue to deplete forcing underground coal mining to deeper and sometimes thinner coal seams, which present more difficult productivity challenges.

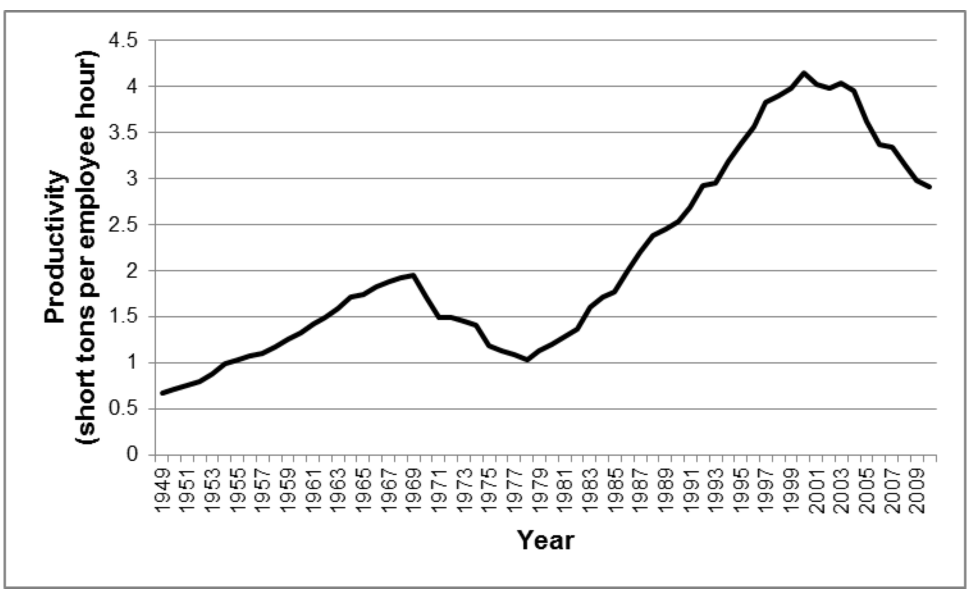

Figure 1.1. Underground coal mining productivity in the US.

Brief description of underground coal mining. Work in an underground coal mine consists of two functions, both of which are essential (Stefanko, 1983). One is the actual production of coal. Underground coal mines consist of one or more units or sections where coal is produced at a location called the "face." Each section has a specific production method that may be either longwall, which has one long face, or room-and-pillar with multiple narrow faces as shown in Figure 1.2, which are often collectively called the face. All other work functions, which could include installation of infrastructure such as conveyor belt and power supply systems, maintenance of production equipment and safety devices, and delivery of materials and supplies, are classified as auxiliary operations. They are no less essential than production, but they take place "outby" the face and do not contribute directly to the output of coal. This study focuses exclusively on production functions of a room-and-pillar section.

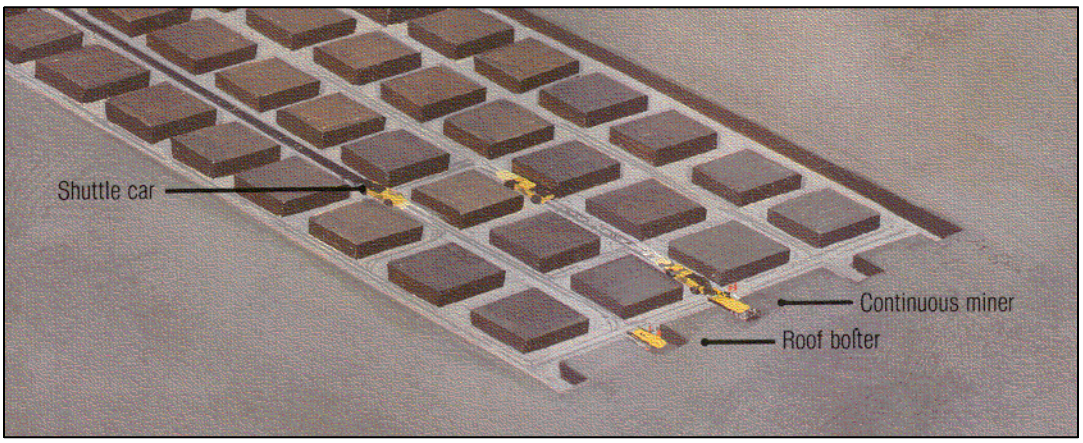

Figure 1.2. Cut-away showing face area layout of room-and-pillar mining section (scan of graphic in promotional literature from Kerr-McGee Coal Corporation's Galatia Mine, 1985).

Room-and-pillar mining is so named because pillars of coal are left in place to support rock above openings (rooms) that are created by the mining equipment. Room-and-pillar sections typically have as few as three or as many as fifteen parallel rooms called entries that are connected at regular intervals by additional rooms called crosscuts. Crosscuts are usually oriented perpendicular to entries unless adverse horizontal stresses create unstable conditions or continuous haulage systems are used, in which case, crosscuts may be oriented diagonally in a chevron or herringbone pattern. Entry and crosscut spacing combined with room width determine pillar size. Rooms vary between 16 and 22 feet in width with 20 feet being the most common. Entry and crosscut spacing, room width, and cut depth constitute the mine geometry and are functions of geological parameters such as depth of coal seam, equipment characteristics such as haulage unit size, and production requirements.

Entries and crosscuts are mined in small segments called "cuts" that vary in depth from as little as five feet up to as much as 40 feet. Figure 1.3 shows the plan view of a 7-entry room-and-pillar mining section. Each numbered block represents an individual cut and ordered numbers identify a mining or cut sequence. All 26 cuts make up a cut-cycle for one crosscut of advance. Once two or more of these cycles are completed, mine infrastructure such as the section's conveyor belt and power center are moved forward to keep them in close proximity to the face.

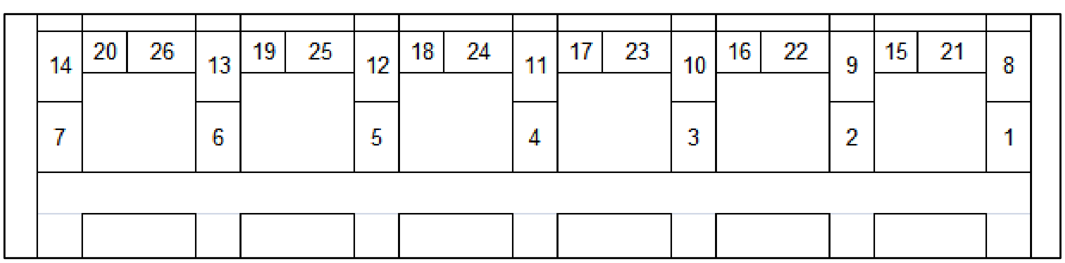

Figure 1.3. Plan view of room-and-pillar section showing hypothetical cut sequence for a cut-cycle that achieves one crosscut of advance.

In the 1970s, the super-section concept was developed with two production machines operating in the same section (Suboleski, 1975). Initial super-sections had a single crew with two machine operators but only one machine produced coal at any given time. At the end of a cut, one of the machine operators commenced a new cut with the machine that had not been producing coal while the second operator moved the machine that had been producing coal to the next face. This was called a walk-between or single-crew super-section. More recently, ventilation plans have changed allowing intake air to travel up the center entries of a section to the face area where it splits providing fresh air to both sides of the section (fish-tail mine ventilation) enabling both machines to produce coal simultaneously. This is called a split-ventilation or dual-crew super-section. The effect has been to combine two smaller sections into one large section. Super-sections offered several productivity gains to the mine operator. Initially, the primary benefits were a huge reduction in place change time and fewer production stoppages due to breakdowns because the second machine functioned as a spare when one machine had problems. Other benefits included a reduction in manpower (the section could be run with one foreman, one utility man, and one repairman) and a reduction in required capital (only one feeder breaker and one section conveyor were needed instead of two). While the number of super-sections now in operation is not specifically reported, an informal review of large (>100,000 tons annual production) eastern US underground mines indicated that at least one-third employed some form of super-section (Suboleski and Donovan, 2000). All active room-and-pillar mines in Illinois employ super-sections.

Continuous miners – the workhorse of the underground coal mine. A continuous miner (CM) is the primary piece of production equipment used in room-and-pillar coal mines worldwide. In the US, the only room-and-pillar mines not utilizing CMs are conventional mines employing drilling and blasting techniques, of which there are very few still in operation. In addition to one or more CMs, a CM production system includes either batch or continuous haulage equipment that transports excavated coal from the face to a network of conveyors that remove it from the mine. Another component of a CM production system is the roof bolter that installs supports into the rock above the coal that was removed allowing miners to work in the excavated area.

Given the two production methods identified previously, a mine may consist totally of CMs operating in room-and-pillar sections, but it cannot operate exclusively with longwall sections because room-and-pillar gate entries and set-up rooms must be mined by CMs before longwall equipment can be deployed. Thus, in a typical longwall mine, 20% of production comes from CMs in room-and-pillar sections developing longwall gate entries and set-up rooms (EIA, 2011b). Consequently, as indicated in Table 1.1, CMs account for almost 60% of total underground coal production in the US.

Table 1.1. US underground coal mine production in 2010 by mining method (EIA, 2011b).

Type of Mine	Extraction Method	Production (MM tons)	% of Total
Room-and-Pillar	Continuous Miner	163	49
	Conventional	5	1
Longwall	Continuous Miner	34	10
	Longwall	135	40
Total		337	100

CM systems have the advantage over longwall systems in mine planning flexibility by being able to size up or down. They are compatible with all types of reserve configurations, whereas longwalls require large, contiguous blocks of coal. At mining depths greater than 1,000 feet, longwalls have an advantage due to geologic constraints requiring larger pillars, which generally cause room-and-pillar mining to become less efficient (Darmstadter, 1997). For this reason, longwall mining is more prevalent in deep, thick-seam, mines located in the western states of Colorado, Montana, New Mexico, Utah, and Wyoming where 13 of the 18 operating underground coal mines are longwall mines (EIA, 2011b; Fiscor, 2011). While flexibility and efficiency are important, the chief factor enabling CM systems to be the primary means of production at most of the nation's underground coal mines is the comparatively low capital investment required to operate them (Thomas, 2002). Larger producers generally have more capital to invest explaining why half of the active longwall faces (22 out of 44) in the US are operated by the four largest coal producers – Alpha Natural Resources, Arch Coal, CONSOL Energy, and Peabody Energy/Patriot Coal (EIA, 2011b; Fiscor, 2011).

There is no doubt that longwall systems have played a major role in the underground coal mine productivity gains cited earlier. In the decade from 1983 to 1993, average longwall productivity rose from 2% lower to 19% higher than average CM productivity, all while CM productivity was increasing rapidly (EIA, 1995); however, of 497 underground coal mines in the US in 2010, only 44 were longwall mines (EIA, 2011b; Fiscor, 2011). Thus, CM systems remain well established as the backbone of the industry and, despite the fact that tons per miner may be higher for longwalls than for CMs, most mine operators still choose CMs for their production system.

1.2 Problem Definition

The name "continuous" describes the ideal for the CM production system and developing continuous systems has been the focus of considerable research over the past 50 years. Accomplishments include remote-controlled machines, continuous haulage, miner-bolters, dust scrubbers, extended cuts, and high-voltage equipment, to name a few; however, studies continue to suggest that despite all of these progressive developments, the potential still exists to nearly double CM productivity (Davis, 1980; Chugh, 2003). Without diligently focusing on keeping the CM at the face cutting and loading coal, productivity can be needlessly sacrificed. Obviously, over the course of a normal shift, there are regular times when the CM does not produce coal, such as when equipment maintenance is performed, when it is moved from a completed cut to a new cut, or when it is waiting on the roof bolter; however, delays that occur while the CM is at the face ready to load coal have the most significant impact on CM utilization (Davis, 1980; King and Suboleski, 1991; Hirschi *et al.*, 2004). These delays are mostly from changing haulage units at the face or waiting for them to complete their haulage cycle and can cause utilization of production equipment to consistently fall below 50% and mine laborers to be involved in nonproductive work as much as 40% of the time (Douglas, 1980; Hanslovan and Visovsky, 1984).

The critical importance of miner productivity stems from the fact that, even with increasing levels of mechanization, labor costs account for almost half of total production costs (Hanslovan and Visovsky, 1984; Chugh, 2001a; Thomas, 2002; Moharana, 2004). Those tasks that must be completed by human and machine interaction to produce coal are listed in Table 1.2 along with how many times they are repeated in a typical shift. These repetitive tasks are well-suited to industrial engineering analysis using modeling and simulation. Geologic modeling tools exist but integrating them with production modeling is extremely difficult. Thus, miners depend on lessons learned from previous experiences in handling these challenges. The resulting wide variety of experience levels hampers efforts to formulate and define standard operating procedures (SOPs) for repetitious tasks and for critical decision making. This dissertation addresses the need for developing such an SOP for planning and executing optimal cut sequences.

Table 1.2. Critical path mining tasks and repetitions per shift typical of batch haulage CM systems.

Mining Segment	Repetitions/shift
Travel to/from work site (face)	2
Tram CM from cut to cut	10
Clean up after cut	10
Move or handle CM cable	25
Reposition CM during cut	50
Load haulage unit	200
Haulage unit change-out	200
Dump haulage unit	200

In any particular room-and-pillar section, a specified cut sequence can be repeated almost daily making it desirable to specify an optimal sequence. Engineering tools such as computer modeling exist for analytically determining a mining sequence, but the time required to make use of them is often more than the busy mine engineer has, especially if that modeling involves a trial-and-error approach. Furthermore, the mine foreman, who directs underground operations including the sequence of cuts, has limited access to computers and no time for trial-and-error approaches. Consequently, one of them typically designates a sequence based on previous mining experience, which becomes the standard procedure for the mine. This practice compromises any effort to achieve ongoing process improvement because the cut sequence upon which such effort is based has no scientific criteria for evaluation. Computer modeling does enable mine engineers to evaluate operational scenarios on paper instead of in the mine, but to be relevant, they require certain input data and the cut sequence is one of the foundational inputs.

To illustrate the scope and importance of defining an optimum cut sequence by computer modeling, consider Figure 1.4, which shows only the left side of a super-section where 38 cuts are to be mined in a two-crosscut cut-cycle that creates pillars with non-uniform geometries. Bold red numbers define entry spacing, italicized blue numbers define cut lengths, and black letters and numbers identify cuts. The number of permutations for sequencing these 38 cuts is 38! or 5.23×10^{44}. Obviously, the vast majority of these sequences are not feasible. For example, any sequence that places E1.4 before E1.1 is not possible since E1.1, E1.2, and E1.3 must all be mined before the CM has access to E1.4.

If the problem is reduced to just the first cut in each entry, there are still 6! or 720 different sequences. Using a simple computer production model to determine productivity (measured in tons per minute) for each one of those 720 sequences yields the range of productivities shown by the histogram in Figure 1.5, which clearly identifies a very limited number of optimal sequences. Only two of the 720 possible sequences (0.28%) are optimum (i.e., tons per minute is maximized) and only eight (1.11%) are optimal as defined by the top 25% of the productivity range. Unfortunately, most cut sequence options are sub-optimal, thereby increasing the chance that the mine foreman’s experience-based sequence selection will be as well.

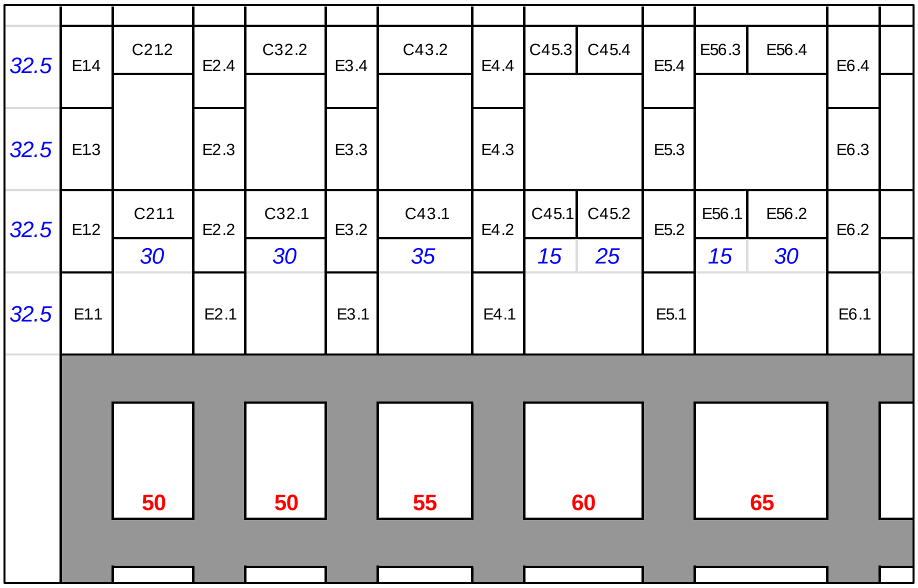

Figure 1.4. Two-crosscut cut-cycle for the left-side CM of a super-section using alternate mining geometries.

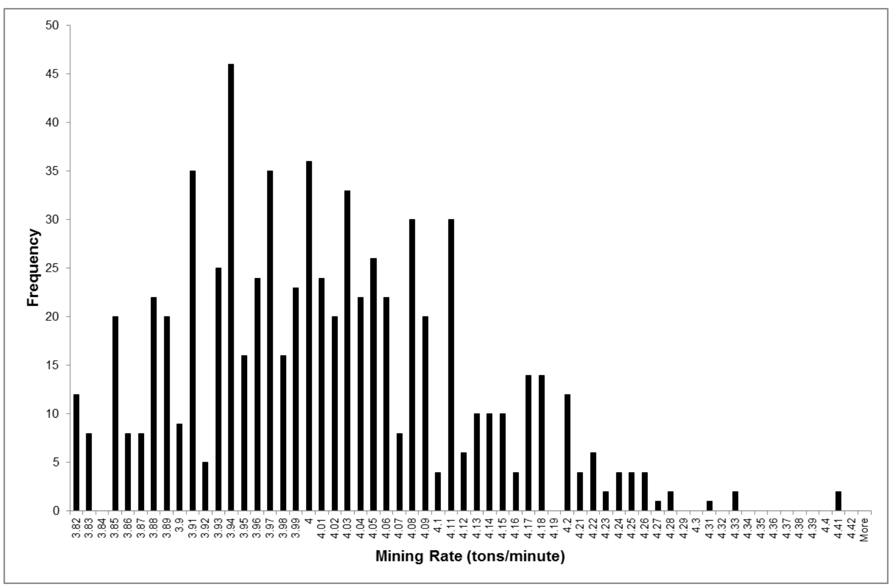

Figure 1.5. Histogram of modeled productivity for each possible cut sequence if only cuts E1.1, E2.1, E3.1, E4.1, E5.1, and E6.1 in Figure 1.3 are mined.

Computer modeling and simulation can be used to identify an optimum mining sequence, but existing techniques, such as the Monte Carlo method, rely on repeated and often random computations in a time consuming trial-and-error process, even when begun with a sequence based on reputable mining experience. Multiple model iterations are required to find an optimal sequence or confirm that the starting sequence was indeed optimal. The computational aspect of the work is not a problem for today's powerful computers, but the time commitment required to supply all necessary input parameters and set up a model for repeated simulations is beyond the scope of time and computing resources available to most mine engineers. This work aims to replace the trial-and-error approach with an algorithm developed to specifically identify optimal cut sequences prior to commencing any production modeling for process improvement purposes.

1.3 Objectives

The overall objective of this dissertation is to utilize an optimization technique known as dynamic programming (DP) to develop an algorithm for determining an optimized mining sequence (OMS) for any type of room-and-pillar mining. The goal is that application of an identified OMS will result in demonstrated improvements in face productivity while simultaneously having a positive impact on the health and safety of underground coal miners.

DP is a technique used for optimizing multi-stage decision processes, which are processes that can be separated into a number of sequential steps called "stages" with one or more options for completion to choose from. It is based on Bellman's principle of optimality which states that "*an optimal policy has the property that, regardless of the decisions taken to enter a particular state in a particular stage, the remaining decisions must constitute an optimal policy for leaving that state*" (Bellman, 1957; Bronson and Naadimuthu, 1997). Each decision has a return usually described in terms of costs or benefits. The objective of DP analysis is to determine an optimal policy or sequence of decisions that results in the best total return. DP has been used in mining to optimize multi-stage decision processes where parameters are stage-specific; however, this is the inaugural effort to consider parameters that are specific to paths between stages.

Mining sequence optimization is just one component of the overall mine engineering process. Over the past decade, the author has been part of a Southern Illinois University (SIU) research team working on optimization concepts for underground coal mines in Illinois in an effort to slow or reverse the downward trend in productivity previously mentioned. This research attempts to bridge the gap back to earlier industrial engineering studies that are described in a journal paper (Hirschi, 2007), most of which is included in the literature review of Chapter 2. Initial efforts focused on production modeling leading to the development of the SIU/Suboleski Production (SSP) Model, which was used to provide productivity training at mines throughout the Illinois Basin (Hirschi *et al*., 2004; Kroeger, 2004; Moharana, 2004; Kroeger, 2006). Later efforts focused on finite element (FE) modeling of alternate mining geometries that were successfully demonstrated in the field (Chugh, 2006a; Chugh, 2007). Both efforts required developing cut sequences for computer modeling and underground application that had to be done using the conventional trial-and-error approach. The DP model presented in Chapter 3 completes the suite of optimization concepts developed by the team. These concepts are just as important for the new mine as they are for existing operations where performance evaluation should be an ongoing process.

1.4 Scope of Work

To accomplish the overall objective of this study, the following three tasks were completed:

1.4.1 Development of a DP Algorithm for Identifying Optimal Mining Sequences

The DP algorithm described in Chapter 3 is built on an optimal value function, which seeks to maximize or minimize a particular parameter, such as revenue or cost. In this coal mining study, minimizing cycle time was selected over maximizing production for the optimal value function as it relies on time study data, which is easier to obtain than production data for individual mining units. This is because most room-and-pillar coal mines are comprised of several mining sections with production from each section pooled to form total output from the mine, the level at which production is generally measured. At the section level, the production process is measured in terms of time increments for completing the various steps in the production process. The optimal value function developed in this study is comprised of production and place change time elements with industrial engineering studies providing data required to determine time values for these elements.

For the purposes of this study, minimizing cycle time and maximizing CM utilization are synonymous. CM utilization refers to the time the CM spends actually producing or loading coal. Moving the CM from cut to cut (the place change element) is a necessary part of the mining cycle during which the CM is definitely in use; however, since coal is not produced during place changing, it is not considered as CM utilization. Furthermore, the production element consists of loading and change-out components. As with the place change element, coal is not loaded during change-out time periods, so they are also not considered as CM utilization.

For a given mine plan, most cuts are of uniform volume causing the loading component to approach a constant value. Thus, seeking to minimize non-producing place change and change-out functions while maintaining a near constant loading function allows the DP algorithm to select cuts with minimum cycle times, which are analogous to cuts that achieve maximum CM utilization.

1.4.2 Validation of the DP Algorithm

As described in Chapter 4, the DP algorithm was validated by comparing cycle times generated by the model with cycle times measured in the mine of a cooperating company and cycle times reported by mine foremen for an actual mining sequence completed at the same mine over a two-week time period during which 331 cuts were made. Industrial engineering data collected for seven cuts on the last day of the two-week period was used to define various operational parameters within the algorithm.

It is important to note that in addition to validating the DP algorithm, this effort also fine-tuned the algorithm. Initial comparisons of algorithm outputs and shift report data revealed some significant differences in both the place change element and the production element. Discrepancies causing these differences were identified by revising algorithm parameters until algorithm outputs and time study data matched for the seven cuts that were studied. This led to one parameter in the change-out component of the production element of the DP algorithm being revised and a new parameter being added to the loading component of the production element of the DP algorithm. The algorithm presented in Chapter 3 is the product of the fine tuning validation process.

1.4.3 Application Case Study

The final task, described in Chapter 5, utilizes the developed DP model with its validated optimization algorithm to identify optimal mining sequences for a particular case study. The case chosen for application of the DP model is the same mining section that provided time study and shift report data utilized in the validation process. This allows for easier identification of optimal patterns as data on actual mining sequences are readily available and differences between them and optimal sequences predicted by the model can be clearly shown.

The application case study was completed in two parts. First, the DP model was used to identify one-day (12-18 cuts) optimal mining sequences for four different scenarios selected from actual settings experienced at the mine during the two-week study period. Second, the DP model was used to identify an optimal mining sequence for advancing the entire section by three crosscuts or enough to complete belt and power moves. These are compared with sequences that were actually mined to show productivity improvement potential from following optimal sequences.

1.5 Significant Contributions and a Limitation

Path-specific versus state-specific parameters. DP was created to provide mathematical solutions to any type of multi-stage decision process. These processes can be found in virtually every aspect of human life, such as factory production lines, warehouse inventories, hospital waiting rooms, classroom scheduling, and investment decisions, to name just a few. The mining industry is replete with such processes and DP has been used extensively, even for solving OMS problems. In every DP study reviewed by the author, defining parameters have been specific to a particular state or stage and in most cases those parameters are fixed or constant. For example, determining an OMS for an underground sub-level stoping operation depends on knowing the ore grade for each block to be mined, which is accomplished through an exploratory drilling program. That grade remains constant throughout the iterative DP process. Storage capacities and equipment/human resources availabilities are examples of other fixed or constant parameters used in DP applications to mining. The optimal value function for these DP models generally takes the big picture approach seeking to maximize revenues or minimize costs.

For the research topic under consideration of an OMS in an underground room-and-pillar coal mining operation, parameters such as cut volume and grade have little if any bearing on the problem. Those parameters that were considered critical are distances between cuts, of which there are a fixed amount that are easily defined; however, as the DP process progresses through each stage, the distance used to evaluate a particular stage at any point in time changes, generating a degree of complexity not found in other DP applications. To simplify the problem to the extent possible, rather than examining the bigger picture of maximizing production/revenue, an optimal value function was chosen for development that seeks to minimize the very basic variable of cut-cycle time, which is a function of parameters specific to the path between feasible states within a stage rather than to parameters defining the state itself. Therefore, to the best of author's knowledge, not only is this the first known attempt at using DP to analytically determine an OMS for a room-and-pillar coal mine, it is the inaugural attempt at DP modeling of path-specific parameters rather than state-specific parameters.

Bridging the gap. Industrial engineering concepts born out of the Industrial Revolution reached a level of maturity as the world endured two wars in the first half of the twentieth century. As industry emerged from the strain of these wars, mechanization was starting to take hold in the nation's coal mines providing a ready-made environment for the application of newly developed operations research techniques. Furthermore, the advent of the computer spurred growth in the use of applied statistics and higher mathematics to generate models that produced results quickly using exactly repeatable methods allowing the engineer to focus on the relevancy and accuracy of input data and analytical methods rather than on computational mechanics (Douglas et al, 1983).

Charting developments in production modeling for underground coal mines that is reviewed in Chapter 2 creates an interesting historical timeline that clearly identifies three distinct phases. In the twenty years following World War II, with academic institutions leading the way, a foundation was put in place for computer modeling and simulation in underground coal mines. With the foundation in place, there followed a second twenty-year period of intense activity in mine modeling and simulation, again driven primarily by academic institutions, but spreading into industry as college graduates built their careers. Then suddenly, just as computing capabilities exploded, computer modeling in the underground coal mining industry seemed to vanish. As the next twenty years passed, a gap emerged in the development and use of computer models for production process improvement engineering. While it is not hard to find a computer at today's coal mine, their use for modeling and simulation is not happening even though simulation programming languages are more readily available and powerful. Computer modeling languages have become so complex that developing models and simulators for underground coal mining and getting them to run is generally beyond the knowledge level of the experienced mine engineer, and recent college graduates with the latest computing skills and knowledge are more interested in production management as face bosses and mine managers where there is more money to be earned.

The widespread acceptance and use of the SSP Model experienced by the SIU research team highlights the demand for simplicity offered by a simple, deterministic, spreadsheet model that can be easily manipulated with quick responses. The one drawback of the SSP Model has been the time-consuming process of entering cut sequences, which had to be repeated any time a different sequence was to be evaluated for productivity improvement potential. Thus, one of the significant contributions of this work is to bridge the computing gap for the modern mine engineer by providing the foundational algorithm for creating an OMS model that integrates well with the SSP Model by reducing the effort required to input a cut sequence and by eliminating the heuristic nature of evaluating multiple sequences for optimality.

Limitation. Having identified the contribution of providing the basis for a simple OMS model, it must now be pointed out that this dissertation only goes as far as defining and validating the OMS model algorithm. For it to become an effective tool in the hands of a mine engineer, a person skilled in Excel® programming will have to build an OMS module for integration with the SSP Model. The author learned FORTRAN programming as an undergraduate student before beginning a career in mining that has lasted more than 25 years, during which time that learning was never utilized. Upon returning to the academic setting 12 years ago, the author found that advances in computer programming had progressed far beyond the limited capabilities of basic FORTRAN programming. Therefore, the author would leave it to some interested person with the necessary programming skills to incorporate the algorithm outlined in this dissertation into a spreadsheet module for integration with the SSP Model.

CHAPTER 2

LITERATURE REVIEW

2.1 Introduction

The literature review for this dissertation has two objectives. The first is to provide the reader who might not be familiar with continuous miner (CM) coal production systems, which are not very well known by those outside of the underground coal mining industry, with a basic understanding of such systems and the setting in which they operate. The second is to review research pertaining to both the development of these mining systems as well as efforts to improve their efficiencies.

The next three sections of this chapter focus on the mining process reviewing research efforts to develop and improve all of the principal components of CM coal production systems and the designing of underground coal mines in which they operate. This includes information on dust control research with specific emphasis on characterizing the environment in which CM production systems operate. The last two sections of this chapter focus on operations research studies related to underground coal mines reviewing research efforts to develop models for simulating CM production systems and optimization techniques for improving those systems.

2.2 Continuous Miner Coal Production Systems

Continuous miners. Joy Machine Company, the predecessor to today's Joy Global, a leading original equipment manufacturer (OEM) of underground coal mining equipment, shipped their first CM with a ripper head in 1948 (Harrold, 1980). Boring machines actually preceded CMs developing miles of entries in Illinois in the 1920s (Stefanko, 1983), but they lacked mobility and depended on mining conditions that would allow cutting in one entry for long distances without regard for roof conditions. Furthermore, the product was so fine that it was not very salable for domestic heating, the primary market of the day. As the market shifted to utilities burning pulverized coal, CMs gained wide acceptance easily supplanting conventional systems prevalent at the time by incorporating all of the system's production functions (undercutting, drilling, blasting, and loading) into one machine (Harrold, 1980).

In 1969, CM production surpassed conventional production for the first time (Keystone, 1981). Manufacturing of new CMs peaked a few years later in 1975 and by 1980, roughly 3,000 machines were producing approximately one-fourth of the total annual US coal production (Harrold, 1980). Then, as production shifted to large western surface mines and longwall mining increased in popularity, the number of CMs in use and their production both leveled off. In 1994, production from longwall mines exceeded that of room-and-pillar mines for the first time (NMA, 2003). OEMs bore the brunt of this industry transitioning and went through a difficult period of bankruptcies, consolidations, and mergers. Nevertheless, because of its versatility, the popularity of the CM has endured, and today, because of a combination of depleting longwall reserves, increasing stripping ratios, and a continually expanding energy market, CM production systems remain at the forefront of underground coal mining in the US.

Over the years, CM design has evolved from ripper heads to oscillating heads and finally to the milling head or hardhead common today (Stefanko, 1983). This head is a large metal drum laced with conical metal bits in a spiral winding supported by a boom at the front end of the CM (see Figure 2.1). As the drum turns, bits dig into the coal seam cutting loose various-sized pieces of coal, which fall to the ground where they are gathered by a large scoop called a pan onto a steel chain conveyor and carried to the rear of the CM where they are dumped into a haulage unit for transport away from the face.

Figure 2.1. Joy Global’s Model 14CM15 continuous miner.

Research in the 1970s (Campbell *et al*., 1978) produced the flooded-bed scrubber for capturing dust generated by the CM cutting drum (Campbell *et al*., 1983). This system transforms the cutter head boom into something like a ventilation hood above a kitchen stove with multiple inlets. The duct work on the boom connects to another duct on the side of the CM’s main chassis that runs the length of the chassis. At the mid-point of this duct is an inclined filter that is sprayed with water. Further to the rear of the CM, the duct contains a demister box that removes dust laden water droplets from the air. Near the tail end of the CM, an axial vane fan is mounted to the duct to create the suction needed to draw air into the ductwork. Dusty air around the cutter drum is sucked into the hood on the boom and pulled through the filter duct where dust is wetted and removed from the air by the filter panel and demister box before "scrubbed" air is discharged back into the mine atmosphere. A schematic of the CM scrubber is shown in Figure 2.2.

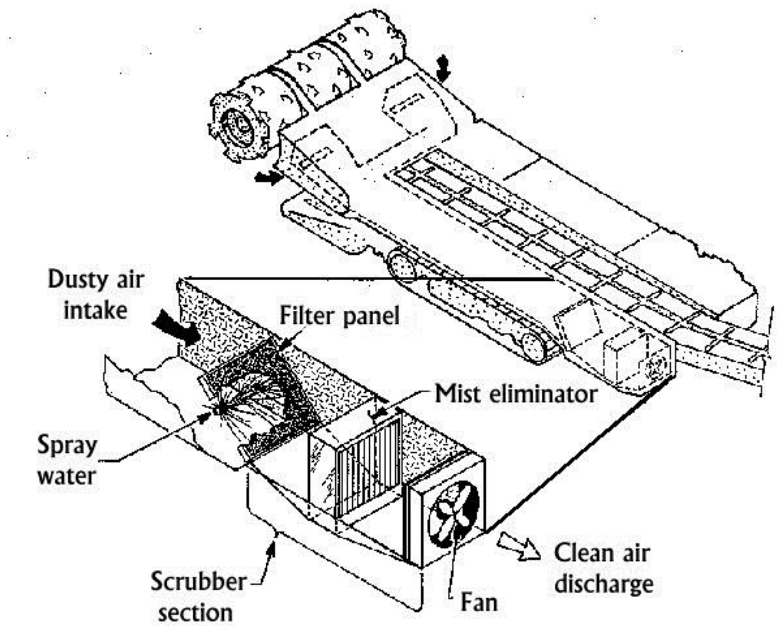

Figure 2.2. CM flooded-bed scrubber schematic (Thatavarthy, 2003).

Another development with CMs has been increased horsepower. Cutting motors typically experience the most severe duty cycles and the objective behind increasing horsepower has been to extend the life of these motors. A prototype high-voltage, high-profile CM went into production at an Illinois mine in 1997. The power supply was 2,300 volts for 966 horsepower as compared to 950 volts and 740 horsepower on a standard machine. It weighed 82 tons, 17 tons more than the standard machine and the cutting drum was seven inches bigger in diameter with 6-inch bit spacing as compared to 4.5 inches. This high-voltage machine utilized continuous haulage and productivity increased to the point that the haulage system had to be redesigned (Sprouls, 1998).

In 2001, the SIU research team, including the author, conducted an industrial engineering analysis of a prototype high-voltage, medium-profile CM, also operated in an Illinois mine. The high-voltage machine operated alongside a standard-voltage machine with the same profile in a super-section with haulage provided by battery ramcars. The study determined in a comparison of the two machines that the loading rate (measured in tons per minute) of the high-voltage machine was 30% greater, the tram speed (measured in feet per minute) of the high-voltage machine was 8% faster, and the increase in tons per car being loaded by the high-voltage machine was almost 9% or one ton per car (Moore, 2001). However, when comparing actual unit shift productivity, no difference was found between the two machines. The study concluded that the high-voltage miner can provide productivity increases only if the haulage system can transport coal away from the face fast enough (Chugh, 2001a).

Super-sections. As already pointed out in Chapter 1, the idea of a super-section with two CMs operating in the same production unit was developed nearly three decades ago (Suboleski, 1975) and the concept caught on until it is hard to find room-and-pillar operations that do not employ it. In Illinois, every room-and-pillar mine operates super-sections. Initial super-sections were single-crew (SCSS) or walk-between (WBSS) super-sections where only one CM was producing coal at any given time with the CM operator and his or her helper walking back and forth between machines. Eventually, the helper was replaced with a second operator and each operator was assigned to one CM, moving it to a new face when it was not loading coal. As super-sections increased in popularity, the dual-crew (DCSS) or dual-split (DSSS) super-section with fish-tail ventilation was developed to enable both machines to produce coal simultaneously. The effect was to combine two single CM sections into one.

Super-sections offer several productivity gains to the mine operator. Initially, primary benefits were a huge reduction in the time required to move from one face to another (place change time) and fewer production stoppages due to breakdowns because the second CM functioned as a spare when one CM had problems. Other benefits included a reduction in manpower because the section could be run with one foreman, one utility man, and one repairman; and a reduction in required capital because only one feeder breaker and one section conveyor were needed instead of two. Developments in the underground coal mining industry such as higher horsepower machinery, deeper cuts, more stringent dust regulations, rising capital costs, and declining yields, have changed the economics of super-sections to a certain extent, but their use is still justified and they remain a popular and effective mining method (Suboleski and Donovan, 2000).

Haulage systems. To the mine engineer, haulage encompasses a broad array of activities including transporting workers to their stations in personnel carriers, moving supplies from storage areas to the work area on supply cars, and moving coal from the mining section to the surface on conveyor belts, in skips, or in rail cars. While these haulage systems are important, this dissertation focuses strictly on face haulage or the movement of coal from the CM to a conveyor feeder. Due to the repetitive nature of face haulage, as was shown in Table 1.2, it is the foundation of CM production systems and is the part of the system that offers the greatest potential for productivity improvements. Face haulage systems are either continuous or batch, the latter being more common.

Batch haulage. Batch haulage systems consist of individual vehicles called cars. Coal is loaded into cars by the CM when they arrive at the face and they haul that coal to a dump point where a feeder transfers it onto a conveyor belt. The number of cars used depends on the car design and the mine plan. Because entry widths and car sizes make it impossible for cars to pass each other in the same entry, "change-out delays" are inherent. A change-out delay begins when a loaded car departs from the CM and lasts until the next car arrives for loading. During this delay, the CM operator usually continues to cut coal and fill the pan; however, no loading occurs. While change-out delays cannot be eliminated, they can be reduced by maximizing car capacity, minimizing pillar size, keeping the change-out point as close to the face as possible, and routing cars to avoid having to turn around at the change-out point. Of these factors, numerous studies clearly show that haulage unit capacity has the greatest impact on productivity (Smith and Blohm, 1978; Hanslovan and Visovsky, 1984; King and Suboleski, 1991; Sanda, 1998; Chugh, 2001a; Chugh, 2003; Hirschi *et al*., 2004).

In addition to change-out delays, a second delay is possible in batch haulage systems. If an empty car is not waiting at the change-out point when the loaded car passes, then a "wait – no car" delay occurs. Most operations are able to eliminate this delay by keeping a sufficient number of cars in the loop so that an empty car is always waiting at the change-out point; however, this does not always benefit the operation in terms of equipment utilization. In addition to adding cars, the "wait – no car" delay can be minimized by keeping the dump point close to the face to minimize haul distances and by optimizing mine planning in terms of number of entries and entry spacing.

Cars used in batch haulage systems are either tethered or untethered. Tethered cars are called shuttle cars and untethered cars are called ramcars (see Figure 2.3). Joy Machine

Company introduced the first shuttle car into a US coal mine in 1938 (Brezovec, 1982) and they continue to dominate the shuttle car market. Although a great variety of haulage equipment has been tried and continues to be used throughout the industry, the shuttle car remains the preferred haulage device accounting for 78% of all underground haulage (Stefanko, 1983; Sanda, 1998). A shuttle car is open on either end and has a chain conveyor in the bed to “shuttle” coal from the loading end to the discharge end. These cars are powered by an electric cable connected to the section power supply transformer. A powered cable reel on the car lets cable out and takes it in as the car travels back and forth between the miner and dump point. The number of shuttle cars used and shuttle car haulage routes are limited because one car cannot cross the cable of another car. This limitation results in large change-out delays and some “wait – no car” delays. Studies indicate that even in the best shuttle car systems, 15-25% of available production time is lost to change-out delays (Stefanko, 1983; King and Suboleski, 1991).

Figure 2.3. Batch haulage units – shuttle car (left) and ramcar (right) (pictures provided by Joy Global).

Ramcars are cable-less vehicles developed to alleviate shuttle car restrictions. They are built like a truck with a power unit on one end and a trailer bed on the other. Hydraulic steering articulates the car around a center joint between the motor end and the trailer bed. Hence, ramcars are often referred to as articulated haulage. The power unit can be diesel or battery. The bed has a ram plate in it that is retracted as the car is loaded. The ram plate then pushes coal out of the bed at the dump point. Because no cable restrictions exist, there is much more flexibility in the number of cars and haulage routes used; however, ramcars load and dump from the same end requiring the car to turn around twice in each haulage cycle, which can add to change-out and “wait – no car” delays. Also, space taken up by the power unit typically reduces the capacity of a ramcar when compared with a similar sized shuttle car.

Ironically, the first shuttle car was battery-powered (Brezovec, 1982); however, a cable version came out the next year and it has been the industry standard since. Early battery units could not provide sufficient power and traction in difficult mining conditions. In 1978, diesel ramcars were introduced to the underground coal industry as a cable-less alternative with 4-wheel-drive power sufficient to navigate steep grades on wet mine bottoms. The initial US Bureau of Mines test was successful with vehicle performance far exceeding original expectations. Payloads up to 13 tons were hauled in wet conditions up and across grades pitching 25% (Gunderman, 1979). Proven productivity gains quickly boosted the popularity of diesel ramcars while regulatory agencies studied health and safety aspects of diesel emissions. When filtration and ventilation requirements finally became law, diesel equipment popularity disappeared as fast as it had come into being; however, their introduction clearly showed the value of flexible cable-less haulage and equipment manufacturers took up the torch and ran with it, taking advantage of improved battery technology to develop battery ramcars that were competent hauling significant loads in difficult conditions (Sanda, 1998).

In 1998, Phillips Machine Service began development of a diesel-electric shuttle car in an effort to combine the capacity advantage of shuttle cars with the flexibility advantage of cable-less haulage. The end result was a battery-powered shuttle car, named the Freedom Car, shown in Figure 2.4. MSHA permissibility testing and approval was completed in 2001, the car was demonstrated with very favorable results at several Appalachian coal mines throughout 2002 (Skinner, 2003), and the first commercial units were shipped in 2003 (Hirschi *et al*., 2004). The author was involved in arranging a trial demonstration of the Freedom Car at an Illinois mine in 2006. Phillips Machine Service is primarily a rebuild shop and their inability to be competitive with larger OEMs kept the Freedom Car from gaining the interest it deserved.

Figure 2.4. Freedom Car (picture taken by the author).

Table 2.1 (from Hirschi *et al*., 2004) summarizes strengths and weaknesses of batch haulage systems used in underground coal mines during the last decade. Joy Global dominates the cable shuttle car market, OEM mergers and acquisitions have reduced battery ramcar suppliers to basically two – Joy Global and Caterpillar, only Phillips Machine Service offers battery-powered shuttle cars, and diesel ramcars have pretty much been regulated out of existence.

Table 2.1. Advantages and disadvantages of common batch haulage systems (from Hirschi *et al.*, 2004).

Advantages	**Disadvantages**
Cable Shuttle Car	
• Capacity	• Fixed haulage route, no overlap
• No turn around required	• Number of units is limited
• Controlled dumping	• Cable maintenance delays
• No harmful emissions	• Confined travel distance
Battery Shuttle Car	
• Same as cable shuttle car	• Higher capital costs
• Flexible haulage routes	• Battery reliability and maintenance
• Number of units not limited	• Battery change-out delay
Diesel Ramcar	
• Flexible haulage routes	• Risk from diesel PM undefined
• More power in bad conditions	• Capacity
• Number of units not limited	• Switch out or turning required
• Unconfined travel distance	• Uncontrolled dumping
Battery Ramcar	
• Flexible haulage routes	• Capacity
• Number of units not limited	• Switch out or turning required
• Unconfined travel distance	• Battery change-out delay
• No harmful emissions	• Battery reliability and maintenance

Continuous haulage. The CM produces only intermittently in a batch haulage system because of the change-out requirement. To realize their full potential as "continuous" miners, continuous haulage was devised with the US Bureau of Mines playing the key role in establishing multiple industry collaborations to develop the technology. Chain and belt conveyors from CMs and regular conveyor belts were adapted beginning with simple bridge systems followed by extensible belts, bridge conveyor systems, and modular interconnected conveyors (Evans and Mayercheck, 1988). Successful systems included the monorail bridge conveyor, the multiple-unit continuous haulage (MUCH) system, the mobile bridge conveyor (MBC), and the flexible conveyor train (FCT), each of which went on to successful commercialization by cooperating OEM partners. In the last two decades of the 20th century, five OEMs supplied about 150 continuous haulage systems with the MBC and the FCT shown in Figure 2.5 faring the best. Systems cover a range of mining heights including western seams as high as eight feet, but continuous haulage has a definite advantage in lower seams, particularly 40 inches and below, because low seam height restricts the full utilization of batch haulage capacity (Sanda, 1998).

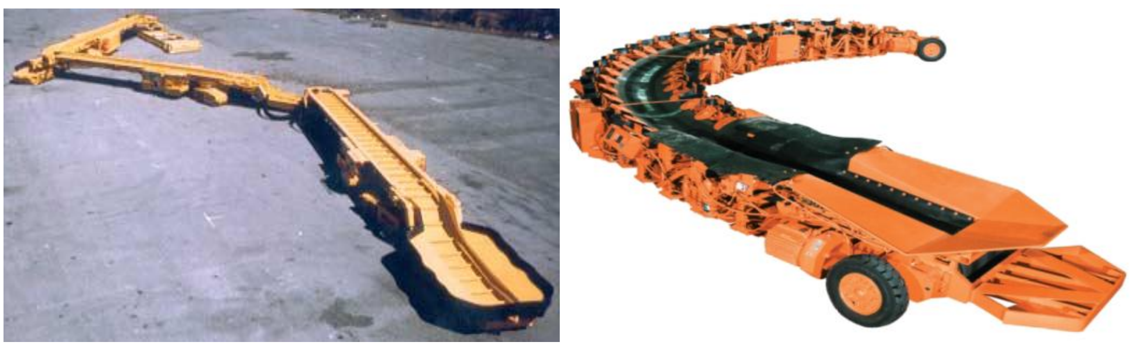

Figure 2.5. Fairchild's MBC (left) and Joy Global's FCT (right) (pictures provided by OEMs).

Continuous haulage connects the CM with the section conveyor belt allowing the CM to load non-stop from beginning to end of cut; however, they are usually slow changing places and they make it difficult to move the roof bolter and supply vehicles across the section (King and Suboleski, 1991). The importance of continuous haulage technology has steadily increased due to the natural depletion of massive blocks of coal best suited for longwall mining. Nevertheless, while much of the remaining proven underground reserves may require using continuous haulage systems to be economically mineable (Coal Age, 2003), experience has shown that continuous haulage can be very productive only under certain ideal conditions, and it is not very flexible in adapting to complications when one or more of those conditions do not exist. The ideal conditions are as follows (McGolden, 2003):

1. Reserve configuration – resources necessary to set up panels are significant; short panels are generally not cost effective (Sprouls, 1998); in addition to long panels, adjacent panels should be close; long moves between panels and irregular-shaped reserves restrict productivity (Sanda, 1998).
2. Roof stability – belt entry must be wide enough for the system to move alongside the belt; 22 feet is the typical width that must be maintained keeping supplemental support to a minimum.
3. Coal seam consistency – continuous haulage can only negotiate limited height variations and undulations in the coal seam; any limits affect overall productivity.
4. Geologic conditions – continuous haulage is not as adaptable to poor conditions as batch haulage; wet, muddy mine floors will especially hamper productivity.

Surge car. One as yet undeveloped option that could merge the continuous loading capabilities of continuous haulage systems with the flexibility of batch haulage systems is the surge car concept. The surge car is a pass-through hopper positioned directly behind the CM for the duration of each cut providing storage capacity for CM output while batch haulage units are changing out. When a haulage unit is in place, it unloads quickly due to a much higher loading rate than the CM. This accelerates haulage cycle times and eliminates time the CM spends waiting for haulage units thereby improving overall cut-cycle time. The surge car would also allow the CM to operate independent of haulage units during the clean-up segment of the cut when loading rates are lower and CM and haulage units are continually repositioning. (Hirschi *et al*., 2004).

Previous attempts to develop a surge car have met with little success, primarily because too many "extras" were included rather than focusing solely on providing surge capacity. The first documented surge car attempt occurred in the 1970s combining surge car and continuous haulage concepts. The system consisted of an extendable conveyor belt system with the tail roller mounted to the rear of a crawler-mounted hopper car. As the hopper car advanced with the CM, belt was pulled out of a take-up device and it ran on the floor and on itself until the distance of advance permitted belt structure to be manually inserted between carrying and returning sections of belt, which could be done while the system was in operation provided workers stayed caught up (Haynes, 1975). The system was straight-line and could not negotiate turns. After a relatively short trial, it was abandoned due to a cumbersome roof control technique (McWhorter, 2004).

In the 1980s, the US Bureau of Mines designed, fabricated, and tested a Hopper-Feeder-Bolter (HFB). The objective was to combine roof bolting, lump breaking, and surge capacity functions into one machine. Because the HFB could bolt beside the CM, entry-to-entry places changes were replaced with in-place side-to-side equipment changes (Evans *et al*., 1988). The inclusion of roof bolting limited CM production and after a 12-week trial in a Midwestern coal mine, it was abandoned (Mayercheck, 1988).

In the 1990s, Stamler Corporation introduced a surge car designed to reduce total production time for each cut by loading haulage units faster and fuller (a natural byproduct of the faster loading rate) than the CM thereby reducing haulage cycle times and the number of haulage unit trips required for a cut. It was mounted on crawlers and resembled a feeder breaker with power supplied via an electric cable similar to the CM. Underground trials were performed and despite the fact that the cutting and loading portion of the total cut-cycle was reduced, because two machines had to be moved during place changes, the net effect on unit shift productivity was negligible and the machine was never used beyond the field trial and demonstration stage (Combs, 1993).

In 2004, the author participated in a collaborative effort between academia and industry to secure Federal funding for designing, developing, and demonstrating a surge car based the Freedom Car concept. Studies indicated that a surge car with the mobility and flexibility of a haulage unit had the potential to increase productivity of a typical batch haulage system by as much as 30% (Johnson and McGolden, 2004). This study also showed, however, that the value of the surge car is a function of the haulage system and without an existing optimized haulage system, little gain would be realized by incorporating a surge car. In fact, because the surge car adds an additional piece of equipment at the face requiring an extra person to operate it, some applications would result in a negative impact on productivity.

2.3 Room-and-Pillar Mine Design

Room-and-pillar mine plans divide a coal reserve into sections or blocks known as production panels, which are accessed and connected by a network of main and submain excavations. Main and submain entries and crosscuts are designed to remain open for the life of the mine as they become part of the mine's infrastructure and are used for travel, ventilation, conveying, storage, and other purposes after actual mining takes place. Production panel entries and crosscuts are designed to remain open for the life of the panel during which as much coal as possible is removed. For economic viability of the coal mine, pursuing maximum “extraction” consistent with desired structural “stability” must be accomplished in both mains and panels.

Extraction. Room-and-pillar systems are inherently limited in the level of extraction achievable because of the pillar component of the system. Extraction ratios (the ratio of coal removed to original coal in place) vary depending on the depth of the coal seam and whether mining is in main entries or panels. Deeper mines require bigger pillars to support the overburden. Main entries also require bigger pillars to insure long-term stability for mine infrastructure. Thus, extraction ratio decreases as depth of coal seam increases and lower extraction ratios are achieved when mining main entries versus production panels. While coal seam depth is fixed, the mine engineer can design the mine layout with minimum main and submain footprints allowing a larger percentage of the coal reserve to be mined in production panels at higher extraction ratios thereby maximizing the amount of coal extracted or recovered from the reserve.

The traditional approach to room-and-pillar design in coal seams has been to create pillars of uniform size, albeit with different sizes for mains, submains, and panels. While this practice is conducive to simplified production management in general and mining sequences in particular, it results in sub-optimal extraction ratios. To increase extraction ratios to more optimal levels without significantly adding to the complexity of production management, the SIU research team investigated appropriate applications of the alternate mining geometry (AMG) concept first proposed by Chugh and Pytel (1992). The AMG concept consists of a mining section layout with pillars of unequal sizes created by varying entry spacing across the section while maintaining uniform crosscut spacing along the section such that larger pillars are created in the center of the section to protect conveyor and power supply systems typically located in that area, and smaller pillars are created at both edges of the section where they benefit from being in the shadow of barrier pillars, blocks of coal separating panels from mains or other panels.

The AMG concept has been demonstrated at two southern Illinois mines, both of which were carefully studied by the SIU research team. In the first demonstration at a deep mine (Chugh, 2001b), the extraction ratio increased from 41.8% to 43.8% for an 8-entry production panel. In the second demonstration at a shallower mine (Chugh, 2006a; Chugh 2007), the extraction ratio increased from 53.3% to 56.5% for a 12-entry submain. Production modeling of the second demonstration indicated that a productivity increase of almost 7% was possible primarily due to the ability to mine crosscuts between outside entries in one cut. Unfortunately, the timing of both demonstrations precluded careful and detailed monitoring of productivity enhancements to confirm model predictions.

Stability. Peng and Finfinger (2001) described the challenge involved in constructing stable openings as follows:

> "*Mining engineers work with the most challenging construction materials. They must deal with rock materials as they exist in their natural states and design mine structures without well-known and defined properties. Further complicating the design process is the variability of the in situ rock materials with rock types and rock properties varying widely from place to place.*"

In any type of underground mining, single entries, or an entry sufficiently isolated to be considered single, have the best stability, but a single-entry coal mine is completely impractical. Hence, in room-and-pillar coal mining sections, entries are usually driven in sets with the number in a set determined by production demands as well as requirements for face haulage and general mine services such as ventilation, escapeways, power supply and conveyor systems, water drainage, and men and material transport. Entry width is a function of roof strength with other factors including floor and coal strength, size of equipment, and regulatory statutes, being of lesser importance (Obert and Duvall, 1973). Factors of safety with respect to roof, pillars, and floor are used to design excavations of the desired stability with numerous studies (see Mark and Barczak, 2010 for an excellent summary) having delineated proper criteria for determining adequate safety factors.

In testing the AMG concept, the objective of the SIU research team was to establish more uniform pillar and floor safety factors across the entire width of a mining section (Chugh, 2007). Pillar safety factor (PSF) refers to failure of a coal pillar. Holland's formula (1964; 1973) is used to find pillar strength as follows:

$$\sigma_p = \sigma_{cc} * \sqrt{\frac{Wp}{h}} \tag{2.1}$$

where σ_p = *in situ* pillar strength,

σ_{cc} = critical size or *in situ* coal strength,

W_p = pillar width,

and h = coal seam height.

The PSF is the ratio of pillar strength to load on the pillar as follows:

$$PSF = (\sigma_p) / [(1.1 * D) / (1 - e)] \tag{2.2}$$

where D = depth of overburden

and e = extraction ratio.

Due to Illinois' weak floor conditions (Chugh *et al*., 1990), floor safety factor (FSF) must also be accounted for. FSF refers to failure of the floor underneath the coal pillar or foundation failure. Mohr-Coulomb failure criterion is commonly used to calculate rock failure or yielding as follows (Hoek and Brown, 1980):

$$\tau = \sigma_n * \tan(\phi) + c \quad (2.3)$$

where τ = shear strength of the failure plane,

σ_n = normal stress on the failure plane,

ϕ = angle of internal friction for rock,

and c = cohesive strength of rock.

Knowing these parameters and the average bearing capacity of the floor obtained from in-mine plate load tests, the ultimate bearing capacity of the roof-coal-floor column is determined as follows (Chugh *et al*., 1990; Chugh and Hao, 1992):

$$S_1 = (\text{ Average Bearing Capacity } / N_c^*) \quad (2.4)$$

where S_1 = cohesion

and N_c^* = bearing capacity factor from literature (6.17 assuming ϕ = 0).

$$q_O = S_1 N_m \quad (2.5)$$

where q_o = ultimate bearing capacity (UBC) of column,

and N_m = modified bearing capacity factor.

Vesic (1975) proposed the following equation for determining N_m:

$$N_m = \frac{KN_C^*(N_C^* + \beta - 1)[(K+1)N_C^{*2} + (1+K\beta)N_C^* + \beta - 1]}{[K(K+1)N_C^* + K + \beta - 1][(N_C^* + \beta)N_C^* + \beta - 1] - (KN_C^* + \beta - 1)(N_C^* + 1)} \quad (2.6)$$

where K = unconfined shear strength ratio of lower hard layer (S_2) to the upper weak layer (S_1), and $$\beta = \frac{BL}{[2(B+L)H]}, \tag{2.7}$$

which can be found from the width (B), length (L), and thickness (H) of the foundation (weak floor). B and L correspond to pillar width (W_p) and pillar length (W_l), respectively.

Finally, the FSF is the ratio of UBC to load on the pillar column as follows:

$$\text{FSF} = q_o \;/\; [(1.1 * D) \;/\; (1 - e)]. \tag{2.8}$$

Stable pillar designs for room-and-pillar coal mines in Illinois require minimum PSF and FSF of 1.5 and 1.3, respectively (Chugh, 2007).

For uniform pillars in a conventional mining geometry (CMG), size is based on the greatest pillar loading, which is seen on middle pillars of the section if using the tributary area theory to determine that load. That being the case, pillars on both outside edges of the mining section are overdesigned based on PSFs and FSFs determined for the CMG. Thus, as shown by the comparison in Figure 2.6 of safety factors for the second AMG demonstration and its corresponding CMG, the unequal-sized pillars of the AMG are more efficiently designed.

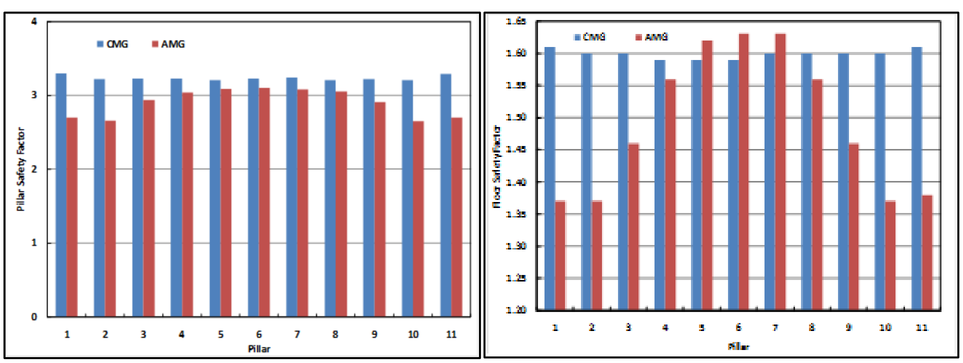

Figure 2.6. Comparison of pillar (left) and floor (right) safety factors across CMG and AMG sections (Chugh, 2007).

Crosscuts. Crosscuts are an essential component of room-and-pillar systems as mining them completes pillar creation, provides ventilation air to the face, and increases face haulage mobility. The first crosscut (or two) in a row of crosscuts must be “turned,” which is generally the slowest mining. At the same time, crosscut centers must be carefully planned as they determine change-out distances, a critical component of any mining sequence optimization algorithm.

Roof support systems. Perhaps noticeably absent in the previous section on continuous miner production systems was any discussion of roof bolting. That is because this dissertation focuses only on those functions that are directly involved with production. While roof bolting is a critical component of any room-and-pillar mining system and installing roof bolts is the most repeated task performed by a production crew, it does not produce any coal. Therefore, roof bolting will only be included in this study to the extent that the CM and the roof bolter must travel in the same entries and crosscuts and work in the same limited number of entries at the face. This will require verifying that CM travel routes in an OMS will not conflict with being able to get the roof bolter into unbolted cuts in a prompt and timely manner.

2.4 Characterization of Dust Exposure for Different Cut Configurations

Dust control research in the US has been extensive as summarized by Colinet *et al.* (2010). This study does not attempt to develop or promote any technologies specifically aimed at improving dust control in underground coal mines. Rather, the objective is to develop a tool that could lead to improved mining practices with regard to mining sequences that minimize exposure to respirable dust. To that extent, only that portion of the dust control literature that relates specifically to dust exposure levels for the various cut configurations encountered in mining sequences will be reviewed.

The National Institute for Occupational Safety and Health (NIOSH) is one of the leading research organizations working on dust control in underground coal mines. They evaluated face dust concentrations at six mines making deep cuts (Potts *et al*., 2011), the practice of mining a cut in two increments to achieve the maximum depth allowable under ventilation and roof control plans approved by MSHA. The ability to make deep cuts is desirable as it reduces the number of times the CM has to change places. Through the 1980s, the CM operator sat in a cab on the back of the machine where all of the controls to operate the machine were located. As the CM operator was not permitted to travel under unsupported roof, the depth of a cut was limited to the a few feet less than the length of the machine, usually about 20 feet. Technology improvements to the CM brought about the previously described flooded-bed scrubber and remote control capabilities. The latter allowed the CM operator to get off of the machine and stay under supported top while the CM advances under unsupported top to depths beyond 20 feet. The suctioning power of the scrubber draws ventilation air into the newly excavated area to control dust and methane even as the cut extents to greater depths. Of concern was that the scrubber did not do an adequate job during the second (deeper) half of the cut, but this was shown not to be the case as long as mining practices previously shown to control dust at any cut depth were carefully followed.

Two studies by the SIU research team provide data on dust exposure levels for different configurations of the CM in terms of entry positioning. During an extensive evaluation of the “wet-head miner” (WHM), a CM with water sprays behind each bit on the cutting drum, seven different types of cuts were identified. While dust sampling was performed for each type, results were not reported as they were inconclusive due to a high degree of variability in the measurements (Chugh, 2006b).

A second attempt was made to quantify dust exposure levels for different configurations during demonstration of SIU's innovative spray system for CMs. This time, eight different configurations were identified and sampled. Results indicate that three cut configurations have the highest dust exposure levels and three cuts have the lowest exposure levels. Those configurations with the highest exposure levels are cuts that begin in an entry that is already deeper than two lengths of the CM, cuts that complete a crosscut with the mining direction opposite that of the ventilation air flow, and cuts that begin a crosscut with the CM turning out of the entry into the crosscut. Those configurations with the lowest exposure levels are cuts that begin in an entry that's depth is less than half the length of the CM, cuts that begin a crosscut with the CM positioned in a completed crosscut perpendicular to the entries being connected (head-on mining), and cuts that complete a crosscut with the mining direction the same as that of the ventilation airflow (Chugh, 2012).

2.5 Operations Research and Computer Modeling in Underground Coal Mining

Industrial engineering. As the Industrial Revolution moved work from the cottage to the factory, industrial engineering (IE) was created to establish and improve work rates, define job descriptions, and implement correct job procedures. From there, IE progressed into plant location and design, tool design, quality control, inventory control, and production scheduling. The strain of war brought about the development of operations research (OR) techniques, which is generally concerned with optimal decision making through the process of applying statistics and higher mathematics to the solution of real-world, human-based problems. Initially, most IE efforts were corrective in nature, addressing problems as they arose. Following World War II, the attention of IE professionals turned to prevention and quality control. It was only natural that, in the underground coal mining industry where the increasing presence of mechanized equipment provided a ready-made environment, IE methods would be used to evaluate the complex interaction between humans, machines and their surroundings (Douglas, 1980). Throughout the 1950s, considerable effort went into conducting time studies and developing work improvement methods for underground coal mines.

Computer applications. One of the major contributing factors to the rapid application of IE and OR concepts throughout the industrial work place, including underground coal mines, was development of the computer. Computers enabled IE professionals to utilize computer modeling and simulation techniques. Mine engineers have been interested in using computers to build models that simulate mining operations ever since the computer was introduced to the industry in the 1960s (Sturgul, 1995). 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. Mine engineers were applying IE principles and techniques before the computer, but computerized procedures provided the important advantage of producing results quickly using exactly repeatable methodologies that allowed the engineer to focus on relevancy and accuracy of input data and analytical methods used rather than on computational mechanics (Douglas *et al*., 1983).

Mine modeling. The development of IE systems analysis in the mining industry most likely began with the “Combined Study Method” developed at Pennsylvania State University (Penn State) in 1949. It explained how to layout and time study a mechanized section leading to widespread use of time studies and modeling of production methods by mining companies and consultants. It included information gathered from over 100 coal mines (Bise and Albert, 1984) and became the foundation for later models and simulators used throughout the industry.

In the early 1950s, a West Virginia company known as Coal Standards developed what many believe to be the first mathematical model of an underground coal mine. Several of Coal Standards' founders and engineers had been trained on the Penn State time study method and went on to become prominent mining engineering professors.

During the 1960s, researchers started building computer simulation models of mining operations. FORTRAN was used primarily because it was the standard scientific programming language at the time. Programming was slow and a considerable amount of time and effort was spent writing and de-bugging programs. Most of this work was done on college campuses as few mine engineers in the field had the time or computer resources to devote to such a task.

Engineers at Virginia Polytechnic Institute and State University (Virginia Tech) developed some of the first mathematical programs to model loading and hauling components of conventional and continuous mining systems (Prelaz *et al*., 1964). The original "Mathematical Model" was a FORTRAN model that took several months to write. After spreadsheet software revolutionized computing in the early 1980s, the "Mathematical Model" was converted to a spreadsheet (Lotus) model in just a few days and in the 1990s it was converted to Microsoft® Excel (Suboleski, 2004). The model had the limitation that only average values could be used to evaluate equipment; variability could not be introduced.

To overcome this limitation and generally improve the model, SIMULATOR1 (later known as FACESIM) was developed to simulate the entire operation of conventional or continuous mining systems (Prelaz *et al*., 1968). FACESIM allowed distributions to be entered to account for variations in equipment rates and speeds. It was an event-oriented program wherein the time or rate for each event was sampled throughout the entire shift from user-supplied distributions. FACESIM was eventually converted to a PC-based program and was still being used in the 1990s.

Mine Simulation. Modeling with average values produces a single mathematical solution to an analytical problem. Simulation, on the other hand, is a technique that is "run" rather than "solved". The difference is notable in both the effort required to develop input and in the extent of output. To get a distribution of results from FACESIM, a mining scenario had to be repeatedly modeled and each result manually recorded. Wanting output answers to appear as a distribution produced from hundreds of runs, a Virginia Tech team modified it, developing CONSIM, a discrete, event-oriented simulation model that tracks events updating the status of the system as well as tracking clocks every time an event is completed (Topuz and Nasuf, 1985). It automatically generates much of the input data while allowing for both deterministic and probabilistic interaction between equipment over any time period specified by the user.

Penn State also had a team working on computerized simulation. The first simulation model was completed in 1961 and others soon followed. As various projects took shape, it was decided to merge them all into a Master Design Simulator (MDS) that would include geology and reserves, ventilation, methane drainage, rail haulage, ground control, and cost as well as production systems analysis (Ramani *et al.*, 1983). The production model, called Underground Generalized Materials Handling Simulator (UGMHS), utilized equipment performance characteristics rather than time study data and had the ability to provide detailed outputs such as motor torque curves for predicting rim-pull on shuttle car drive wheels (Sanford and Manula, 1969). UGMHS was a "time-slice" simulation model written in FORTRAN but a specialized "language" was used to input geometric data such as tramming paths of equipment (Suboleski, 2005).

The Bureau of Mines was very active during this period of computer modeling and simulation and they also developed a production simulator (Hanson and Selim, 1975). Their model had the user specify a calendar of events to be simulated, thus permitting ready application to any type of mining operation including longwalls.

In the 1980s, the KETRON consulting group availed themselves of Penn State's ventilation module and Virginia Tech's conveyor simulator and developed their own production module to form a Mineral Industries Software System. The production module, called CPMINE, employed the critical path method (CPM) combined with statistical variations in data inputs to produce histograms of various mine parameters (Douglas *et al*., 1983). KETRON also modified UGMHS to produce graphical output of equipment working (Suboleski, 2005).

As computing became more commonplace, familiarity with computers brought about increased use and vice versa. Higher level programming languages made their appearance including those specifically designed for simulation, such as GPSS, SIMSCRIPT, and GASP. These languages reduced the programming skills needed to create detailed simulation models and later versions even included capability for animation (Sturgul, 2000).

Southern Illinois University/Suboleski Production (SSP) Model. To effectively analyze production systems by way of modeling and simulation, two items are critical. The first is to have a simple model that is easy to use and understand. Creating a simple model requires two key decisions (Leemis, 1995). First, which elements of systems should be excluded or included? Second, what level of detail should be used to represent components? Eighty percent of the value of a simulator is derived from thinking through the mine plan and inputting data

(Suboleski, 2004). If the model is so complex that the human mind cannot follow the logic that is used and visualize how the model mathematically describes the mining process, then there is little practical value and the likelihood of the model being used for process improvement is slim.

The other key item required of a useful model is accuracy and relevancy of input data. Development of realistic production data is not an easy task due to varying conditions, which are often used as an excuse for not formulating an objective approach using controllable factors to predict productivities. This lack of information limits the control ongoing operations have in affecting productivity improvements since changes cannot be measured against expected practice to determine which factors have the most impact.

In 2000, the SIU research team identified in Chapter 1 began investigating strategies for reducing the cost of producing Illinois Basin coal. A deficiency in readily available production models was recognized and the team went back to models of the 1960s. Dr. Stan Suboleski, one of the "Mathematical Model" creators, provided the Microsoft® Excel version (Suboleski, 2002), which he had used in various productivity studies (King and Suboleski, 1991; Suboleski and Donovan, 2000). The model's predicted production outputs were compared with data from several Illinois Basin mines and proved to be extremely accurate. Most importantly, the model enhanced the exercise of thinking through the mine plan during the modeling process.

After developing a familiarity with the model, the team felt that data input was somewhat cumbersome and prone to error propagation, and the size of the spreadsheet made finding desired output a challenge. Consequently, the model was upgraded with separate input and output sections and named the SIU/Suboleski Production Model or SSP Model (Hirschi *et al*., 2004). The SIU research team then developed additional modules for analyzing roof bolting constraints, out-of-seam dilution (OSD), and production costs, which were incorporated into the SSP Model

(Moharana, 2004; Chugh *et al*., 2005). The SSP Model was used extensively throughout the Illinois Basin as part of a miner training program to enhance underground coal mine productivity (Kroeger, 2004; Kroeger, 2006). The one drawback of the SSP Model has been the time-consuming process of entering cut sequences, which had to be repeated any time a different sequence was to be evaluated for productivity improvement potential.

Computer modeling and simulation are valuable tools in mine planning and process improvement efforts, but to be relevant, computer models require accurate input data. There are many input parameters, but the cut sequence is a foundational one. All of the production models just reviewed would heuristically determine an optimum cut sequence, if that is one of the desired outputs. Doing so requires numerous iterations, each modeling a different cut sequence. Inputting a cut sequence is a time consuming process even with the most advanced input modules. Thus, even when beginning with a sequence expected to be optimal based on reputable mining experience, inputting cut sequences for multiple iterations of the model is required to confirm that the starting sequence is indeed optimal.

2.6 Dynamic Programming Optimization Techniques in Mining

As stated previously, OR is the engineering discipline focused on optimal decision making for and modeling of real-life situations. A survey of OR applications in the mining industry (Topuz and Duan, 1989) identified eleven different techniques that had been widely used. Dynamic programming (DP) is just one OR methodology used for optimizing multi-stage decision processes. Others methods for solving these types of problems include decision theory, Markov chains, Monte-Carlo method, and stochastic programming (Bronson and Naadimuthu, 1997). The Topuz-Duan survey distinguished between methods better suited for deterministic systems with no randomness and stochastic or probabilistic systems, and DP was identified as a deterministic method while the other methods were identified as probabilistic methods. Because the original aim of this project was to develop a cut sequencing module for integration with the SSP Model, which is a deterministic model, DP was chosen as the methodology to solve the OMS problem.

OR techniques in general and DP in particular have seen many applications in different mining scenarios. Production planning and scheduling is the area of application that has seen the greatest use of OR methods according to the Topuz-Duan survey (1989). Long-range production planning problems such as determining optimal open pit limits (Lerchs and Grossman, 1965; Johnson and Sharp, 1971; Caccetta and Giannini, 1986) and short-range production scheduling problems such as grade control in underground sub-level stoping operations (Dowd, 1976; Dowd, 1980; Ribeiro, 1982; Dowd and Elvan, 1987) have been solved using DP. Wang and Huang (1997) used DP to develop a cut sequence for an open pit mine that optimized the net present value of the ore produced. Equipment selection and scheduling has been another frequent application of DP methods such as a study by White and Olsen (1986) on dispatching haul trucks. Grayson (1989) formulated a DP model to optimally allocate coal miners among various work activities on an operating shift basis. The author believes this is the first application of DP to the problem of determining an OMS for a room-and-pillar coal mine. The methodology used in this study show some similarities to the cut-sequencing study by Wang and Huang as well as those production scheduling applications just cited; however, all of these applications develop decision policies based on parameters (both deterministic and stochastic) that define fixed stages in the problem, whereas this study develops a decision policy that is based on parameters defining paths between different stages as well as stage-specific parameters.

DP is a recursive, or step-by-step, approach, where at each stage, decisions are made after analysis, providing information used in succeeding decisions. The process continues through all stages until the decision criteria is optimized in the final stage giving an optimal policy that can be traced back through the process. To apply DP to the mining sequence optimization problem of a CM production system in a room-and-pillar coal mine, the following items must be defined (Grayson, 2002):

- Stages,
- Constraints or feasible states at each stage,
- Optimal value function, and
- Recurrence relation.

The state, X, of the system is the number and position of cuts remaining to be mined. For example, cuts remaining to be mined for one crosscut of advance in a 3-entry longwall gate road development are illustrated by dashed lines in Figure 2.7. In a room-and-pillar mining system, a stage, identified by the letter *i*, is a cut. "*Cut i-1*" identifies a cut that has just been mined. The objective at each stage is to select a cut for mining that best satisfies the optimal value function, $f_i(X)$. For the sequence depicted in Figure 2.7, it is assumed the cut i_b best satisfies $f_i(X)$ at each stage. At each stage, only those cuts that are feasible can be selected from. To be feasible, the cut must be accessible by way of previously mined cuts. As shown by subscript letters *a* and *b* in Figure 2.7, more than one feasible state may exist at each stage.

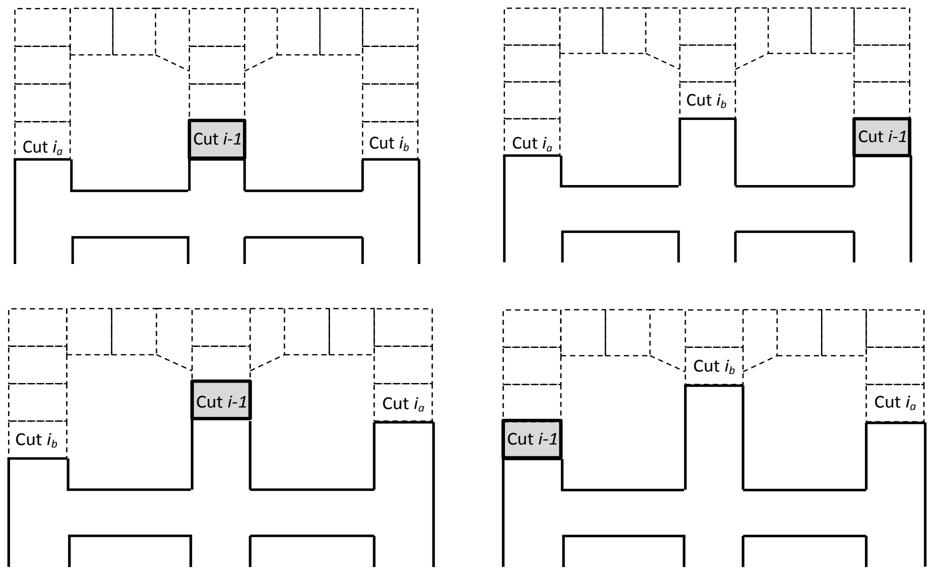

Figure 2.7. A sequence of four stages in a 3-entry longwall gate road development section with cut i_b selected at each stage. Stage 1 (top left), Stage 2 (top right), Stage 3 (bottom left), and Stage 4 (bottom right).

Constraints can also be used to define feasible states. For example, cuts just "inby" "*Cut i-1*" in Figure 2.7 are not identified as feasible because they have not yet been bolted. Constraints also define how and when crosscuts are mined to satisfy ventilation and roof control plan requirements. Other irregular factors, such as abnormal geologic conditions or equipment breakdowns, can also be defined as constraints.

The optimal value function can be a maximization or minimization problem, such as maximizing production in a given timeframe (unit shift productivity) or minimizing haulage distance or delays (equipment utilization). The DP algorithm developed in this dissertation is a minimization function. The objective is to minimize cut-cycle time, which could also be construed as maximizing CM utilization given the distinction as stated earlier that CM utilization is defined as the CM cutting and loading coal.

The recurrence relation considers the repetitive nature of cut sequences, such as haulage distances based on mining geometry, roof bolting constraints based on time and space, and CM movement, also based on time and space. An optimal decision is made at each stage, and this process continues through all stages until the optimal value function is optimized in the last stage giving an optimal cut sequence for a given cut-cycle (Grayson, 1989; Chugh, 2006b).

2.7 Chapter Summary

The Industrial Revolution was slow coming to the industry that fueled it, but by the mid-1900s, underground coal mining was steeped in mechanization and the trend toward greater mechanization continues to this day. Mechanization lends itself to industrial engineering analysis. Focusing specifically on the room-and-pillar mining method in underground coal mines, industrial engineering work was prolific from 1965 to 1985, most of it being research on college campuses where computing resources were more available. As computing capabilities became more widespread, their use in coal mining shifted from specialized engineering applications to generic management information systems. From 1985 to 2005, industrial engineering activity at underground coal mines was almost nonexistent, but there has been a rebirth in the last few years with the advent of process improvement teams at many operations. Gone are the days when productivity improvements could be realized simply by purchasing bigger, stronger mining machines.

As more attention is devoted to process improvement, the importance of repetitive cycles so common in underground room-and-pillar mining methods is being realized. As was pointed out in Table 1.2, several tasks that constitute the critical path in CM production systems are repeated hundreds of times in a single shift. The mining sequence forms the foundation for all of these repetitive tasks and it too is regularly repeated, at least to the extent that standard operating procedures are in place. Otherwise, the mining sequence depends on the whims of the mine foremen who relies on instincts developed through personal experiences during his or her tenure in the mines. This experience-based mentality seems to have been the methodology used to identify mining sequences that were modeled and simulated during the industrial engineering heyday described earlier, and it continues to be common practice today, despite the existence of techniques, such as dynamic programming, that are well-suited for optimizing repetitive processes.

Dynamic programming is not a new optimization technique; however, it has been used sparingly in mining and this work appears to be the first documented application of the technique to the problem of cut sequencing in underground room-and-pillar coal mines. The algorithm presented in the next chapter applies the dynamic programming optimization technique to the problem of identifying optimal mining sequences utilizing in the DP algorithm both state-specific parameters as well as path-specific parameters, the latter for what the author believes to be the first time.

CHAPTER 3

DYNAMIC PROGRAMMING ALGORITHM FOR OPTIMIZING CONTINUOUS MINER CUT SEQUENCES

3.1 Introduction

At the core of this study is the DP algorithm for optimizing mining sequences, which is presented in this chapter. Development of the algorithm was guided by several fundamental policies, which are presented first. Then the optimal value function, which is the core of the algorithm, is defined. The optimal value function has two primary components which are fully described. The DP algorithm operates within the recurrence relation, which constitutes the DP model. The recurrence relation and how it is used to determine global versus local optimization are explained next. Finally, the DP model can be made to operate within certain constraints, two of which are presented and described.

3.2 Guiding Policies and Practices

The following policies and practices are adhered to in developing the DP model for optimizing mining sequences:

Complete crosscuts in a timely fashion. Completing crosscuts is critical because doing so accomplishes three things. First, completed crosscuts establish and maintain the flow of ventilation air across the mining section and promote the availability of adequate face ventilation as close to each face as possible. Line curtain is hung from roof bolts on one side of the entry to direct ventilation air from the last open crosscut (LOXC) to each face in entries where mining has advanced the face beyond the LOXC (see Figure 3.1); however, ventilation air follows the path of least resistance making line curtain leakage a common occurrence, which reduces the volume of air reaching the face to inadequate levels. Furthermore, line curtain restricts entry width forcing face haulage units to travel at reduced speed, which lengthens change-out delays. Completing crosscuts minimizes the amount of line curtain that must be hung. Second, as crosscuts are used by face haulage units traveling to and from the CM, each completed crosscut establishes a change-out point that is closer to the face thereby minimizing change-out distances. Third, completed crosscuts provide for uniform advance of the section face area. As CMs tram through crosscuts to access entry faces, uniform advance of the section face area minimizes CM tram distances between cuts. Therefore, cuts that deepen entries that are already deep enough to start or complete crosscuts should be avoided.

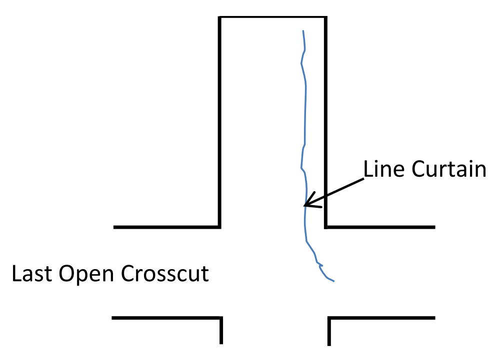

Figure 3.1. Line curtain restricting entry width.

This is illustrated by an example. Consider a mining section with entries and crosscuts on 80-ft centers making 32-ft deep extended cuts. After three cuts in an entry, the face is 96 feet from the LOXC, which means the entry is deep enough to complete one or both of the crosscuts that will connect it to adjacent entries. If a fourth cut is made before a crosscut is completed, nearly 100 feet of line curtain will be required to get air to the face. The cut will be very dusty due to air losses and take a long time to complete due to reduced speeds over longer distances for face haulage units. If the same cut is made after a crosscut connecting it to an adjacent entry is completed, there will be better ventilation at the face because the line curtain will have been removed and what had been the LOXC will now have drop curtain hung in it forcing air up the entry to the new LOXC. Furthermore, with no line curtain restricting the entry, face haulage units can travel at normal speeds reducing change-out delays during the cut.

Maximize starting crosscuts “head-on.” Turning a crosscut from an entry requires the CM to start at an acute angle with the entry and turn while making the cut until it is perpendicular with the entry. This allows the CM cutting drum only incremental contact with the coal face (see left side of Figure 3.2) for much of the cut. Furthermore, it also requires the CM operator to frequently back up and reposition the machine. With each repositioning, the operator needs to check alignment and will usually stand behind the CM to do it, which is a danger zone with face haulage units coming and going. Starting a crosscut head-on with the CM perpendicular to entries being connected allows full contact (see right side of Figure 3.2).

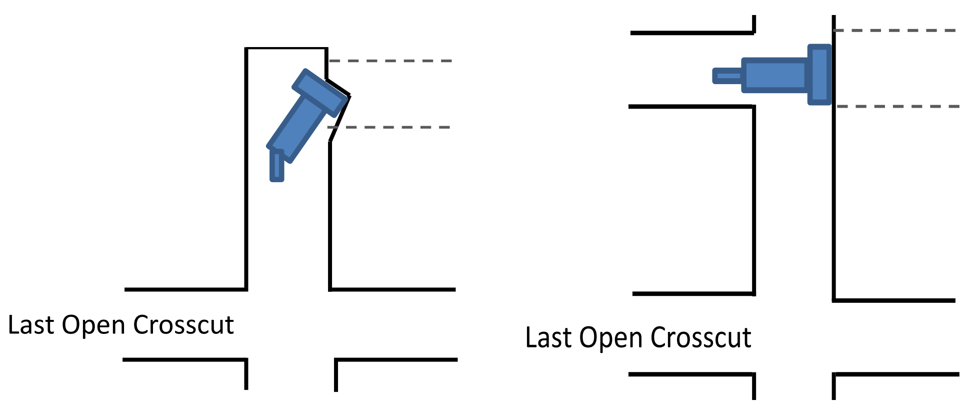

Figure 3.2. Starting a crosscut by turning (left) and head-on (right).

Research by Chugh (2009) found that 80% of roof falls in underground coal mines occur at intersections where entries and crosscuts meet. The diagonal span across an intersection is the widest part of a mine opening and it should be minimized to reduce exposure to unstable roof and roof falls. Crosscuts that are turned create wider diagonals at intersections than crosscuts that are started head-on.

Mine crosscuts in the direction of ventilation air flow. The last cut in a crosscut is called the “hole-through.” When this cut is completed, that crosscut becomes the LOXC and the path of least resistance for the ventilation air sweeping the section face area. If the hole-through is made with the CM cutting in the opposite direction of the ventilation air flow, all of the dust surrounding the CM cutter head envelopes the area occupied by CM and face haulage unit operators. Starting and completing a crosscut head-on in the direction of ventilation air flow allows both operators to remain in fresh air where dust levels are lower.

Follow a hole-through cut in a crosscut with a cut in the entry from which the crosscut was accessed. One of the delays encountered in the mining cycle is handling cable as the CM changes places. Cable handling is avoided if the CM can make two cuts during one trip into an entry as shown in Figure 3.3. Doing so is called “double cutting” and it substantially reduces cable handling and travel time to the next cut. Double cutting is not an option if the crosscut is short enough that it can be mined in one cut because immediately following completion of that cut, the crosscut has not yet been supported by the roof bolter and neither the CM nor any face haulage unit operators can travel past or work inby any area considered unsupported.

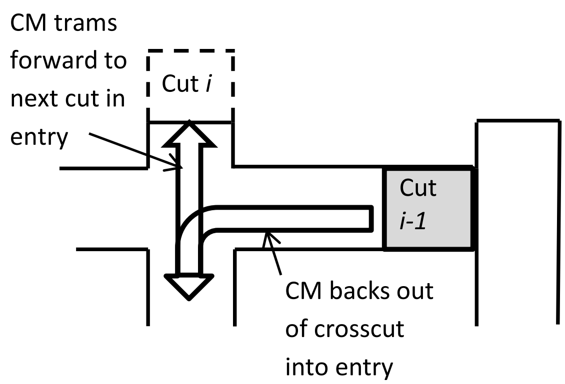

Figure 3.3. Double cutting illustration.

Keep a buffer of one or more cuts between the cut most recently mined by the CM and the cut most recently supported by the roof bolter. The roof bolter can normally support a cut faster than the CM can mine it; however, roof conditions, stocking the roof bolter, and mechanical problems can create unexpected roof bolting delays. A buffer minimizes any influence these delays might have on determining an optimum mining sequence. Furthermore, this buffer helps to reduce congestion in the face area, as illustrated by the example of a 2-entry section. When the CM completes a cut in one entry, it has to wait for the roof bolter to travel through the crosscut between entries before it can travel through that crosscut to the next cut. A buffer serves to minimize the number of instances where this type of congestion occurs on a mining section

Repeat mining sequences every crosscut. Uniformly advancing the section face area reduces CM tram distances, allows for easier ventilation of the section face area, improves face haulage unit efficiency by minimizing travel distance to the dump point, and allows for more efficient scheduling of conveyor belt and power supply extensions. But perhaps the most significant advantage of uniform face advance is having a repeatable mining sequence that becomes automatic for the mining crew and eliminates "thinking" delays that can occur when the section foreman is unsure of the best option (Suboleski, 2011).

3.3 Optimal Value Function

Every DP model is an algorithm built upon an optimal value function, which seeks to optimize (maximize or minimize) a particular parameter. In determining an optimum mining sequence, cycle time, production, or feet of advance may be the target of the optimization algorithm. Minimizing cycle time was selected over maximizing production for the optimal value function as cycle times are based on time study data, which is more readily available than production data for individual mining units. Accurate production data from belt scales and storage facilities are usually for the entire mine and not for individual sections. At the section level, the production process is measured in terms of time increments for completing each of the various steps in the production process. Thus, the basic optimal value function is defined as:

$f_i(X)$ = minimum cut-cycle time that results from following an optimal policy for stage i given state X.

Cut-cycle time is composed of two elements and defined as follows:

$$CCT_i(X) = MOVE^{i}_{i-1}(X) + PROD_i(X), \tag{3.1}$$

where $CCT_i(X)$ = cut-cycle time for stage i given state X,

$MOVE^{i}_{i-1}(X)$ = place change element from stage i-1 to stage i in state X,

and $PROD_i(X)$ = production element for stage i given state X.

The two elements in Equation 3.1 are described in detail below.

Place change element. The place change element of the cut cycle is the time required to move the CM from cut to cut. The basic components in place change element are the tram distance between cuts and CM tram speed, which are defined as follows:

$TD^{i}_{i-1}(X)$ = tram distance (in feet) from stage *i-1* to stage *i* given state X,

and SPD_{CM} = CM tram speed (in feet per minute).

Tram distance is measured from the face or deepest point of the last completed cut (stage *i-1*) to the face of each feasible cut (stage *i*) in state X as shown in Figure 3.4. CM tram speed is measured in time studies but in the absence of data measured in the mine, speeds provided by the equipment vendor in their service literature may be used until such data are available.

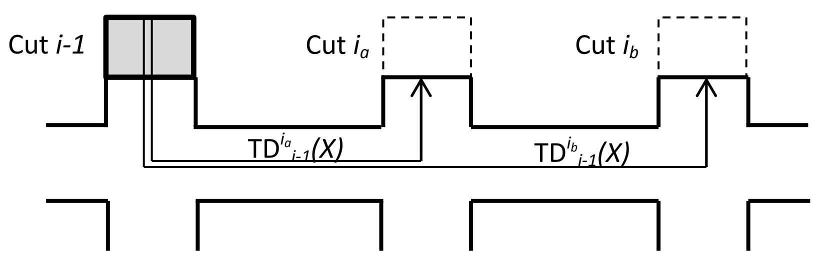

Figure 3.4. Diagram illustrating tram distance measurement.

The basic time value for the place change element is $[TD^{i}_{i-1}(X) / SPD_{CM}]$. Because negotiating corners, handling cable, and road conditions introduce delays into the place change function, the basic time for the place change element must be adjusted to account for these delays.

Cornering adjustment factor. The place change time element is adjusted for extra time required to negotiate corners by adding a cornering factor, $TCOR^{i}_{i-1}(X)$, which is a function of the number of corners negotiated moving from stage *i-1* to stage *i* given state X. Three typical cornering maneuvers are shown in Figure 3.5.

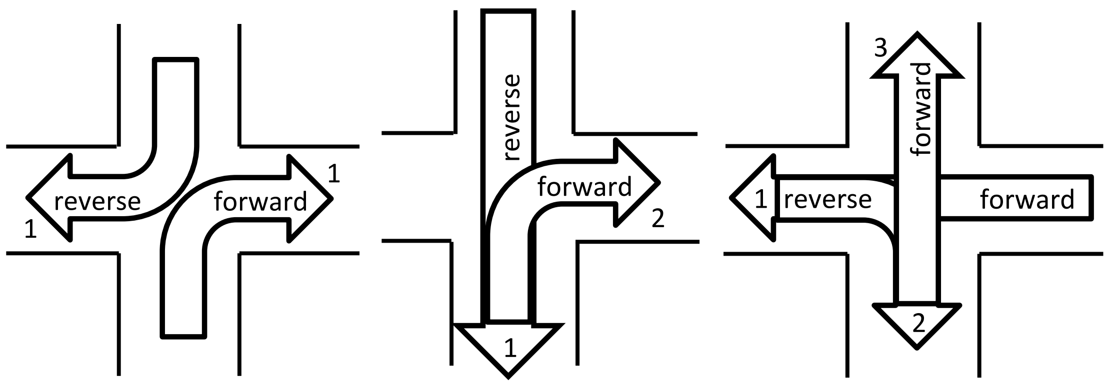

Figure 3.5. Illustration of cornering maneuvers with 0-point turn (left), 1-point turn (center), and 2-point turn (right).

Each of the above scenarios illustrates the negotiation of one corner, which adds an additional amount of time, $CORTM_{CM}$, to the basic time for tramming the CM through the distance moved in turning the corner.

In addition to negotiating the corner, the middle and right illustrations in Figure 3.5 identify additional legs involved in making 1-point and 2-point turns, respectively. The number of points in a turn is the number of times the CM reverses directions. Reversing directions adds another time segment, $DRCHTM_{CM}$, to the basic time because the CM must tram beyond the intersection being turned in and then back to it. The cornering adjustment factor is thus defined as follows:

$$TCOR^{i}_{i-1}(X) = [NUMCOR^{i}_{i-1}(X) * CORTM_{CM}] + [NUMDRCH^{i}_{i-1}(X) * DRCHTM_{CM}] \tag{3.2}$$

where $NUMCOR^{i}_{i-1}(X)$ = number of corners negotiated moving from stage i-1 to stage i given state X,

$CORTM_{CM}$ = extra time required for CM to negotiate a corner,

$NUMDRCH^{i}_{i-1}(X)$ = number of times CM reverses direction while negotiating corners in moving from stage i-1 to stage i given state X,

and $DRCHTM_{CM}$ = extra time required when CM reverses direction.

Values for $CORTM_{CM}$ and $DRCHTM_{CM}$ are determined from time studies. For functional testing of the model, values of 0.5 and 0.25 minutes, respectively, were used.

Cable handling adjustment factor. As the CM changes places, several stops are made to handle the cable requiring a cable handling adjustment, $CH^{i}_{i-1}(X)$, which is extra time that must be added to the basic time. The three most common types of cable handling are described as follows:

$HANGTM_{CM}$: When CM cable is hung against the roof to allow the CM or haulage units to pass through where the CM cable is routed (see left side illustration of Figure 3.6).

$HOOKTM_{CM}$: When the CM stops so a loop (or loops) of CM cable can be placed on or removed from hooks or bits on the CM (see middle illustration of Figure 3.6).

$HANDTM_{CM}$: When CM cable that has been unhooked from the CM is picked up off the ground and moved to a position against the rib so an entry or crosscut is unrestricted for travel by either the CM or haulage units (see right side illustration of Figure 3.6).

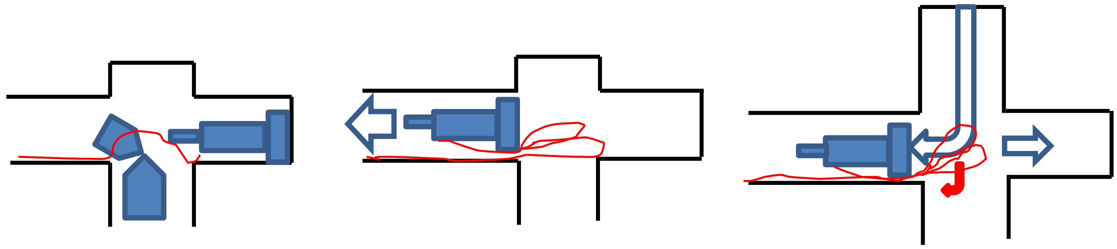

Figure 3.6. Cable handling scenarios: hanging (left), hooking on CM (middle), and handling (right).

Value for $HANGTM_{CM}$, $HOOKTM_{CM}$, and $HANDTM_{CM}$ are determined from time studies. For functional testing of the model, values of 2.0, 0.25, and 1.0 minutes, respectively, were used.

The cable handling adjustment factor is defined as follows:

$$CH^{i}_{i\text{-}1}(X) = [NUMHANG^{i}_{i\text{-}1}(X) * HANGTM_{CM}] + [NUMHOOK^{i}_{i\text{-}1}(X) * HOOKTM_{CM}] + [NUMHAND^{i}_{i\text{-}1}(X) * HANDTM_{CM}] \quad (3.3)$$

where $NUMHANG^{i}_{i\text{-}1}(X)$ = number of times the CM cable must be hung or unhung while moving from stage *i*-1 to stage *i* given state X,

$NUMHOOK^{i}_{i\text{-}1}(X)$ = number of times the CM stops while moving from stage *i*-1 to stage *i* given state X so that a loop (or loops) of CM cable can be hooked on to or unhooked from the CM,

and $NUMHAND^{i}_{i\text{-}1}(X)$ = number of times the CM stops while moving from stage *i*-1 to stage *i* given state X so that CM cable on the ground can be moved out of an entry or crosscut by operators who pick it up and handle it.

To minimize cable handling, the CM generally moves across the mining section away from the location where slack cable is stored, which is typically on the side of the section where ventilation air reaches the face area (intake), until it eventually reaches the outer edge of the mining section, which is typically where ventilation air leaves the face area (return). Then, extra cable handling is required to re-route the cable back to the intake or the entry or crosscut where slack cable is stored. Because re-routing the CM cable is necessary when the outermost return entry has been mined to a depth aligning it with the rest of the mining section, and the previously defined cable handling adjustments would penalize such a move to the extent that it would never be selected in the optimal value function, no cable handling adjustment is factored into the path from the outermost entry to the entry adjacent to where slack cable is stored. In other words,

$CH^{i}_{i-1}(X)$ = 0.0 minutes when re-routing cable back to the stage *i* that is the intake or entry where slack cable is stored after completing a stage *i-1* that squares the outermost return entry with the rest of the section.

This allows for periodic rerouting of the miner cable at certain identifiable points in the mining sequence without incurring the normal cable handling delay that would always penalize such a move under any other circumstances than those specified.

Road condition adjustment factor. The basic place change time element is the amount of time required to tram at normal speed from stage *i-1* to stage *i* given state X. The road condition adjustment factor, $RDCON^{i}_{i-1}(X)$, is the additional time needed to pass through any areas with poor road conditions while moving from stage *i-1* to stage *i* given state X. Poor road conditions may be caused by water accumulations, soft bottoms due to floor heave, and bad top where fallen roof rock is obstructing the roadway.

To determine the road condition adjustment factor, three variables, along with their associated effects on CM tram speed used in functional testing of the model, are defined. Actual effects on CM tram speed would be measured in time studies.

$WATERSPD_{CM}$ = CM speed when traveling through water,
= $SPD_{CM}/2.5$, i.e. speed through water is 40% of normal;

$SOFTBTSPD_{CM}$ = CM speed when traveling over soft roads (bottoms),
= $SPD_{CM}/2.0$, i.e. speed over soft roads is 50% of normal;

$BDTOPSPD_{CM}$	=	CM speed when traveling through areas of fallen roof rock,
	=	$SPD_{CM}/1.5$, i.e. speed through fallen rock is 67% of normal;
$WATER^{i}_{i-1}(X)$	=	length of water hazard(s) encountered moving from stage *i-1* to stage *i* given state X;
$SOFTBT^{i}_{i-1}(X)$	=	length of soft roadway hazard(s) encountered moving from stage *i-1* to stage *i* given state X;
$BDTOP^{i}_{i-1}(X)$	=	length of bad top hazard(s) encountered moving from stage *i-1* to stage *i* given state X.

The method for determining the road condition adjustment factor is the same for each detrimental road condition and is described in the example shown in Figure 3.7, which considers a place change from stage *i-1* to stage *i* through two water hazards of length d_{1w} and d_{2w}.

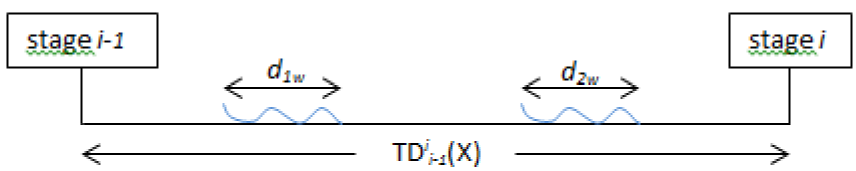

Figure 3.7. Diagram of place change with two water hazards.

The total length of water hazards encountered moving from stage *i-1* to stage *i* is d_{1w} + d_{2w}. The time required to go through these hazards is $(d_{1w} + d_{2w}) / WATERSPD_{CM}$. The time required to go the same distance without water is $(d_{1w} + d_{2w}) / SPD_{CM}$. The road condition adjustment factor is the difference between these two times. Since $(d_{1w} + d_{2w}) = WATER^{i}_{i-1}(X)$, then $RDCON^{i}_{i-1}(X)$ for this example scenario is given as:

$$RDCON^{i}_{i-1}(X) = [WATER^{i}_{i-1}(X)/WATERSPD_{CM}] - [WATER^{i}_{i-1}(X)/SPD_{CM}].$$

Similarly, for soft roads, the road condition adjustment factor is:

$$RDCON^{i}_{i-1}(X) = [SOFTBT^{i}_{i-1}(X)/SOFTBTSPD_{CM}] - [SOFTBT^{i}_{i-1}(X)/SPD_{CM}];$$

and for areas of bad top where roof rock has fallen, it is:

$$RDCON^{i}_{i-1}(X) = [BDTOP^{i}_{i-1}(X)/BDTOPSPD_{CM}] - [BDTOP^{i}_{i-1}(X)/SPD_{CM}].$$

Thus, the complete road condition adjustment factor is defined as follows:

$$\begin{aligned} RDCON^{i}_{i-1}(X) = {} & [WATER^{i}_{i-1}(X)/WATERSPD_{CM}] - [WATER^{i}_{i-1}(X)/SPD_{CM}] \\ & + [SOFTBT^{i}_{i-1}(X)/SOFTBTSPD_{CM}] - [SOFTBT^{i}_{i-1}(X)/SPD_{CM}] \\ & + [BDTOP^{i}_{i-1}(X) / BDTOPSPD_{CM}] - [BDTOP^{i}_{i-1}(X) / SPD_{CM}]. \end{aligned} \tag{3.4}$$

Integrating all three adjustment factors with the basic time for the place change element of $CCT_i(X)$ gives the following:

$$MOVE^{i}_{i-1}(X)=[TD^{i}_{i-1}(X)/SPD_{CM}]+TCOR^{i}_{i-1}(X)+CH^{i}_{i-1}(X)+RDCON^{i}_{i-1}(X). \tag{3.5}$$

Production element. The production element of cut-cycle time has two components – change-out time and loading time – and is expressed mathematically as:

$$PROD_i(X) = COT_i(X) + LT_i(X) \tag{3.6}$$

where $COT_i(X)$ = change out time for stage *i* given state X,

and $LT_i(X)$ = loading time for stage *i* given state X.

Change-out time. Change-out time (COT) is the time required to move loaded cars away from the face and empty cars to the face. Because entries and crosscuts are not wide enough to allow haulage units to pass, loaded haulage units must exit the face area before empty haulage units can enter the face area. The intersection where loaded and empty haulage units cross paths is called the change-out point (COP). It is illustrated in Figure 3.8. The distance between the face and the COP is change-out distance (COD), as shown in Figure 3.8. This is not a true COD as haulage units travel to and from the back of the CM, not the face; however the back of the CM is a moving point. Production models account for haulage unit and CM lengths in calculating adjusted change-out and haul distances. That level of accuracy is beyond the scope of this cut sequence algorithm that would essentially be a subroutine of the broader production model.

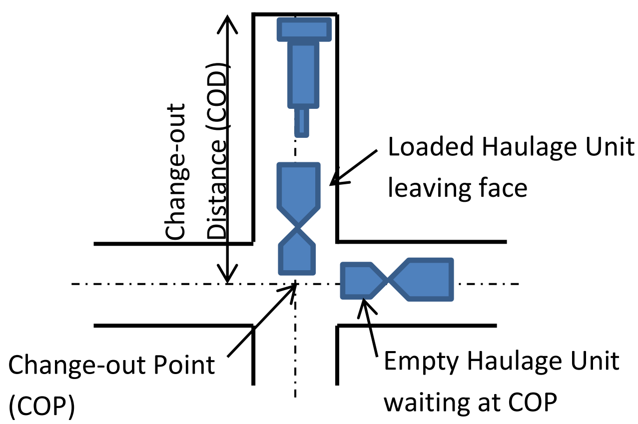

Figure 3.8. Illustration of change-out point and change-out distance.

COD is a measurable parameter based on mining geometry. Haulage unit tram speed provided in equipment specifications by vendors may be used until time study data are collected. Total COT for a cut depends on the number of haulage units being operated, their capacity, and cut volume. Two conditional adjustments are also required.

Mathematically, the basic incremental value for COT is expressed as [$COD_i(X)$ / SPD_{HU}]. Total unadjusted COT is found as follows:

$$COT_i(X) = \{[COD_i(X)/SPD_{HU}]*2*TRIPS_i(X)\} + [SWIN* TRIPS_i(X)] \quad (3.7)$$

where $COD_i(X)$ = change-out distance for stage *i* given state X (in feet).

SPD_{HU} = haulage unit (HU) tram speed (in feet per minute),

$TRIPS_i(X)$ = number of haulage unit loads in stage *i* given state X

and SWIN = time required to turn haulage unit at COP (in minutes).

The number of haulage unit trips made for a given cut is a function of cut volume, haulage unit capacity, and how much of the haulage unit capacity is utilized. Each trip requires traversing the distance between change-out point and face twice, once coming in empty and once going out loaded. Theoretically, the first haulage unit trip in and the last haulage unit trip out should not be counted as they coincide with the CM tramming to and from the face, respectively, which is part of the CCT place change element. This only happens in very efficient operations. Mathematically, $TRIPS_i(X)$ is expressed as follows:

$$TRIPS_i(X) = \textbf{ROUNDUP}\ [CUTVOL_i(X) / (PLD_{HU} * FILL)], \quad (3.8)$$

where **ROUNDUP** indicates the bracketed quantity is rounded up to the next integer;

$CUTVOL_i(X)$= volume of coal in stage *i* given state X (in tons),

= $DEPTH_i(X) * WIDTH_i(X) * HEIGHT_i(X) * RCDEN_i(X)$;

$DEPTH_i(X)$ = depth of stage *i* given state X (in feet);

$WIDTH_i(X)$ = width of stage *i* given state X (in feet);

$HEIGHT_i(X)$ = mined height of stage *i* given state X (in feet);

$RCDEN_i(X)$ = unit weight of raw coal for stage *i* given state X (in tons/ft^3);

PLD_{HU} = haulage unit capacity (in tons);

and FILL = percentage of haulage unit capacity utilized (determined in time studies).

While width is typically constant for all cuts in a given mine, mined height and raw coal unit weight vary with seam thickness. Seam thickness may be relatively constant across and along a mining section or it may vary widely from cut to cut. There are two options to consider for mined height. It can be maintained at a constant level in which case the amount of out-of-seam dilution (OSD) varies as seam height varies, or it can be the seam height plus a fixed percent of OSD. The option selected depends on the mining company's operating philosophy and preference. For both options, mined height is expressed as follows:

$$\text{HEIGHT}_i(\text{X}) = \text{SEAM}_i(\text{X}) * [1 + \text{OSD}_i(\text{X})], \tag{3.9}$$

where $\text{SEAM}_i(\text{X})$ = seam thickness of stage *i* given state X (in feet),

and $\text{OSD}_i(\text{X})$ = OSD mined in stage *i* given state X (% of seam thickness).

Hence, $\text{RCDEN}_i(\text{X})$ is determined as follows:

$$\text{RCDEN}_i(\text{X}) = \frac{\{\text{SEAM}i(\text{X}) * [\text{COALDEN} + \text{OSD}i(\text{X}) * \text{OSDDEN}]\}}{\text{HEIGHT}i(\text{X})} \tag{3.10}$$

where COALDEN = unit weight of coal

and OSDDEN = unit weight of OSD.

Values of 83 and 144 pounds per cubic foot (pcf) are used for COALDEN and OSDDEN, respectively, in functional testing of the model.

$\text{WIDTH}_i(\text{X})$, $\text{HEIGHT}_i(\text{X})$, and $\text{RCDEN}_i(\text{X})$ can be combined into a single factor – tons per foot of cut depth or tons per foot of advance, $\text{TFA}_i(\text{X})$, defined as follows:

$$\text{TFA}_i(\text{X}) = \text{WIDTH}_i(\text{X}) * \text{HEIGHT}_i(\text{X}) * \text{RCDEN}_i(\text{X}) \,/\, (2{,}000 \text{ lb/ton}). \tag{3.11}$$

Thus, Equation 3.8 becomes:

$$TRIPS_i(X) = \mathbf{ROUNDUP}\ [TFA_i(X) * DEPTH_i(X) / (PLD_{HU} * FILL)]. \qquad (3.12)$$

Change-out condition adjustment factor. The presence of line curtain, corners, and/or poor road conditions in the change-out path prevents the haulage unit operator from negotiating the change-out path at normal speed. Curtain and corners are a regular part of the mining process as shown in Figure 3.9. Time studies conducted by the author measuring speeds for empty and loaded haulage units operating between the COP and the face and between the COP and the dump point show that haulage unit tram speed can be reduced by as much as 50% when the change-out path is restricted by curtain or includes corners. Poor road conditions, while not a normal occurrence, have the same effect as curtain or corners.

Figure 3.9. Normal change-out conditions showing haulage restrictions with line curtain (left) and corner (right).

The change-out condition adjustment factor, $COCON_i(X)$, is defined as:

$COCON_i(X)$ = 1.0 if change-out path is not obstructed by line curtain, corners that must be negotiated, or poor road conditions,

= 1.5 if one of either line curtain, corners that must be negotiated, or poor road conditions obstruct the change-out path,

= 2.0 if two of either line curtain, corners that must be negotiated, or poor road conditions obstruct the change-out path.

To include this adjustment, haulage unit tram speed, SPD_{HU}, in Equation (3.7) is divided by $COCON_i(X)$, giving the following expression for $COT_i(X)$:

$$COT_i(X) = \{COD_i(X) / [SPD_{HU} / COCON_i(X)]\}*2*TRIPS_i(X) + [SWIN* TRIPS_i(X)] \tag{3.13}$$

which can be simplified to:

$$COT_i(X) = (2/ SPD_{HU})*[COD_i(X)*COCON_i(X)*TRIPS_i(X)] + [SWIN* TRIPS_i(X)]. \tag{3.14}$$

Some occasions require averaging two values for $COCON_i(X)$. For example, during a cut that completes a crosscut curtain is obstructing the change-out path, but the moment the CM holes through, that curtain comes down. Thus, half of the cut has $COCON_i(X)$ = 1.5 or 2.0 and the other half of the cut has $COCON_i(X)$ = 1.0 or 1.5, depending on whether negotiating corners is also involved. For a cut that completes a crosscut that was turned, the haulage unit must negotiate a corner and stay clear of line curtain for most of a long change-out path until the hole is made and then it only has to deal with the corner. $COCON_i(X)$ for this scenario is (2.0 + 1.5)/2 = 1.75. For a cut that completes a crosscut that was started head-on, the haulage unit typically has straight-line access to the CM at the face and only has to deal with line curtain till the hole is made. $COCON_i(X)$ for this scenario is (1.5 + 1.0)/2 = 1.25.

Wait-no-car adjustment factor. COT does not include any time that a haulage unit spends waiting at the COP, nor does it include any time that the CM spends waiting because there is not an empty haulage unit at the COP when a loaded haulage unit passes that point after leaving the face area. The latter condition is, however, accounted for, as it must be, with a wait-no-car adjustment factor. $\mathrm{WAITNOCAR}_i(\mathrm{X})$ is the time per occurrence that a CM is not loading because there is no haulage unit at the COP when a loaded haulage unit leaving the face passes that point. The equation for determining $\mathrm{WAITNOCAR}_i(\mathrm{X})$ is taken from the SSP Model described in Chapter 2, and is written as follows:

$$\begin{aligned}\mathrm{WAITNOCAR}_i(\mathrm{X}) = {} & \mathrm{HD}_i(\mathrm{X})/\mathrm{SPD}_{\mathrm{HU}}(CP\text{-}D) + \mathrm{HD}_i(\mathrm{X})/\mathrm{SPD}_{\mathrm{HU}}(D\text{-}CP)\\ & + \mathrm{PLD}_{\mathrm{HU}}/\mathrm{DR}_{\mathrm{HU}} + \mathrm{SWIN}\\ & - [\mathrm{NCARS}_i(\mathrm{X}) - 1] * [\mathrm{PLD}_{\mathrm{HU}}/\mathrm{LR}_{\mathrm{CM}} + \mathrm{COD}_i(\mathrm{X})/\mathrm{SPD}_{\mathrm{HU}}(CP\text{-}F)\\ & + \mathrm{COD}_i(\mathrm{X})/\mathrm{SPD}_{\mathrm{HU}}(F\text{-}CP) + \mathrm{SWOUT}],\end{aligned} \tag{3.15}$$

where CP = change-out point,

D = dump point,

F = face,

$\mathrm{HD}_i(\mathrm{X})$ = distance from the change-out point for stage *i* given state X to the dump point (in feet),

$\mathrm{NCARS}_i(\mathrm{X})$ = number of haulage units in operation for stage *i* given state X,

SWIN = time required to turn haulage unit at CP (in minutes),

SWOUT = time required to turn haulage unit other than at CP (in minutes),

$\mathrm{DR}_{\mathrm{HU}}$ = haulage unit dumping rate (in tons per minute),

and $\mathrm{LR}_{\mathrm{CM}}$ = CM loading rate (in tons per minute).

Data collected in time studies are used to designate values for all of the above variables.

For simplicity, uniform SPD_{HU} for all haulage segments in Equation 3.15 (*CP-D*, *D-CP*, *CP-F*, and *F-CP*) is assumed. Additionally, SWIN = SWOUT is assumed. These assumptions have been validated with time study data collected by the author at several mines. Based on these assumptions, Equation 3.15 can be simplified to Equation 3.16:

$$\begin{aligned} \mathrm{WAITNOCAR}_i(\mathrm{X}) = {} & 2*\mathrm{HD}_i(\mathrm{X})/\mathrm{SPD}_{\mathrm{HU}} - [\mathrm{NCARS}_i(\mathrm{X}) - 1]*2*\mathrm{COD}_i(\mathrm{X})/\mathrm{SPD}_{\mathrm{HU}} \\ & + \mathrm{PLD}/\mathrm{DR}_{\mathrm{HU}}*[3 - 2*\mathrm{NCARS}_i(\mathrm{X})] \\ & + \mathrm{SWIN}*[2 - \mathrm{NCARS}_i(\mathrm{X})]. \end{aligned} \tag{3.16}$$

Since $\mathrm{WAITNOCAR}_i(\mathrm{X})$ is per occurrence, it must be multiplied by the number of occurrences to obtain a total wait-no-car adjustment factor. Thus, the wait-no-car adjustment factor, $\mathrm{WOC}_i(\mathrm{X})$, is defined as follows:

$$\begin{aligned} \mathrm{WOC}_i(\mathrm{X}) = {} & \boldsymbol{\{}(2/\mathrm{SPD}_{\mathrm{HU}})*\{\mathrm{HD}_i(\mathrm{X}) - [\mathrm{NCARS}_i(\mathrm{X}) - 1]*\mathrm{COD}_i(\mathrm{X})\} \\ & + \{\mathrm{PLD}/\mathrm{DR}_{\mathrm{HU}}*[3 - 2*\mathrm{NCARS}_i(\mathrm{X})]\} \\ & + \{\mathrm{SWIN}*[2 - \mathrm{NCARS}_i(\mathrm{X})]\}\boldsymbol{\}} \\ & *\mathbf{TRUNC}[\mathrm{TRIPS}_i(\mathrm{X})/\mathrm{NCARS}]. \end{aligned} \tag{3.17}$$

where **TRUNC** indicates the bracketed quantity is truncated to an integer by removing the decimal or fractional part of the number.

Incorporating the wait-no-car adjustment factor, $\mathrm{WOC}_i(\mathrm{X})$, into Equation 3.14 gives the following mathematical expression for the change-out time element of the production component of CCT found in Equation 3.6:

$$\begin{aligned} \mathrm{COT}_i(\mathrm{X}) = {} & (2/\ \mathrm{SPD}_{\mathrm{HU}})*[\mathrm{COD}_i(\mathrm{X})*\mathrm{COCON}_i(\mathrm{X})*\mathrm{TRIPS}_i(\mathrm{X})] \\ & + [\mathrm{SWIN}*\ \mathrm{TRIPS}_i(\mathrm{X})] + \mathrm{WOC}_i(\mathrm{X}). \end{aligned} \tag{3.18}$$

Loading time. The second element of the production component is a function of the CM loading rate, as measured in time studies, and the cut volume of stage *i*, which is expressed mathematically as follows:

$$LT_i(X) = CUTVOL_i(X) / LR_{CM}. \qquad (3.19)$$

$CUTVOL_i(X)$ was previously defined as $TFA_i(X) * DEPTH_i(X)$, which gives:

$$LT_i(X) = \{[TFA_i(X) * DEPTH_i(X)] / LR_{CM}\}. \qquad (3.20)$$

Most mining geometries are designed to have cuts of uniform depth, which is usually the maximum mining depth allowed in the mine's roof control plan approved by regulatory agencies. Cuts of uniform depth can be achieved for the most part when mining in entries. Where cuts of varying depth occur is during mining of crosscuts, which cannot be started or completed until the two entries connected by the crosscut have been mined deep enough. Crosscuts typically consist of one or more cuts that are shorter than the standard (or maximum) mining depth. These shorter cuts are the first and/or last cut made in a crosscut. To the extent uniform cut depth is achieved, loading time becomes a constant; however, because shorter cuts are associated with completing crosscuts in a timely fashion, either by starting or finishing them, loading time is a crucial part of the basic optimal value function, which cannot be ignored.

Clean-up adjustment factor. At the end of each cut, the CM backs out of the cut, then makes a clean-up pass down both sides gathering loose material in the pan of the CM which is loaded into the last haulage unit for that cut. Since this material will all fit into one car, the last car stays in the face area while the CM repositions from one side to the other during the clean-up process; however, the haulage unit must also reposition. The time required for the haulage unit to reposition is identified as $RESET_{HU}$, which is measured in the time study process. Including this adjustment gives the following equation for $LT_i(X)$:

$$LT_i(X) = \{[TFA_i(X) * DEPTH_i(X)] / LR_{CM}\} + RESET_{HU}. \qquad (3.21)$$

Substituting Equations 3.12, 3.18, and 3.21 into Equation 3.6 gives the following complete mathematical expression for the production element of $CCT_i(X)$ in Equation 3.1:

$$\begin{aligned} PROD_i(X) = {} & \{(2/ SPD_{HU})*[COD_i(X)*COCON_i(X)] + SWIN\} \\ & * \mathbf{ROUNDUP}[TFA_i(X)*DEPTH_i(X)/(PLD_{HU}*FILL)] \qquad (3.22) \\ & + WOC_i(X) + RESET_{HU} \\ & + \{[TFA_i(X) * DEPTH_i(X)] / LR_{CM}\}. \end{aligned}$$

3.4 Recurrence Relation

$CCT_i(X)$ is a DP algorithm. With it defined, a DP OMS model is developed using the recurrence relation or recursion formula, which is:

$$f_i(X) = \text{minimum } [CCT_i(X) + f_{i-1}(X - X_i)] \qquad (3.23)$$

where $f_{i-1}(X - X_i)$ = total cut-cycle time that results from following an optimal policy up to and including stage i-1.

This recursion relation defines a local optimum for a single stage *i* given the existing state X; however, a sequence of local optimums does not guarantee on optimum overall sequence. The goal is to optimize the sequence of cuts for a given period of time such as a shift or a day, or for a complete cut cycle of two or more crosscuts of advance, usually the amount of advance between conveyor belt and power system moves. Therefore, all paths must be considered and the path that yields the overall optimum for the specified time period is the one that is selected.

For a conventional room-and-pillar mining section, the number of possible paths can be very large. To keep the task at a manageable level, only those options within a specified range of the local minimum $CCT_i(X)$ are pursued. This boundary condition is defined by the difference in time, $MOVDIF^{i_a}_{i_b}(X)$, for tramming the CM from stage *i-1* to cuts i_a or i_b at equivalent depths in adjacent entries, as shown in Figure 3.4. Mathematically, this is expressed as:

$$MOVDIF^{i_a}_{i_b}(X) = [TD^{i_a}_{i-1}(X) \;/\; SPD_{CM}] - [TD^{i_b}_{i-1}(X) \;/\; SPD_{CM}] \qquad (3.24)$$

where $TD^{i_a}_{i-1}(X)$ is the distance from cut *i-1* to cut i_a, $TD^{i_b}_{i-1}(X)$ is the distance from cut *i-1* to cut i_b, and SPD_{CM} is the tram speed of the CM. Simply stated, it is the time it takes the CM to tram the entry spacing distance. For example, if entry spacing is 80 feet and the CM tram speed is 30 feet per minute, $MOVDIF^{i_a}_{i_b}(X) = 80/30 = 2.7$ minutes. Thus, any path within 2.7 minutes of the local minimum $CCT_i(X)$ is evaluated further.

3.5 Constraints

Constraints affecting the determination of a CM cut sequence are described in the constraint matrix shown in Table 3.1. A legal or regulatory constraint prohibits mining specific cuts given the existence of certain conditions defined in mining regulations, or if mining a cut would violate approved roof control (bolting) and ventilation plans. For example, a cut in an entry cannot be mined if the previously mined cut in the same entry has not been bolted. An operational constraint is based on those guiding policies and practices listed previously. Operational constraints establish priorities for mining cuts as well as place some restrictions on which feasible cuts can be selected. For example, cuts that start or end a crosscut may be expedited, entry development beyond the point where crosscut development can occur may be restricted, and preference may be given to starting crosscuts head-on and mining in the same direction as ventilation air flow.

Table 3.1. Constraint matrix.

	Bolting	Ventilation
Regulatory	• Travel inby unsupported top is not allowed • Crosscuts may not be turned from certain designated entries • Mined area must be bolted within a specified time period	• Time-weighted average dust exposure limits must not be exceeded • Roof bolter can be downwind of CM for only a designated amount of time per shift
Operational	• Maintain a buffer between bolting and mining functions • Mining cannot occur when bolter blocks haulage path	• Mine crosscuts in the direction of ventilation airflow • Start crosscuts head-on • No turning to the blind side (or scrubber side) of CM

Every mine is required to submit roof control and ventilation plans to MSHA for approval before any mining can take place. MSHA policies are constantly evolving and vary from district to district. In Table 3.1, the second bullet in the regulatory-bolting quadrant and the second bullet in the regulatory-ventilation quadrant are relatively new policies adopted by MSHA. In Illinois, roof control plans are constraining mine operators from turning crosscuts out of entries containing critical infrastructure to minimize intersection diagonals and reduce the probability of roof falls. Nationwide, ventilation plans are restricting the amount of time that roof bolter operators can be downwind of the CM while it is producing coal to reduce their dust exposure levels.

Constraints are used to help insure and maintain a healthy environment and safe working conditions for coal miners. To illustrate how this can be done for the DP OMS model, the optimal value function, $f_i(X)$ = minimum $CCT_i(X)$, is subject to the following two constraints:

$$B_i(X) < PROD_{STD}(X), \text{ and} \tag{3.25}$$

$$V_i(X) \leq DELV, \tag{3.26}$$

where $B_i(X)$ = bolting constraint for stage *i* given state X;

$V_i(X)$ = ventilation constraint for stage *i* given state X;

$PROD_{STD}(X)$ = unadjusted production time for a standard-sized cut in state X,

and DELV = dust exposure limit value, e.g. MSHA standard of 2.0 mg/m^3.

A standard-sized cut for a given state X is defined by the following parameters:

$WIDTH_{STD}(X)$ = width of opening (in feet),

$DEPTH_{STD}(X)$ = depth of cut (in feet),

$SEAM_{STD}(X)$ = seam height (in feet),

and $OSD_{STD}(X)$ = mined OSD thickness (in % seam thickness).

From these parameters, the unadjusted production time for a standard-sized cut given state X is determined based on Equation 3.22 as follows:

$$PROD_{STD}(X) = (2/SPD_{HU})*[TRIPS_{STD}(X)*COD_{MIN}(X)] \\ +\{[TFA_{STD}*DEPTH_{STD}(X)]/LR_{CM}\},$$

where $COD_{MIN}(X)$ = minimum COD for all stages of state X (in feet),

$$= \text{LENGTH}_{CM} + [\text{WIDTH}_{STD}(X) / 2]; \quad (3.27)$$

and LENGTH_{CM} = length of CM (in feet).

Equations 3.11 and 3.12 are used to find $\text{TFA}_{STD}(X)$ and $\text{TRIPS}_{STD}(X)$ as follows:

$$\text{TFA}_{STD} = \text{WIDTH}_{STD}(X)*\{\text{SEAM}_{STD}(X)*[\text{COALDEN}+\text{OSD}_{STD}(X)*\text{OSDDEN}]\}/2{,}000,$$

$$\text{TRIPS}_{STD}(X) = \textbf{ROUNDUP}\ [\text{TFA}_{STD} * \text{DEPTH}_{STD}(X) / \text{PLD}_{HU}].$$

Bolting or roof control constraint. To be considered feasible, a cut must be accessible through previously mined cuts that have been bolted. This constraint prevents the CM operator from working near unsupported roof. This constraint can also be used to maintain a buffer between mining and bolting functions limiting instances where the CM and the roof bolter cross paths. To do so, this constraint prevents the CM from returning to a previously mined cut before mining two or more cuts in other entries and crosscuts. A bolting or roof control constraint designed to maintain a buffer between mining and bolting functions may be defined as follows:

$B_i(X) = \infty$ (i.e. prohibited) if any of the following conditions apply:

- cut *i* is not feasible, i.e., access cuts have not been mined,
- cut *i* is inby stage $i\text{-}1$ and in the same entry or crosscut (see Figure 3.10),
- cut *i* is in an entry and inby the first cut in a crosscut mined from that entry or the last cut in a crosscut mined into that entry, either of which is stage $i\text{-}1$ (see Figure 3.11),
- cut *i* is the first cut in a crosscut that is adjacent to the first or last cut in an adjoining crosscut that is stage $i\text{-}1$ (see Figure 3.12);

otherwise, $B_i(X) = \{[\text{DEPTH}_{i-n}(X) * \text{BOLTTM}] / n\}$, (3.28)

where $\text{DEPTH}_{i-n}(X)$ = depth of stage $i\text{-}n$ through which cut *i* is accessed given state X (in feet);

n = number of stages previous to stage *i*, (i.e., 2, 3, 4, etc.);

and BOLTTM = bolting time per foot of cut depth.

Time studies are used to determine BOLTTM by measuring the amount of time required to install one row of bolts and move to the next row. Consider an example where BOLTTM of 1.625 minutes is observed. Assuming 32-ft depth for all cuts gives [$DEPTH_{i-n}$(X) * BOLTTM] = 52 minutes, leading to the following bolting constraint values:

B_i(X) = 26.0 minutes if cut *i* is accessed through stage *i-2* of standard cut depth,

B_i(X) = 17.3 minutes if cut *i* is accessed through stage *i-3* of standard cut depth, and

B_i(X) = 13.0 minutes if cut *i* is accessed through stage *i-4* of standard cut depth.

These constraint values are compared to the unadjusted production time for a cut of standard depth, which, for the ongoing example assuming $SEAM_{STD}$ = 7.0 feet, OSD_{STD} = 10%, and haulage units with 300 feet per minute tram speed and 10-ton capacity requiring 45 seconds for a 32-foot long CM to load, is determined as follows:

$$\begin{aligned} TFA_{STD} &= WIDTH_{STD}(X)*\{SEAM_{STD}(X) \\ &\quad *[COALDEN+OSD_{STD}(X)*OSDDEN]\}/2{,}000, \\ &= 18*\{7*[83+0.10*144]\}/2000, \\ &= 6.14 \text{ tons/foot of advance;} \end{aligned}$$

$$\begin{aligned} TRIPS_{STD}(X) &= \textbf{ROUNDUP}\ [TFA_{STD} * DEPTH_{STD}(X) / PLD_{HU}], \\ &= \textbf{ROUNDUP}\ [6.14*32/10], \\ &= 20 \text{ trips;} \end{aligned}$$

and

$$\begin{aligned} PROD_{STD}(X) &= [COD_{MIN}(X) / SPD_{HU}]*\{2 * [TRIPS_{STD}(X) - 1]\} \\ &\quad + \{[TFA_{STD} * DEPTH_{STD}(X)] / LR_{CM}\}, \\ &= \{[32+(18/2)] / 300\}*\{2*(20 - 1)\} + [(6.14*32)/(10/0.75)], \\ &= 21.2 \text{ minutes.} \end{aligned}$$

Since B_i(X) is less than $PROD_{STD}$(X) for all cases except n = 2, the CM is not constrained from tramming to stage *i* accessed through stage *i-n*, where $n > 2$.

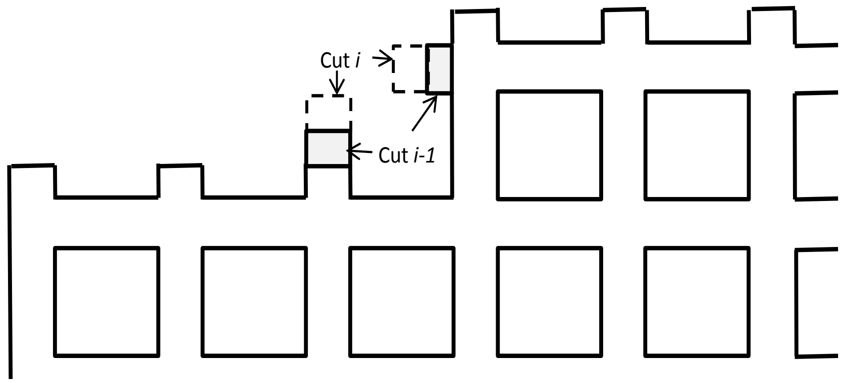

Figure 3.10. Cut *i* inby stage *i-1* and in the same entry or crosscut.

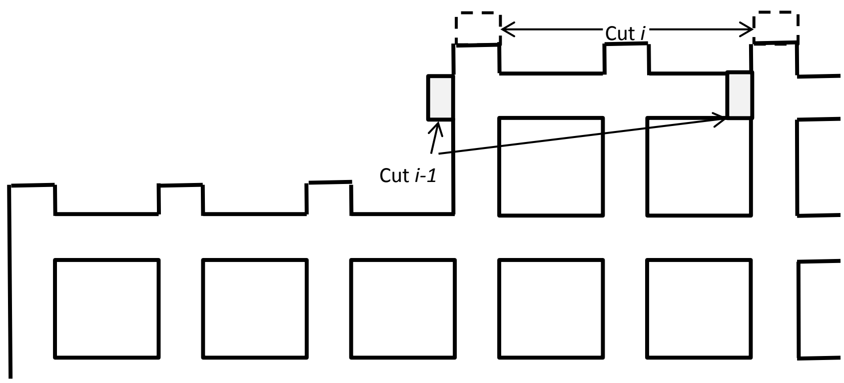

Figure 3.11. Cut *i* in an entry inby stage *i-1*, which is the first or last cut in a crosscut mined from or into that entry, respectively.

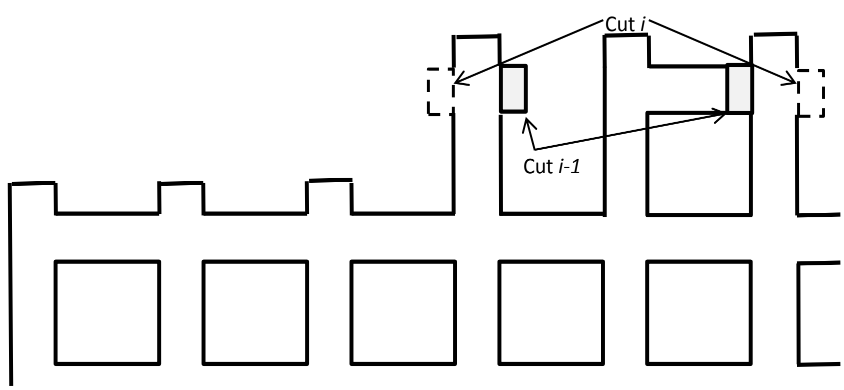

Figure 3.12. Cut *i* is the first cut in a crosscut adjacent to an adjoining crosscut with stage *i-1* as first or last cut.

Ventilation or dust control constraint. Federal regulations limit miners' exposure to respirable dust. The standard currently enforced is a time-weighted average of 2.0 mg/m^3, but the Mine Safety and Health Administration (MSHA) has proposed lowering that to 1.0 mg/m^3. The ventilation or dust control constraint is designed to protect the CM operator and others from exposures to respirable dust that would be out of compliance. This may be accomplished by estimating a production-weighted average of cumulative dust exposure at each stage of a cut sequence and constraining the CM from pursuing a sequence that might result in over exposure at any point in the sequence.

As mining progresses in a room-and-pillar mechanized mining unit (MMU), a number of different types of cuts will be made as defined by CM positioning at the start and/or end of the cut. A summary of primary cut types is given in Table 3.2. A dust exposure level is associated with each cut type based on the difficulty of getting intake air to the working face.

Table 3.2. Primary cut types with their associated relative level of dust exposure.

Description of Cut Type	**Dust Exposure Level**
Entry with initial face < $LENGTH_{CM}$ from LOXC	low
Entry with $LENGTH_{CM}$ < initial face < 2*$LENGTH_{CM}$ from LOXC	medium
Entry with 2*$LENGTH_{CM}$ < initial face < 3*$LENGTH_{CM}$ from LOXC	high
Entry with 3*$LENGTH_{CM}$ < initial face from LOXC	very high
1st cut in crosscut turned to operator side (side opposite CM scrubber)	high
1st cut in crosscut turned to blind side (same side as CM scrubber)	very high
1st cut in crosscut started head-on and mined with the air	low
1st cut in crosscut started head-on and mined against the air	medium
Cut in crosscut (not 1st or last) mined with the air	medium
Cut in crosscut (not 1st or last) mined against the air	medium
Cut that completes crosscut mined with the air	low
Cut that completes crosscut mined against the air	high

Dust sampling programs have been and continue to be conducted by researchers in the field of dust control to quantify the dust exposure of CM operators, haulage unit operators, roof bolters, and other personnel for each cut type as well as dust levels in the return entry of each MMU. (Chugh, 2006a; Chugh 2012). Work is ongoing to develop a dust exposure index that may be factored into ventilation or dust control constraints on the DP OMS model. With that concept in mind, a ventilation or dust control constraint may be defined as follows:

$V_i(X) = \infty$ if situations prohibited by ventilation and roof control plans exist, such as:

- cut *i* turns a crosscut from the belt, power, or travel entries,
- cut *i* turns a crosscut in a row of crosscuts where two have already been turned;

otherwise,

$$V_i(X) = [{}_{c=i-n}\Sigma^{i}DUST_c(X) * CUTVOL_c(X)] / [{}_{c=i-n}\Sigma^{i}CUTVOL_c(X)], \qquad (3.29)$$

where $DUST_c(X)$ = dust exposure index value for cut *c* given state X (in mg/m^3),

and *n* = the number of cuts made since the beginning of the sequence.

Standard dust sampling and monitoring processes are used to determine a time-weighted average dust exposure index, $DUST_c(X)$, corresponding to the dust exposure levels identified in Table 3.2. Continuing with the same example established for the bolting constraint and assuming constant seam thickness and 10% OSD for all cuts, hypothetical dust exposure index values corresponding to low, medium, high, and very high levels of dust exposure are found to be 0.5, 2.0, 3.5, and 5.0 mg/m^3, respectively. For the sequence shown in Figure 3.13, Cuts 1-3 have low levels of dust exposure, Cuts 4-6 have medium levels of dust exposure, and Cuts 7-9 have high levels of dust exposure.

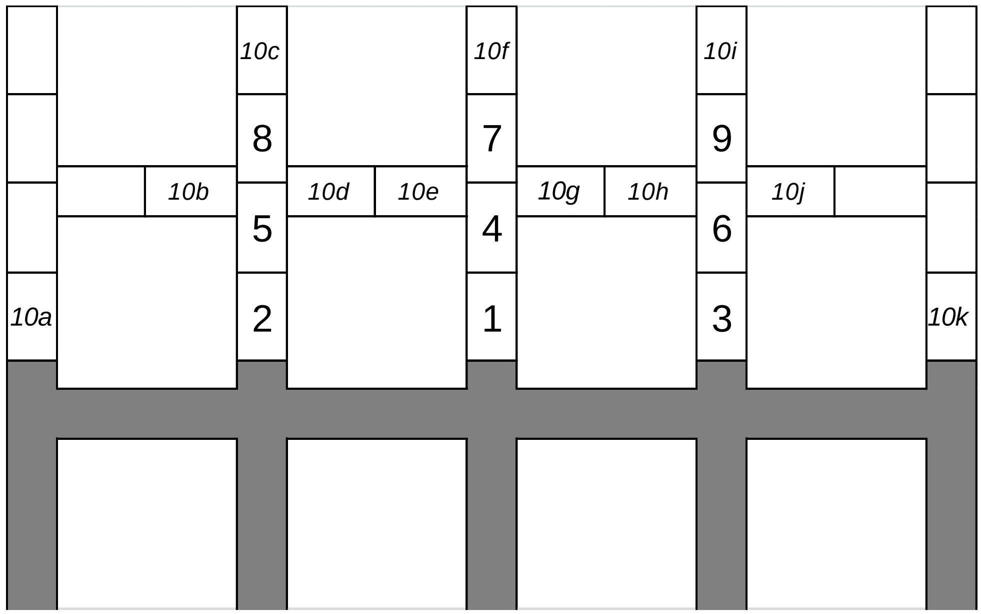

Figure 3.13. Example cut sequence.

Using Equation 3.29, $V_i(X)$ is determined for each stage as follows:

$$V_1(X) = [{}_{c=1-0}\Sigma^{1} DUST_c(X) * CUTVOL_c(X)] / [{}_{c=1-0}\Sigma^{1} CUTVOL_c(X)],$$
$$= [0.5*(32*6.14)] / (32*6.14)$$
$$= 0.5;$$

$$V_2(X) = [{}_{c=2-1}\Sigma^{2} DUST_c(X) * CUTVOL_c(X)] / [{}_{c=2-1}\Sigma^{2} CUTVOL_c(X)],$$
$$= [2*0.5*(32*6.14)] / [2*(32*6.14)],$$
$$= 0.5;$$

$$V_3(X) = [{}_{c=3-2}\Sigma^{3} DUST_c(X) * CUTVOL_c(X)] / [{}_{c=3-2}\Sigma^{3} CUTVOL_c(X)],$$
$$= [3*0.5*(32*6.14)] / [3*(32*6.14)],$$
$$= 0.5;$$

$$V_4(X) = [{}_{c=4-3}\Sigma^{4} DUST_c(X) * CUTVOL_c(X)] / [{}_{c=4-3}\Sigma^{4} CUTVOL_c(X)],$$
$$= [3*0.5*(32*6.14) + 2.0*(32*6.14)] / [4*(32*6.14)],$$
$$= 0.875;$$

$$V_5(X) = [{}_{c=5-4}\Sigma^{5} DUST_c(X) * CUTVOL_c(X)] / [{}_{c=5-4}\Sigma^{5} CUTVOL_c(X)],$$
$$= [3*0.5*(32*6.14) + 2*2.0*(32*6.14)] / [5*(32*6.14)],$$
$$= 1.1;$$

$$V_6(X) = [{}_{c=6-5}\Sigma^{6} DUST_c(X) * CUTVOL_c(X)] / [{}_{c=6-5}\Sigma^{6} CUTVOL_c(X)],$$

$$= [3*0.5*(32*6.14) + 3*2.0*(32*6.14)] / [6*(32*6.14)], \\ = 1.25;$$

$$V_7(X) = [{}_{c=7-6}\Sigma^{7}DUST_c(X) * CUTVOL_c(X)] / [{}_{c=7-6}\Sigma^{7}CUTVOL_c(X)], \\ = [3*0.5*(32*6.14) + 3*2.0*(32*6.14) + 3.5*(32*6.14)] / [7*(32*6.14)], \\ = 1.5714;$$

$$V_8(X) = [{}_{c=8-7}\Sigma^{8}DUST_c(X) * CUTVOL_c(X)] / [{}_{c=8-7}\Sigma^{8}CUTVOL_c(X)], \\ = [3*0.5*(32*6.14)+3*2.0*(32*6.14)+2*3.5*(32*6.14)]/[8*(32*6.14)], \\ = 1.8125;$$

$$V_9(X) = [{}_{c=9-8}\Sigma^{9}DUST_c(X) * CUTVOL_c(X)] / [{}_{c=9-8}\Sigma^{9}CUTVOL_c(X)], \\ = [3*0.5*(32*6.14)+3*2.0*(32*6.14)+3*3.5*(32*6.14)]/[9*(32*6.14)], \\ = 2.0;$$

There are eleven options for the tenth cut, but Cuts 10b-d and 10h-j would be constrained by any reasonable bolting constraint. Assuming right to left ventilation, Cut 10e is a turn to the blind side of the CM with a very high dust exposure level, Cut 10f is a deep cut and Cut 10g is a turn to the operator side of the CM both with high dust exposure levels, and Cuts 10a and 10k are initial cuts in an entry with low dust exposure levels. Cuts 10a, 10f, and 10k are the standard 32 feet in depth, but Cuts 10e and 10g are turned cuts of only 25 feet in depth.

Again using Equation 3.29, $V_i(X)$ is found for Cuts 10a, 10e, 10f, 10g, and 10k. For Cuts 10a and 10k:

$$V_{10}(X) = [{}_{c=10-9}\Sigma^{10}DUST_c(X) * CUTVOL_c(X)] / [{}_{c=10-9}\Sigma^{10}CUTVOL_c(X)], \\ = [4*0.5*(32*6.14)+3*2.0*(32*6.14)+3*3.5*(32*6.14)]/[10*(32*6.14)], \\ = 1.85.$$

For Cuts 10e and 10f:

$$V_{10}(X) = [{}_{c=10-9}\Sigma^{10}DUST_c(X) * CUTVOL_c(X)] / [{}_{c=10-9}\Sigma^{10}CUTVOL_c(X)], \\ = [(3*0.5*32*6.14)+(3*2.0*32*6.14)+(3*3.5*32*6.14)+(5.0*25*6.14)] / \\ [(9*32*6.14) + (25*6.14)], \\ = 2.24.$$

For Cut 10g:

$$
\begin{aligned}
V_{10}(X) &= [{}_{c=10\text{-}9}\Sigma^{10} DUST_c(X) * CUTVOL_c(X)] / [{}_{c=10\text{-}9}\Sigma^{10} CUTVOL_c(X)], \\
&= [(3*0.5*32*6.14)+(3*2.0*32*6.14)+(3*3.5*32*6.14)+(3.5*25*6.14)] / \\
&\quad [(9*32*6.14) + (25*6.14)], \\
&= 2.12.
\end{aligned}
$$

Applying the current MSHA standard of 2.0 mg/m^3 for DELV, only Cuts 10a and 10k satisfy the ventilation constraint of $V_i(X) \leq$ DELV and the CM is constrained from taking any other path for stage i = 10.

3.6 Chapter Summary

With the objective of satisfying six guiding policies or practices designed to achieve uniform advancement of a mining section by expediting the mining of crosscuts without jeopardizing miner health and safety, an algorithm was developed as the basis for a DP model that will identify optimal mining sequences for a CM production system. The algorithm consists of an optimal value function that seeks to minimize cut-cycle time. Minimizing cut-cycle time was selected over maximizing production for the optimal value function as it relies on time study data, which is easier to obtain than production data.

Minimizing cycle time can also be interpreted as maximizing CM utilization, which refers to the time the CM spends actually producing or loading coal. Moving the CM from cut to cut is a necessary part of the mining cycle; however, since coal is not produced during place changing, it is not considered as CM utilization. Nor is the change-out period during which time the CM is also not loading coal considered as CM utilization. Since most cuts in a standard mine plan are of uniform volume, seeking to minimize non-producing place change and change-out functions while maintaining a near constant loading function allows the DP algorithm to select cuts with minimum cut-cycle times, which are analogous to cuts that achieve maximum CM utilization.

The optimal value function of the DP algorithm has two primary components – a production cycle time and a place change cycle time. The production cycle time consists of multiple incremental times for haulage unit loading and change-out functions. Adjustments are made for change-out conditions, waiting on haulage units at the change-out point, and repositioning the haulage unit during clean-up passes at the end of each cut. The place change cycle time consists of one basic time element based on the distance trammed between cuts and the CM tram speed. It is adjusted for extra time required to maneuver around corners and cable handling.

A recurrence relation incorporates the algorithm into a DP model, which defines a local optimum for a single stage. In addition to the local optimum, other near-optimal stages that satisfy specified boundary conditions are evaluated further in search of a globally optimized mining sequence.

Constraints may be imposed on the DP model to insure and maintain a healthy environment and safe working conditions for coal miners. Constraints are designed for compliance with regulations as well as to establish standard operating procedures.

Finally, it should be noted that the algorithm described in this chapter was fine-tuned through an extensive validation process that is the subject of Chapter 4. That fine tuning included revising the wait-no-car adjustment factor associated with the change-out time element and adding the clean-up adjustment factor associated with the loading time element.

CHAPTER 4

VALIDATION OF THE ALGORITHM

4.1 Introduction

The DP algorithm described in Chapter 3 is the product of a development process through which the basic initial concept of an optimized mining sequence (OMS) model was enhanced with additional components and adjustment factors based on experience gained in attempted field applications (Chugh, 2006a; Chugh 2007). The final stage of development was a two-step validation process. This chapter describes each step in detail as well as how the validation process led to fine tuning the algorithm of which the DP model is constituted.

In the first step of the validation process, industrial engineering studies measured all pertinent operational parameters on a room-and-pillar mining section as well as production and place change cycle times for two-thirds of a nine-hour shift. Measured parameter values were incorporated into the main algorithm of the DP model generating cycle time predictions for each component of the cut cycle, which were compared with measured cycle times. In this comparison, consistently occurring and uniformly valued differences observed in loading and change-out cycles led to a revision of the wait-no-car adjustment factor and the addition of the haulage unit reset component of the loading cycle. This fine tuning has resulted in model predictions that were extremely well-matched with measured cycle times.

The second step of the validation process expanded the predicted versus actual comparison from one to twenty-four shifts. Production reports provided actual cycle time data for this step. Despite production report data being accurate to only ± 2.5 minutes, this validation step showed that production cycle times predicted when modeling the actual mining sequence closely matched cycle times reported by the mine operator. Place change cycle times were not as closely matched, which is attributed to a "human factor." This factor is defined and presented as an efficiency improvement training opportunity.

Description of cooperating mine. Algorithm validation requires showing that output from the DP model using the algorithm matches actual results given the same operational conditions and circumstances. Actual results are obtained from industrial engineering time studies and mine production reports, time studies being more accurate and dependable due to the purpose for which data is collected. Production reports are generated to satisfy customer, shareholder, and regulatory requirements. Time studies are performed to identify efficiency improvement opportunities. For researchers who need such data, both require the cooperation of an operating underground coal mine.

The mine cooperating with this study is a new facility in southern Illinois. The company operating the mine has no other operations allowing engineers and operations managers to establish their own policies and procedures instead of having to follow broad mandates from a large organization with several mines trying to maintain uniform standards across a wide variety of mining conditions. This setting provided an ideal opportunity for both validation and application of the DP model and its algorithm.

The mine operates with a non-union workforce in the No. 6 seam of the Illinois Basin at an approximate depth of 250 feet. It is strictly a room-and-pillar operation with no plans for longwall mining during its forecasted 30-year lifetime. Mine-specific geologic data pertinent to DP algorithm parameters are listed in Table 4.1 in order of variability from least variable to most variable. The term "standard" in the table describes normal conditions, that being a thin (less than two feet) layer of shale overlain by competent limestone in the immediate roof. When such conditions exist, the CM can make 32-ft deep cuts and roof rock or out-of-seam dilution (OSD) will not fall before being bolted. When limestone thins or disappears, shale thickness increases and, due to its structural weakness, it falls during mining. This is typical for the No. 6 coal seam throughout the Illinois Basin. When limestone thins, cut depth is shortened to as little as ten feet in an effort to minimize unsupported top and expedite bolting. Time study data were collected on an 11-entry super-section developing a main entry system. Entries are identified with sequential numbering (#1 through #11) from left to right across the section and crosscuts are identified by distance from the main slope.

Table 4.1. Mine-specific geologic data pertinent to the DP algorithm.

DP Algorithm Variable	Description	Unit of Measure	Value
COALDEN	Unit Weight of Coal	lb/ft^3	83
OSDDEN	Unit Weight of Rock	lb/ft^3	144
n/a	Entry/Crosscut Spacing	feet	80
$WIDTH_{STD}(X)$	Standard Entry Width	feet	18
$SEAM_{STD}(X)$	Standard Seam Height	feet	7.5
$OSD_{STD}(X)$	Standard Out-of-Seam Dilution Thickness	feet	0.5
$HEIGHT_{STD}(X)$	Standard Mining Height	feet	8
$DEPTH_{STD}(X)$	Standard Cut Depth	feet	32

At the time this validation study was being conducted, all equipment at the mine was relatively new, having been in full production for less than one year. Battery-powered ramcars provided haulage with six units typically assigned to the section, usually split equally with three on each side. The mining section had three roof bolters with the left-side bolter working in Entries #1, #2, and #3; the middle bolter working in Entries #5, #6, and #7; the right-side bolter working in Entries #9, #10, and #11; and the middle bolter sharing Entry #4 with the left-side bolter and Entry #8 with the right-side bolter. Equipment specifications pertinent to the DP algorithm are given in Table 4.2.

Table 4.2. Equipment specifications pertinent to the DP algorithm.

DP Algorithm Variable	Description	Unit of Measure	Value
PLD_{HU}	Haulage Unit Capacity	tons	12
$LENGTH_{CM}$	CM Length	feet	32
$LENGTH_{BLTR}$	Bolter Length	feet	32

Description of time studies. Time study data were collected by three observers for seven cuts made by the right-side CM. One observer was stationed alongside the CM operator to capture start and stop times for CM loading and place change cycles using the template shown in Table 4.3. Depending on cut configuration, this observer may or may not be able to see the change-out point and record arrival times of loaded and empty haulage units at that point. A second observer was stationed at the haulage unit dump point to capture start and stop times for haulage unit unloading using the template shown in Table 4.4. Typically, "ARRIVE @ FACE" is the loading cycle start time or change-out cycle stop time, "LEAVE FACE LOADED" is the loading cycle stop time or change-out cycle start time, "START DUMPING" is loaded haul cycle stop time, and "LEAVE FEEDER" is the empty haul cycle start time. The third observer is able to roam about the section capturing haulage unit arrival times at the change-out point if the first observer is unable to see that spot, or otherwise watching for delays such as haulage units leaving their rotation to change batteries.

Table 4.3. Time study template for CM operator observer.

Date:			**Mine:**					**Cut #:**			
Start:			**Section:**					**Place:**			
Stop:			**Observer:**					**Type of Cut:**			
Time:			**Operator:**					**# of cars used:**			
Cut Dimensions:							**COD:**				
	HEIGHT:						**HD1:**				
	OSD:						**HD2:**				
	WIDTH:						**HD3:**				
	DEPTH:						**HD4:**				
CAR #	**CAR ID**	**ARRIVE @ CHANGE OUT**	**ARRIVE @ FACE**	**LEAVE FACE LOADED**	**CNUP**	**ARRIVE @ CHANGE OUT**	**MINER RESET**		**OTHER DELAYS**		**NOTES**
							Start	**Stop**	**Start**	**Stop**	
1											
2											
3											
4											
5											
6											
7											
8											
9											
10											
11											
12											
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30											
31											
32											
33											
34											
35											
36											
Totals		0.00	0.00	0.00		0.00					

Table 4.4. Time study template for dump point observer.

Date:		**Mine:**		**Cut #:**			
Start:		**Section:**		**Place:**			
Stop:		**Observer:**		**Type of Cut:**			
Time:		**Operator:**		**# of cars used:**			
HD1:				**HD3:**			
HD2:				**HD4:**			
CAR #	**CAR ID**	**ARRIVE AT FEEDER/ CHANGE OUT**	**START DUMPING**	**LEAVE FEEDER**	**DELAYS (if any)**		**NOTES**
					Start	**Stop**	
1							
2							
3							
4							
5							
6							
7							
8							
9							
10							
11							
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32							
33							
34							
35							
36							
Totals		0.00	0.00	0.00			

Time study data collected for seven cuts made by the right-side CM were compiled to determine and define each of the DP algorithm operational parameters as shown in Table 4.5.

Table 4.5. Equipment-specific average time study data for the DP algorithm.

DP Algorithm Variable	Description	Unit of Measure	Value
FILL	Haulage Unit Fill Factor	%	78
SPD_{HU}	Haulage Unit Speed	feet/minute	392
$RESET_{HU}$	Haulage Unit Reset Time	minutes	1.37
PLD/DR_{HU}	Haulage Unit Dump Time	minutes	0.32
SWIN	Haulage Unit Switch In/Out Time	minutes	0.20
SPD_{CM}	CM Tram Speed	feet/minute	74.23
LR_{CM}	Normal CM Loading Rate	tons/minute	10.9
LR_{CM}	CM Loading Rate when Turning Crosscut	tons/minute	7.0
$CORTM_{CM}$	Time Required for CM to Go Around a Corner	minutes	0.66
$DRCHTM_{CM}$	Time Required when CM Changes Directions	minutes	1.38
$HANGTM_{CM}$ or $HANDTM_{CM}$	Time Required to Hang or Handle CM Cable	minutes	2.00
$HOOKTM_{CM}$	Time Required to Hook or Unhook Cable Loop on CM	minutes	0.22
n/a	Bolt Spacing	feet	4
SPD_{BLTR}	Bolting Speed	minutes/foot	1.59

4.2 Step 1 – Validation with Time Study Data

Fine tuning. The purpose of model validation is to verify that model outputs or predictions are accurate, and if not, to identify sources of inaccuracies and make revisions to model parameters or algorithms that will improve model accuracy. The amount of difference between model outputs or predictions and actual or measured values defines the degree of model validity with zero difference being the desired target.

With operational parameters defined, all seven cuts observed in the time study were modeled using the DP algorithm as it was defined at the time. Model inputs included not only measured and collected data shown in Tables 4.1, 4.2 and 4.5, but also actual observations and measurements of parameters specific to each cut, such as number of haulage units in rotation, haulage unit fill factor, CM loading rate, and OSD removed. Loading, change-out, and place change cycle times predicted by the model were compared with cycle times measured during the time study. The initial comparison clearly revealed that the DP algorithm, as it was defined at the beginning of the validation process, was consistently undervaluing both components of production cycle time as well as the place change cycle time. This revelation led to a more in-depth analysis of time study data including examining each individual haulage unit's cycle times, which brought about the realization that the biggest change-out cycle time discrepancies occurred when less than the normal quota of three haulage units were in the haulage rotation, and particularly when only one haulage unit was in use, as occurred twice during the time study period. The initial "wait-no-car" adjustment factor, defined as a ratio haul distance, $HD_i(X)$, to change-out distance, $COD_i(X)$, multiplied by the reciprocal of $[NCARS_i(X) – 1]$, which became undefined with $NCARS_i(X) = 1$. Consequently, using the SSP Model described in Chapter 2 as a guide, $WOC_i(X)$ was redefined to the form described in Chapter 3. A further inspection of time study data found that the last haulage unit loaded during each cut had to reset when the CM made clean-up passes on both sides of the cut, which had not been accounted for in loading time. Thus, the haulage unit reset factor, $RESET_{HU}$, was added to the loading time algorithm.

Reconciling differences between modeled and actual place change times was a bigger challenge as these differences were considerably larger than production time differences. The production time element consists of multiple incremental segments, each of which is frequently repeated and easily measured by normal time study practices. On the other hand, the place change time element is essentially one event that is often interrupted by unrelated events, such as breaks for food and water, communications with supervision and co-workers, CM inspections and upkeep, establishing and maintaining ventilation controls, and crossing paths with the roof bolter. These "human factors" occur intermittently but when they do occur, the amount of time added to the place change is significant. Incremental times for human factors can be quantified during the time study process, but developing a "human factor adjustment" and including it in every place change would unfavorably skew place change cycle times predicted by the model for the many cuts where they do not occur. Rather, while acknowledging that human factors do exist, the author recommends allowing the DP model to predict optimal place change cycle times that may be used as the focus of productivity improvement training.

Validation of individual parameters and components. The seven cuts for which time study data were collected are described by drawings made at the time of study and shown in Figures 4.1 through 4.7. Tables 4.6 through 4.12 compare cycle times measured during the time study and cycle times produced by the DP model. Although cut depth and OSD thickness are input parameters, they are included in the following tables to provide a frame of reference. Reported data for each cut is also shown in the right-hand column to provide an indication of the accuracy level of reported data, which is an important factor in Step 2 of the validation process.

Values in the "Difference" column of Tables 4.6 through 4.12 are the measure of model validity with differences approaching zero being indicative of greater accuracy. A production time difference of less than one minute for each of the seven cuts indicates that the DP model with the fine-tuned algorithm is valid.

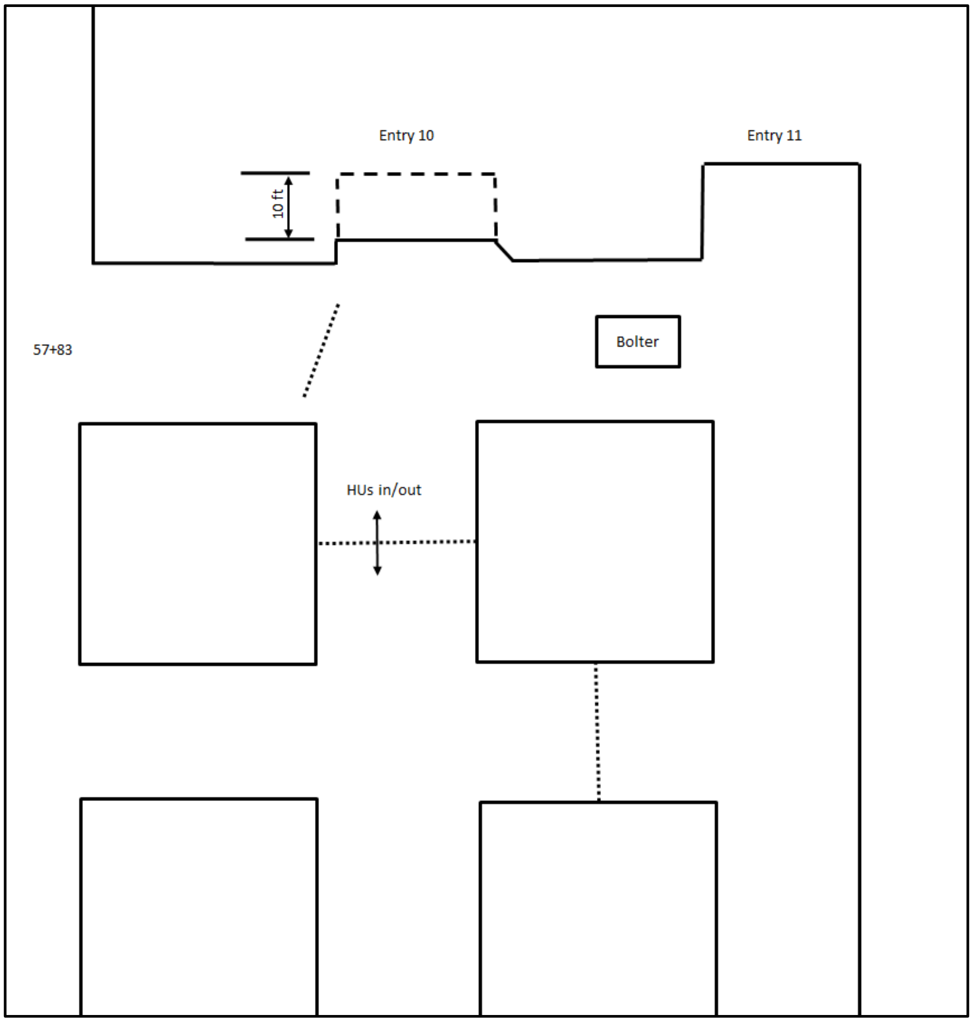

Figure 4.1. Depiction of mine conditions for 1st cut of time study.

Table 4.6. Time study/DP model comparison for 1st cut of time study.

	Time Study (TS)	**Model (DP)**	**Difference (DP – TS)**	**Shift Report**
Place Change Time (minutes)	n/a	n/a	n/a	13
Depth (ft)	10	10	0	10
OSD (in)	20	20	0	adverse
Cars	9	9	0	not reported
Loading Time (minutes)	8.5	9.6	1.1	
Change-out Time (minutes)	19.8	18.4	-1.4	
Production Time (minutes)	28.3	27.7	-0.3	27

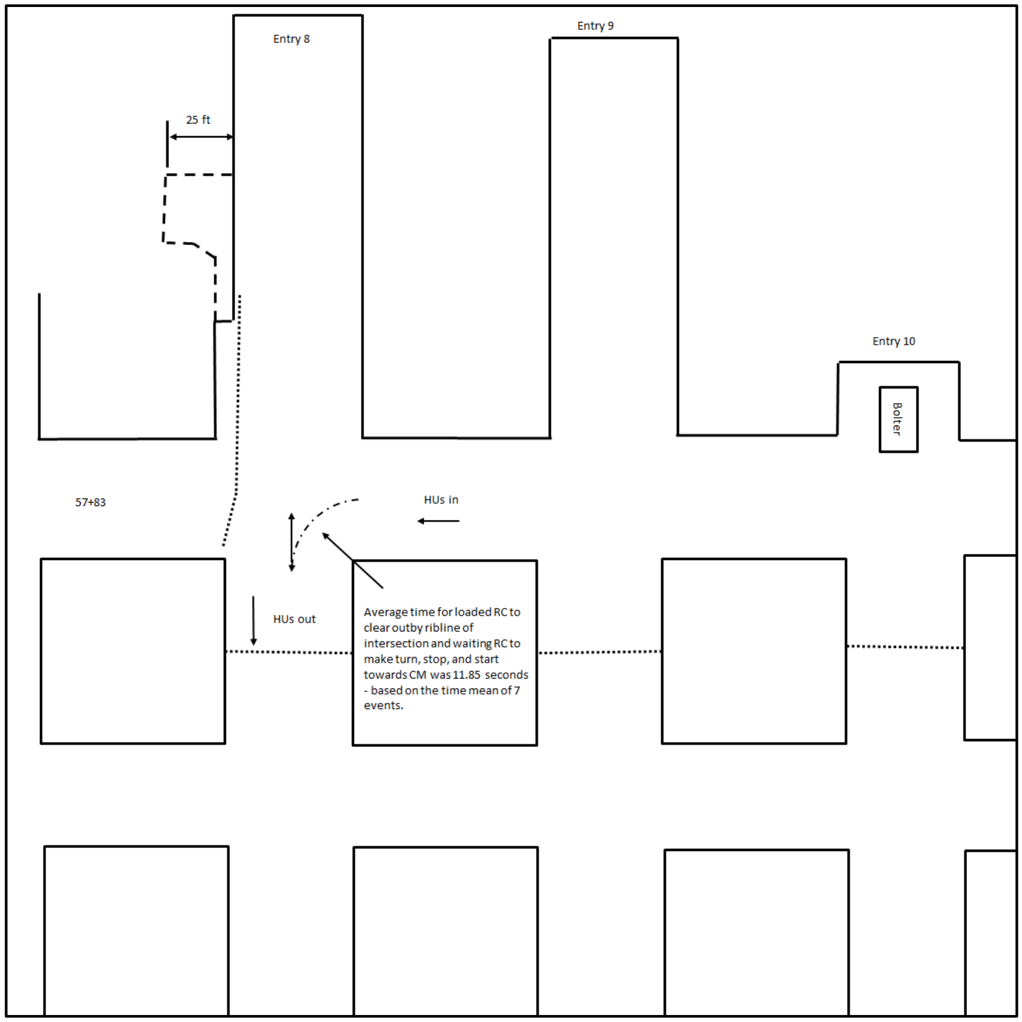

Figure 4.2. Depiction of mine conditions for 2nd cut of time study.

Table 4.7. Time study/DP model comparison for 2nd cut of time study.

	Time Study (TS)	Model (DP)	Difference (DP – TS)	Shift Report
Place Change Time (minutes)	14.8	9.8	-5.0	15
Depth (ft)	23	23	0	20
OSD (in)	6	6	0	good
Cars	17	17	0	not reported
Loading Time (minutes)	21.3	21.8	0.5	
Change-out Time (minutes)	21.6	21.1	-0.5	
Production Time (minutes)	42.9	42.9	0.0	40

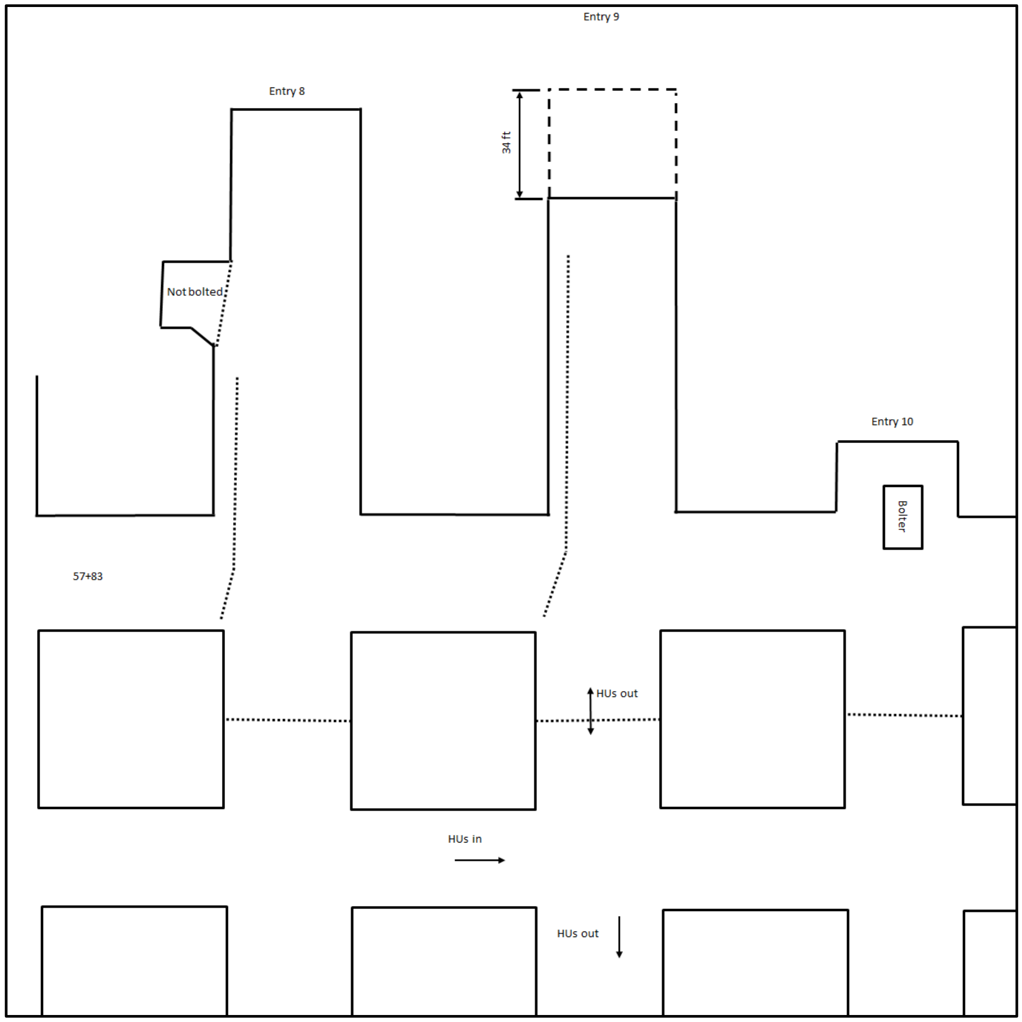

Figure 4.3. Depiction of mine conditions for 3rd cut of time study.

Table 4.8. Time study/DP model comparison for 3rd cut of time study.

	Time Study (TS)	Model (DP)	Difference (DP – TS)	Shift Report
Place Change Time (minutes)	7.9	8.0	0.1	8
Depth (ft)	32	32	0	32
OSD (in)	6	6	0	good
Cars	18	18	0	not reported
Loading Time (minutes)	13.0	13.6	0.6	
Change-out Time (minutes)	34.7	34.9	0.2	
Production Time (minutes)	47.7	48.5	0.8	50

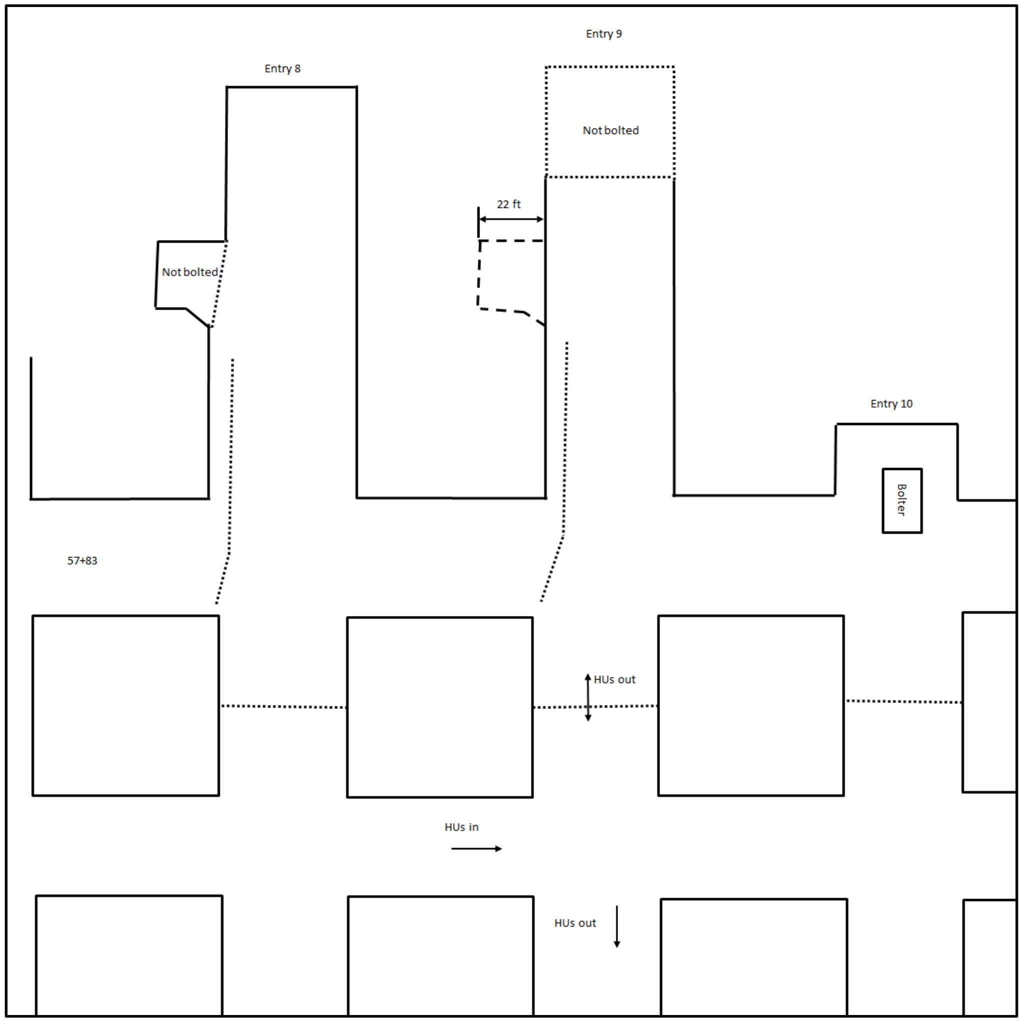

Figure 4.4. Depiction of mine conditions for 4th cut of time study.

Table 4.9. Time study/DP model comparison for 4th cut of time study.

	Time Study (TS)	Model (DP)	Difference (DP – TS)		Shift Report
Place Change Time (minutes)	6.3	5.0	-1.3		13
Depth (ft)	20	20	0		20
OSD (in)	4	4	0		good
Cars	17	17	0		not reported
Loading Time (minutes)	18.1	18.5	0.4		
Change-out Time (minutes)	26.5	26.4	-0.1		
Production Time (minutes)	44.6	44.9	0.3		42

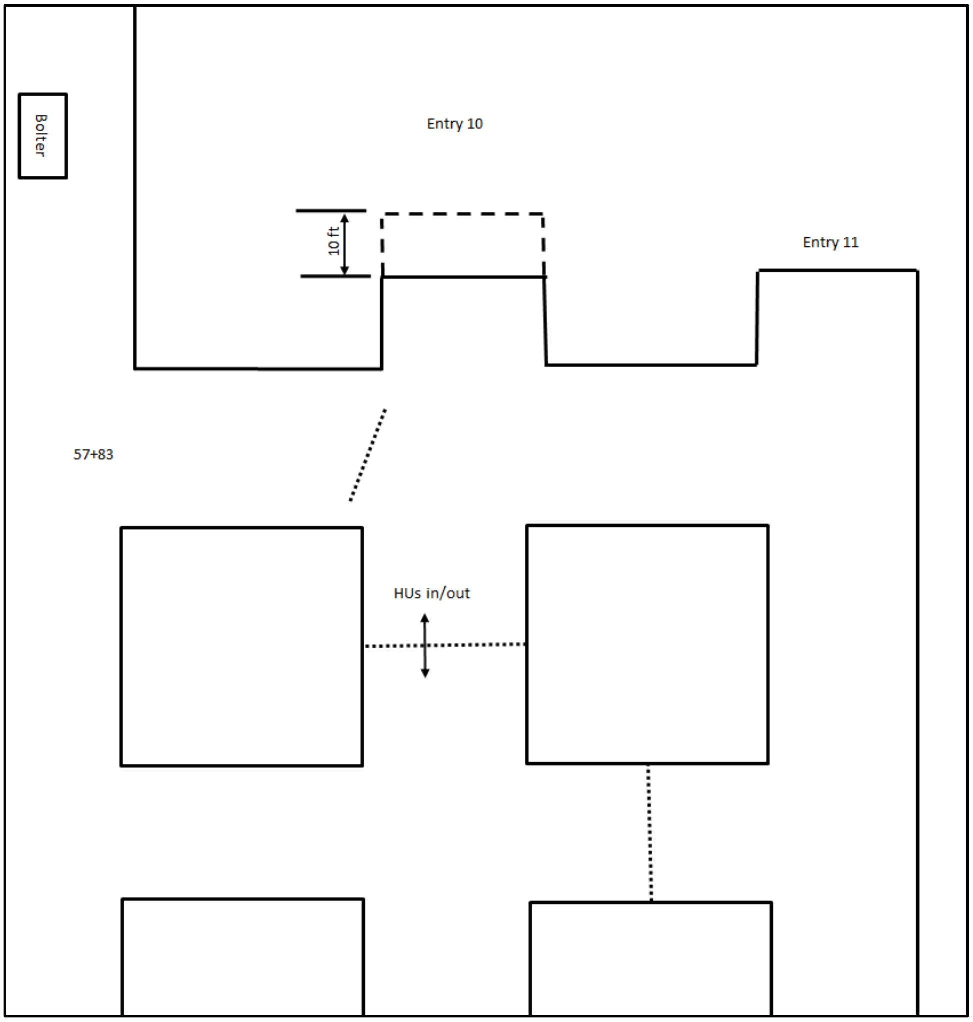

Figure 4.5. Depiction of mine conditions for 5th cut of time study.

Table 4.10. Time study/DP model comparison for 5th cut of time study.

	Time Study (TS)	Model (DP)	Difference (DP – TS)		Shift Report
Place Change Time (minutes)	13.8	6.9	-6.9		11
Depth (ft)	10	10	0		10
OSD (in)	27	27	0		adverse
Cars	9	9	0		not reported
Loading Time (minutes)	7.3	8.4	1.1		
Change-out Time (minutes)	9.0	8.4	-0.6		
Production Time (minutes)	16.3	16.8	0.5		16

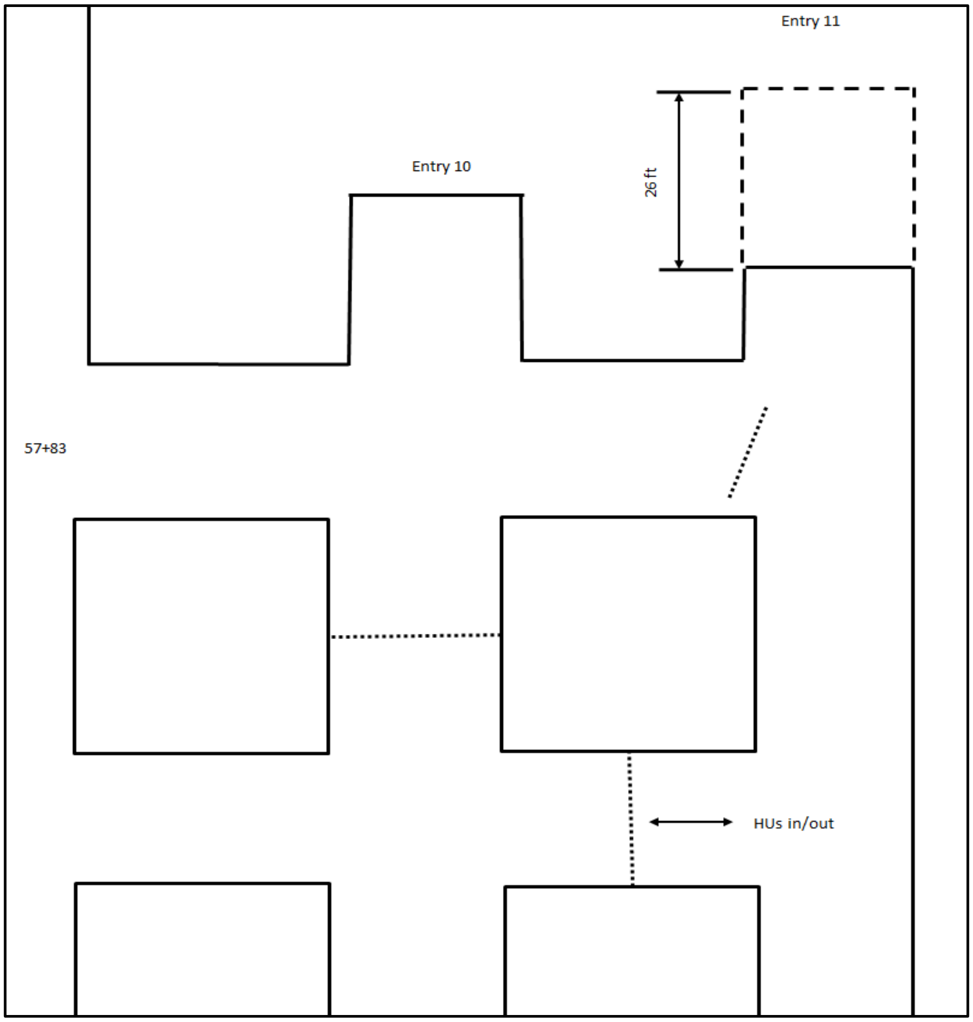

Figure 4.6. Depiction of mine conditions for 6th cut of time study.

Table 4.11. Time study/DP model comparison for 6th cut of time study.

	Time Study (TS)	Model (DP)	Difference (DP – TS)		Shift Report
Place Change Time (minutes)	12.1	4.9	-7.2		13
Depth (ft)	25	25	0		25
OSD (in)	8	8	0		adverse
Cars	17	17	0		not reported
Loading Time (minutes)	11.0	16.2	5.2		
Change-out Time (minutes)	22.5	17.5	-5.0		
Production Time (minutes)	33.5	33.7	0.2		43

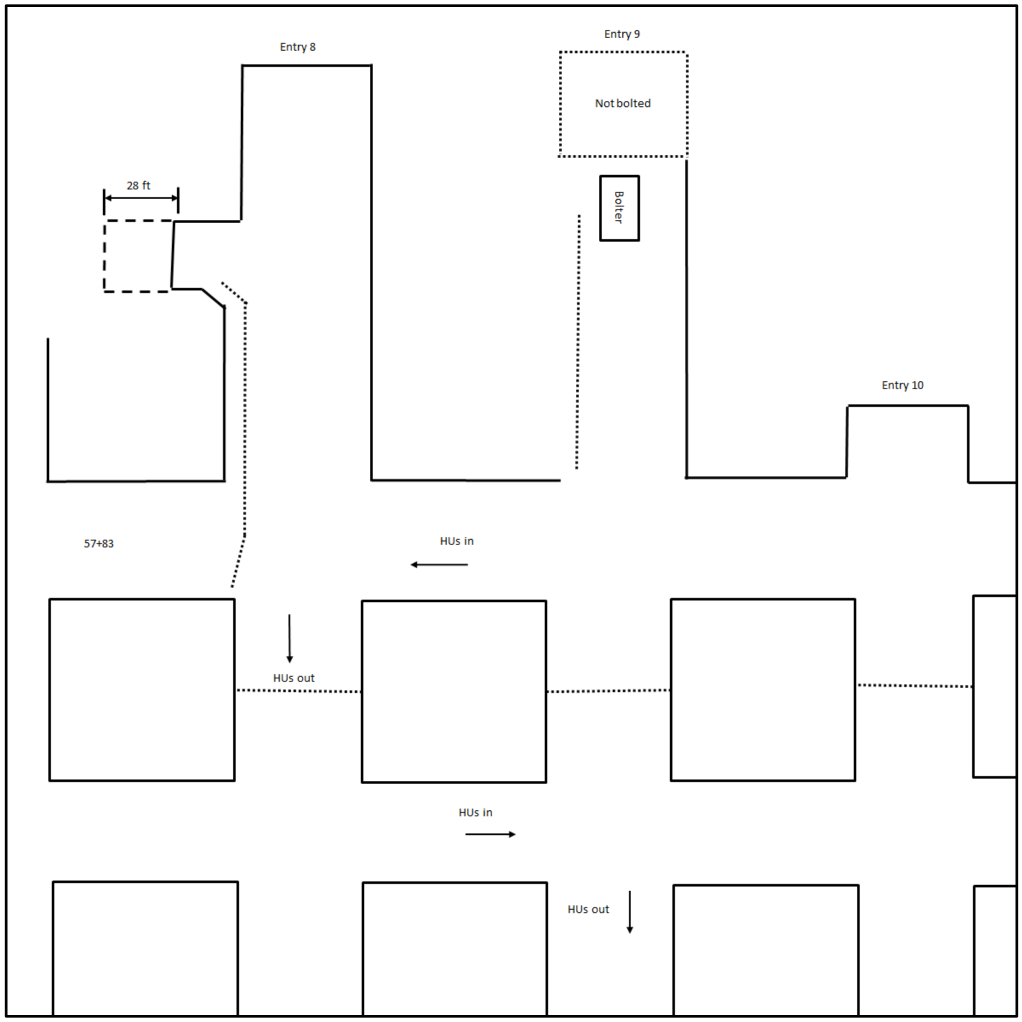

Figure 4.7. Depiction of mine conditions for 7th cut of time study.

Table 4.12. Time study/DP model comparison for 7th cut of time study.

	Time Study (TS)	Model (DP)	Difference (DP – TS)		Shift Report
Place Change Time (minutes)	17.2	11.8	-5.4		10
Depth (ft)	27	27	0		n/a
OSD (in)	7	7	0		good
Cars	17	17	0		not reported
Loading Time (minutes)	17.3	17.2	-0.1		
Change-out Time (minutes)	28.9	29.2	0.3		
Production Time (minutes)	46.2	46.4	0.2		n/a

4.3 Step 2 – Validation with Shift Report Data

Shift reports for two weeks from December 5, 2012 through December 19, 2012 were obtained from the cooperating mine. This data encompassed 24 shifts and 331 cuts and included place change and production cycle times as reported by mine foremen, who typically report times in 5-minute increments, or with an accuracy of ±2.5 minutes (see the sample shift report provided in Appendix A).

Each cut was charted on a sequence map to document the actual mining sequence. This sequence including each individual cut depth was modeled with the DP aglorithm. Cycle times predicted by the algorithm were compared with reported cycle times with the degree of matching used as a measure of the validity of the algorithm. This comparison was done separately for production and place change elements.

Production element. For the production element comparison, cuts were divided into four categories based on cut depth and change-out distance (COD) as follows:

1) Cuts greater than 28 feet in depth (160 cuts),
2) Cut 16 to 28 feet in depth with COD less than 110 feet (66 cuts),
3) Cuts 16 to 28 feet in depth with COD greater than 110 feet (34 cuts), and
4) Cuts less than 15 feet in depth (71 cuts).

Histograms of production cycle time frequency distributions predicted by the DP model versus those reported by mine foremen are shown in Figures 4.8 through 4.11. A smooth curve is fitted to each frequency distribution in Figures 4.12 through 4.15. Table 4.13 shows the closeness of the match between mean ($\overline{X}$) and standard deviation (s) for each category of cuts. Consistently uniform coefficients of variation (cv) between 17% and 29% indicate that the production element of the DP algorithm is valid.

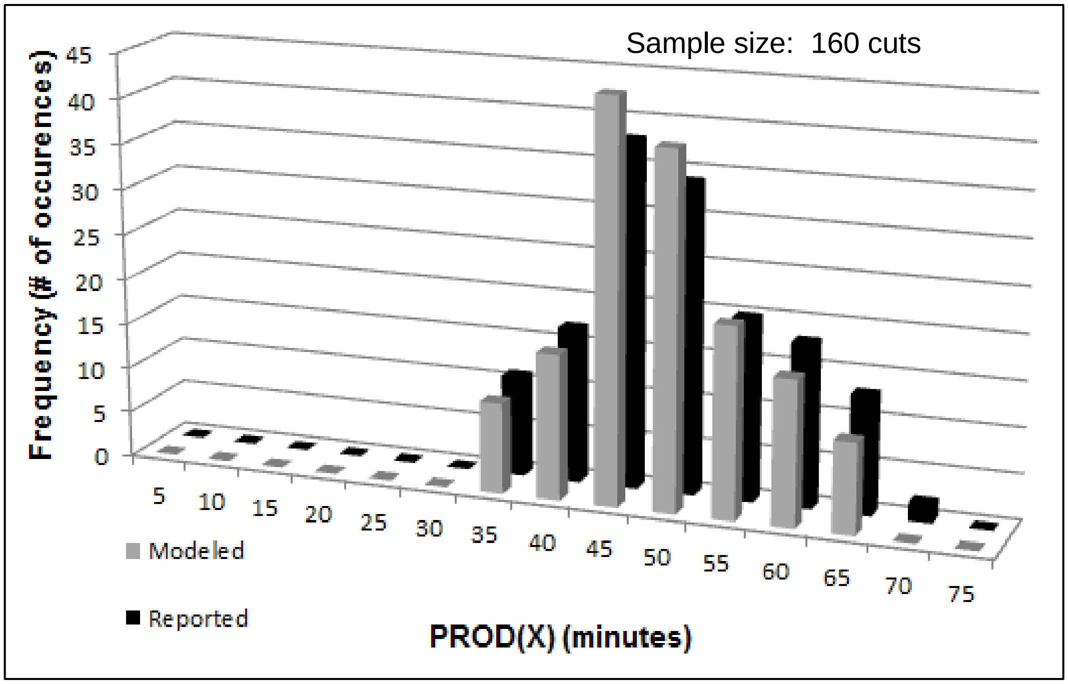

Figure 4.8. Frequency distribution of production cycle times for cuts greater than 28 feet in depth.

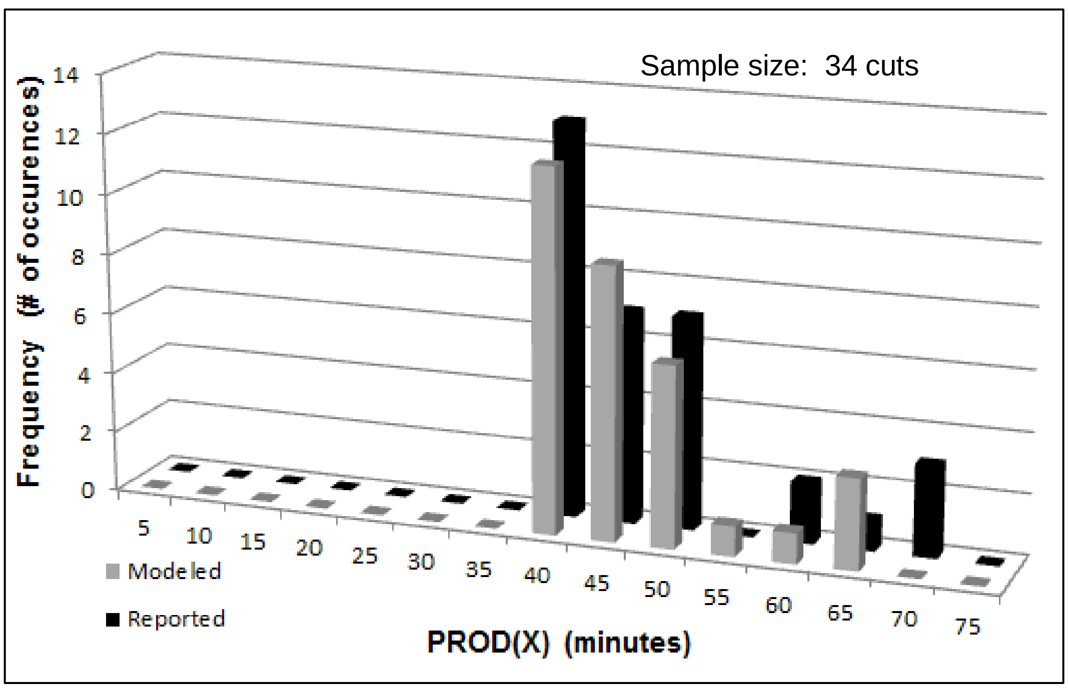

Figure 4.9. Frequency distribution of production cycle times for cuts 16 to 28 feet in depth with change-out distance greater than 110 feet.

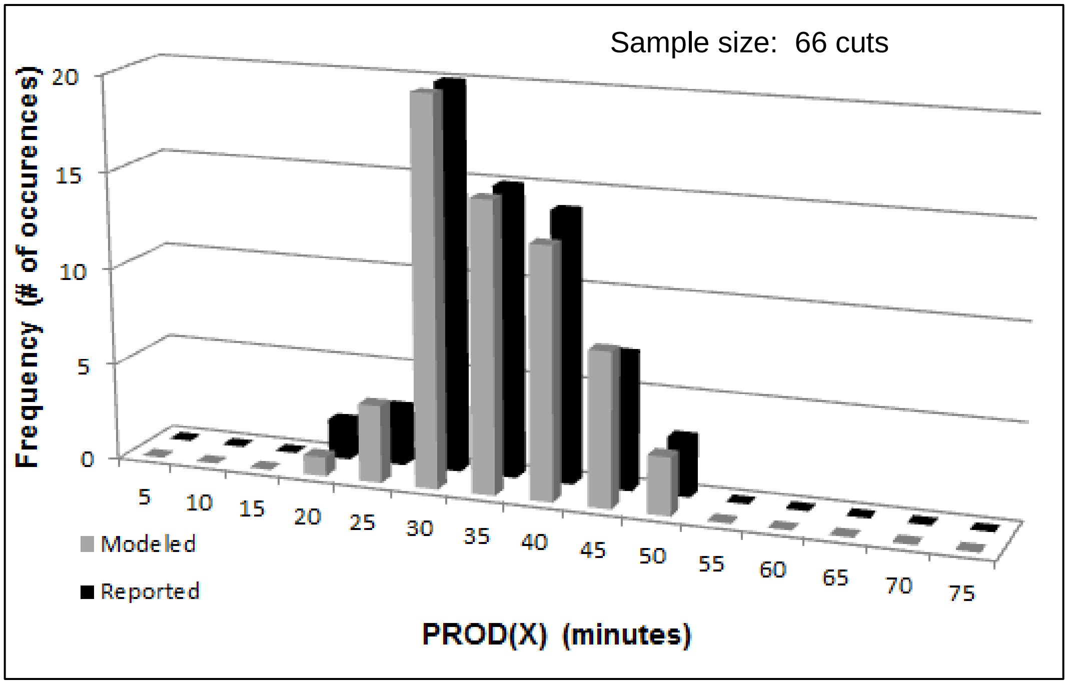

Figure 4.10. Frequency distribution of production cycle times for cuts 16 to 28 feet in depth with change-out distances less than 110 feet.

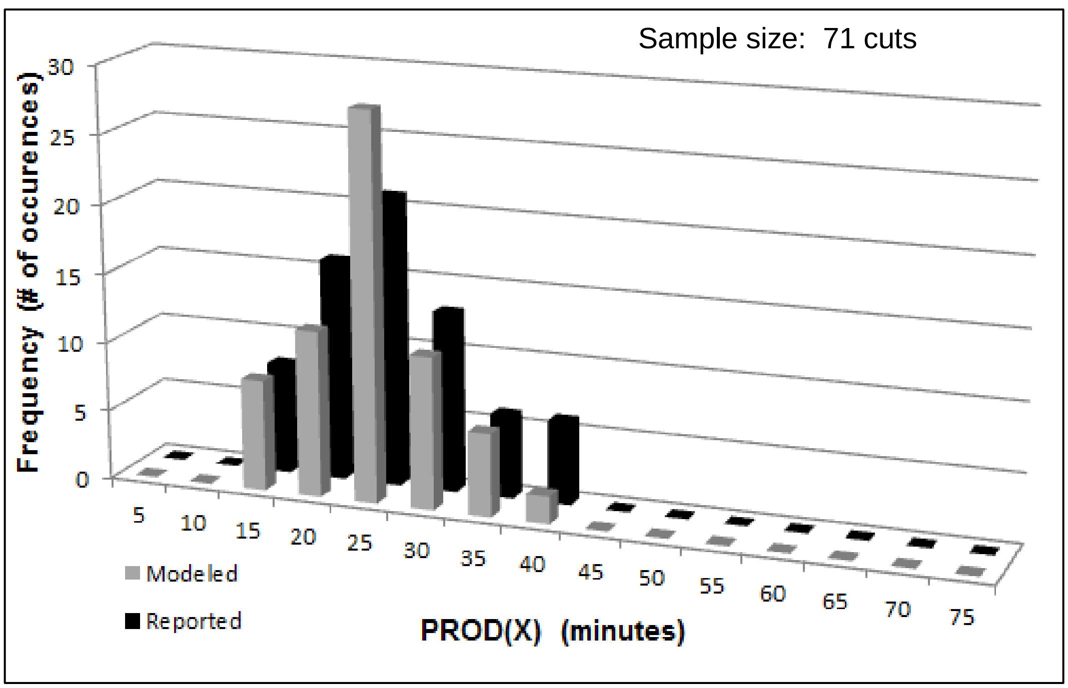

Figure 4.11. Frequency distribution of production cycle times for cuts 15 feet or less in depth.

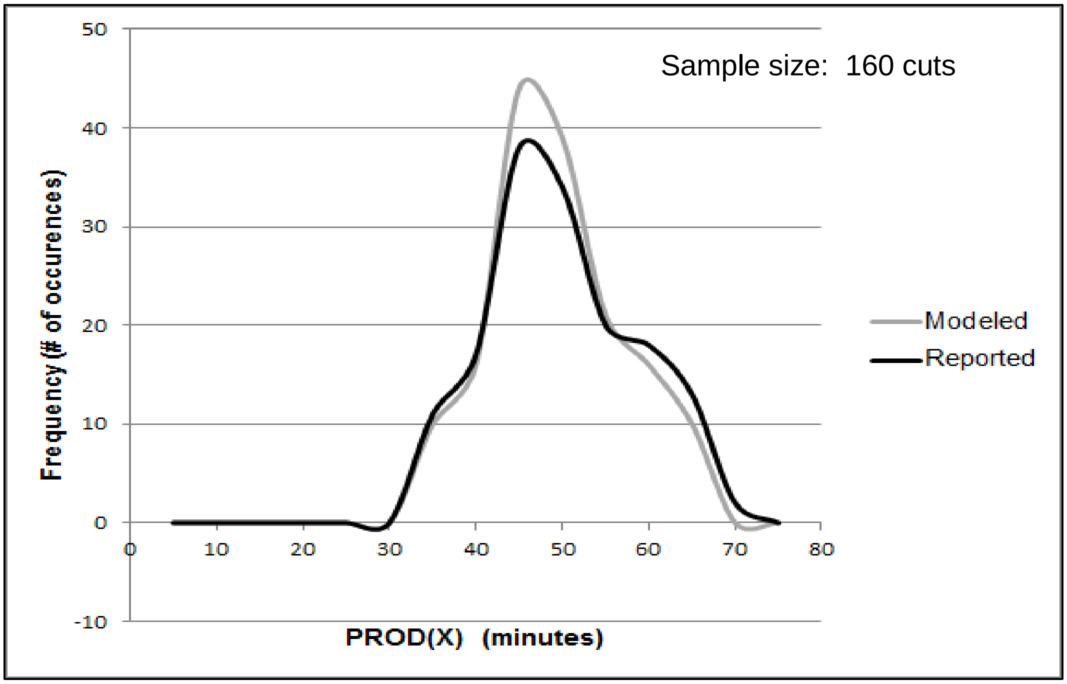

Figure 4.12. Smoothed curve fit to frequency distribution of Figure 4.8.

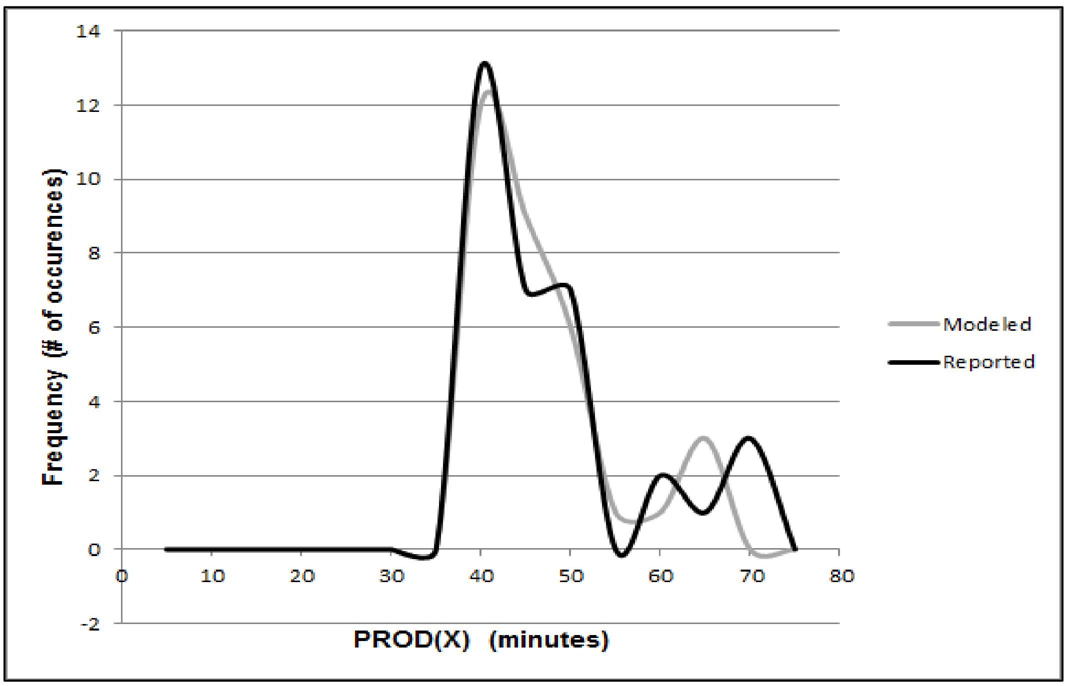

Figure 4.13. Smoothed curve fit to frequency distribution of Figure 4.9.

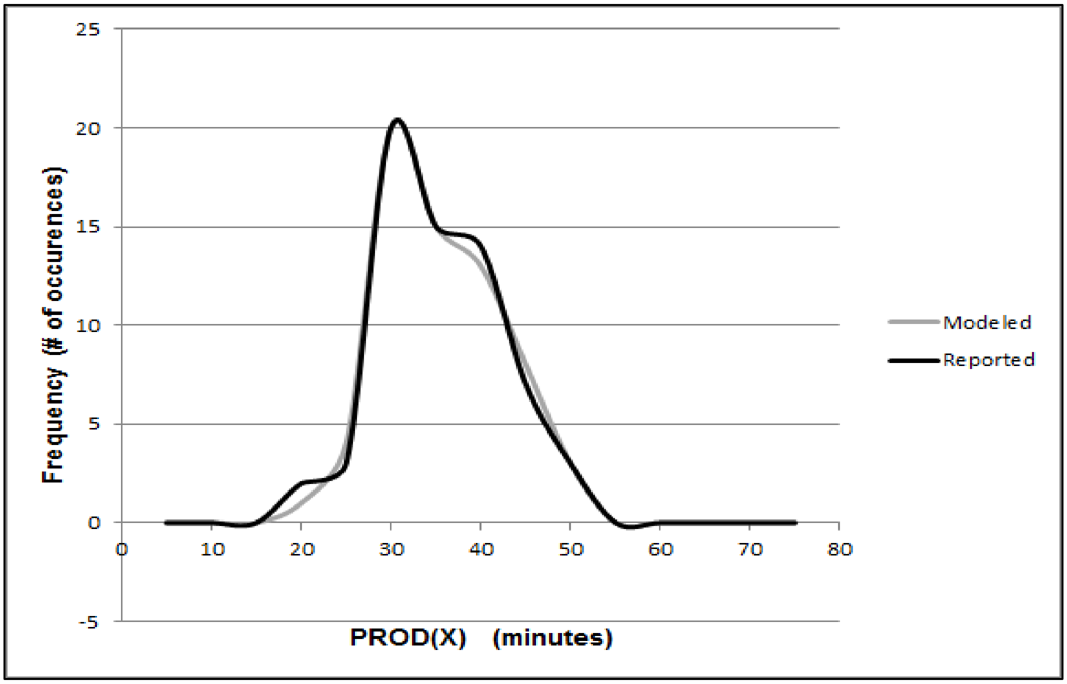

Figure 4.14. Smoothed curve fit to frequency distribution of Figure 4.10.

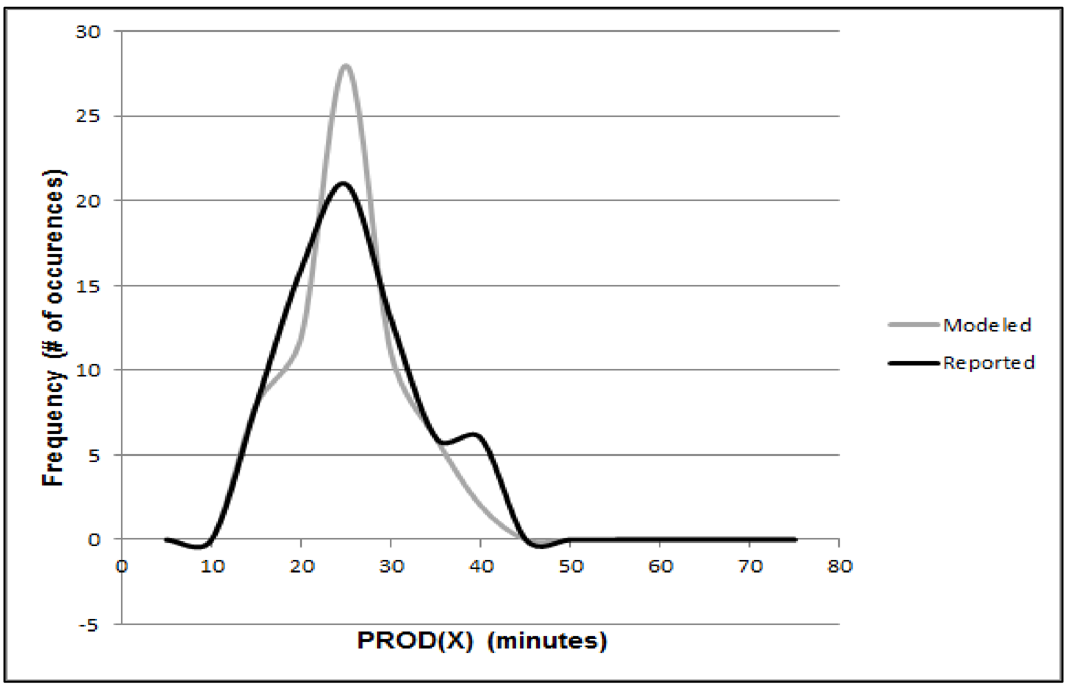

Figure 4.15. Smoothed curve fit to frequency distribution of Figure 4.11.

Table 4.13. Mean and variance comparison for production element of cycle time (in minutes).

Category of Cut	DP Model Data			Shift Report Data		
	$\overline{X}$	*s*	*cv*	$\overline{X}$	*s*	*cv*
> 28' Cuts	49.75	8.41	17%	50.03	9.39	19%
16-28' Cuts, COD > 110'	48.18	9.35	19%	48.68	10.52	22%
16-28' Cuts, COD < 110'	36.19	7.71	22%	36.14	7.93	22%
0-15' Cuts	26.22	7.68	29%	26.28	7.74	29%

Place change element. Place change cycle times were reported for 292 cuts. For the place change element comparison, cuts were divided into four categories as follows:

1) Moves greater than 400 feet in length (40 moves),
2) Moves between 240 and 400 feet in length (118 moves),
3) Moves less than 240 feet in length that are not double cut moves (86 moves), and
4) Moves where a second (double) cut is made without having to move to a different entry than the one used to access the cut the CM is moving from (48 moves).

Histograms of place change cycle time frequency distributions predicted by the DP model versus those reported by mine foremen are shown in Figures 4.16 through 4.19. Fitting a smooth curve to each frequency distribution, as depicted in Figures 4.20 through 4.23, provides a clearer indication of how closely DP model output matches reported place change cycle times. Unlike production cycle time, model predictions and shift report times for place change cycle time are not closely matched. Table 4.14 shows the difference between ($\overline{X}$) and standard deviation (s) values for the place change element of cycle time as predicted by the DP model and as reported by the mine foremen for each category of move. The average difference between modeled and reported place change times for all 292 moves is 6.83 minutes with a standard deviation of 5.49 minutes.

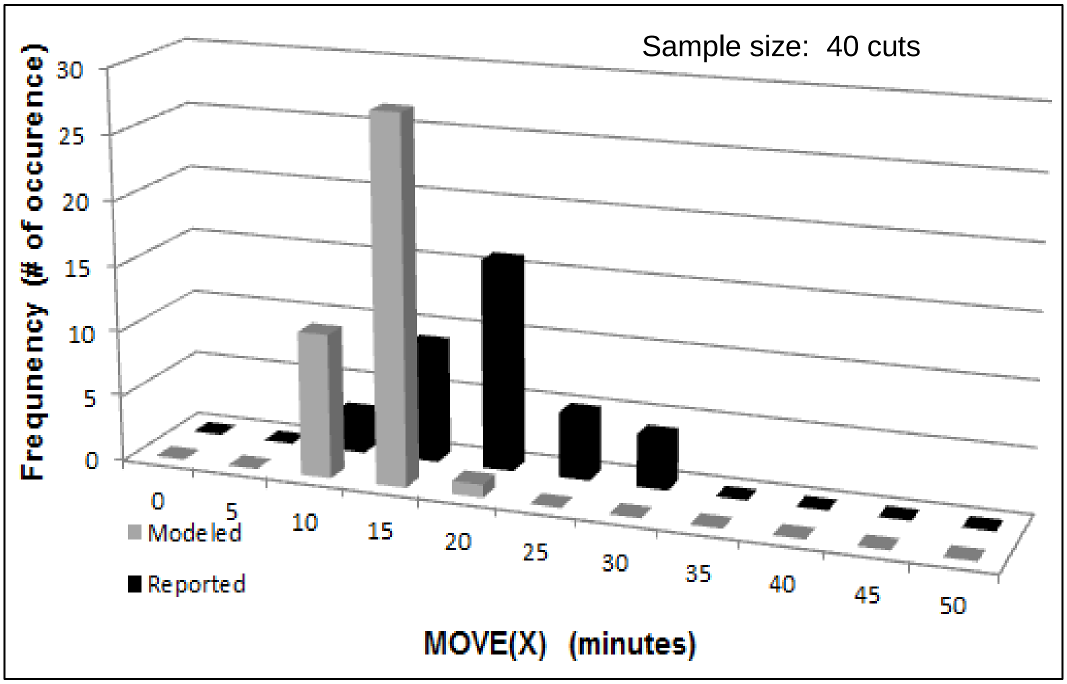

Figure 4.16. Frequency distribution of place change cycle times for moves greater than 400 feet long.

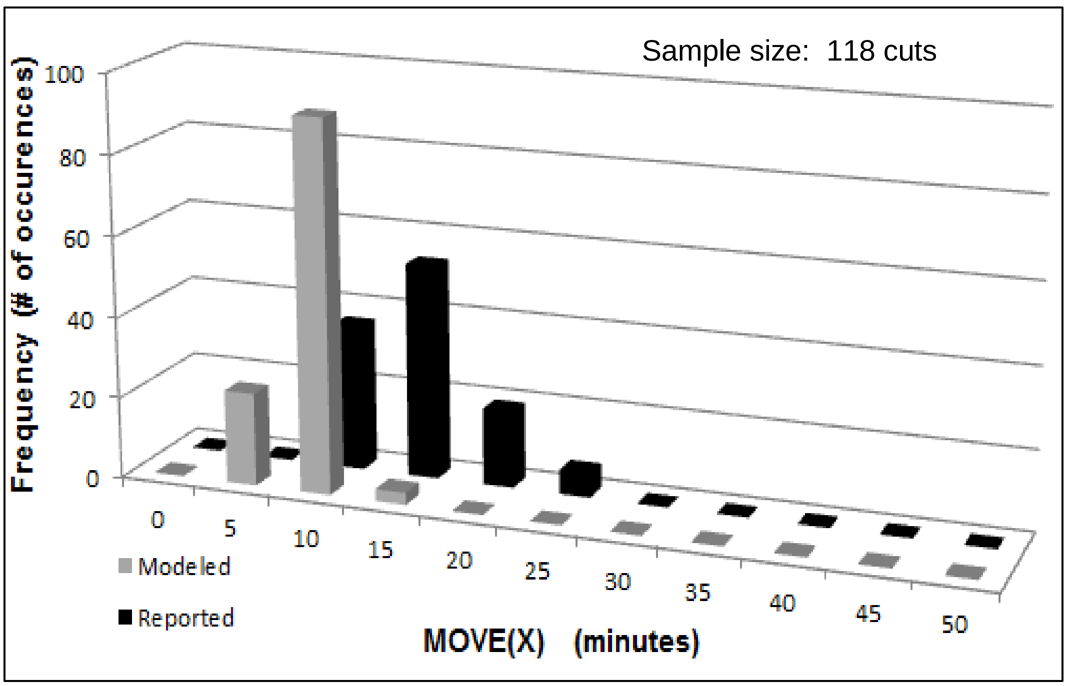

Figure 4.17. Frequency distribution of place change cycle times for moves between 240 and 400 feet long.

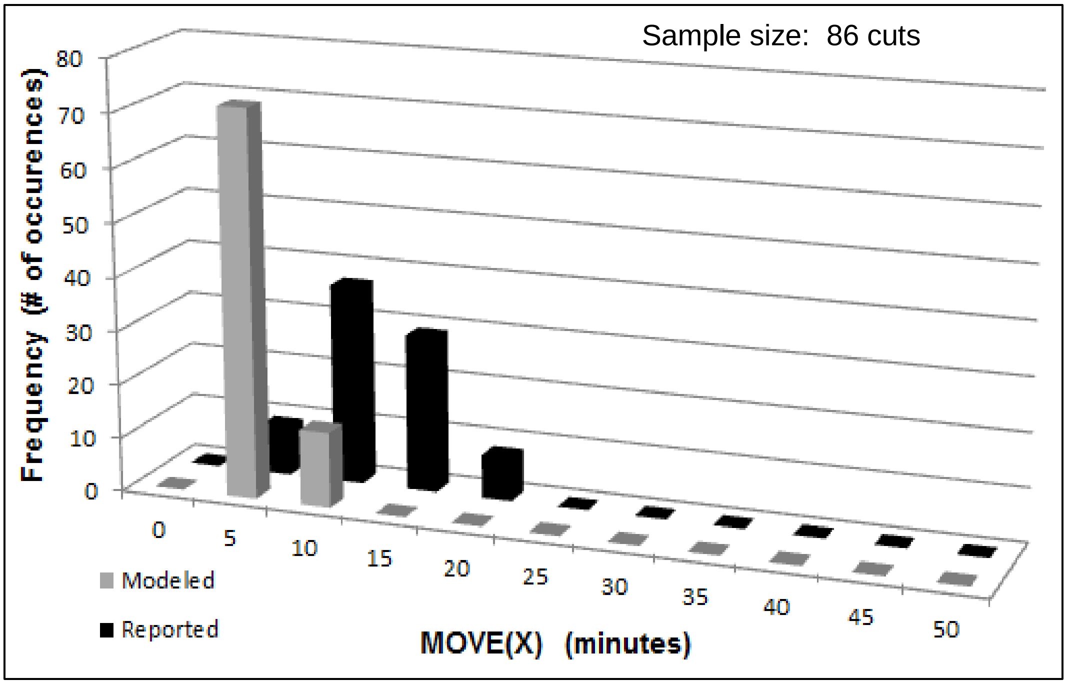

Figure 4.18. Frequency distribution of place change cycle times for moves less than 240 feet long excluding double cuts.

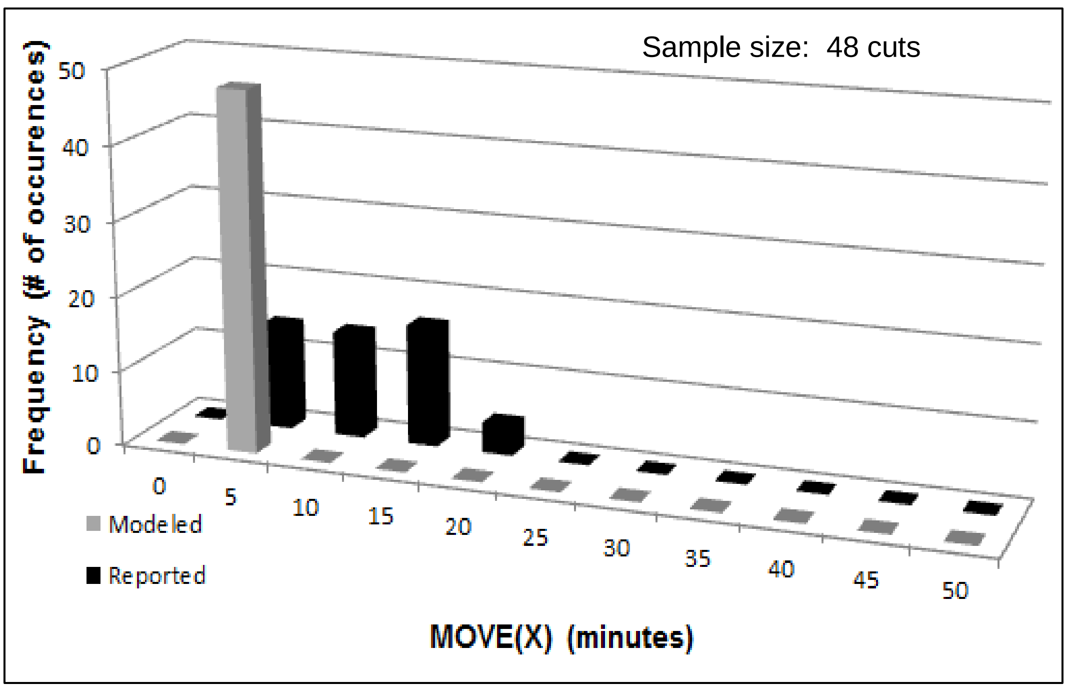

Figure 4.19. Frequency distribution of place change cycle times for double cut moves.

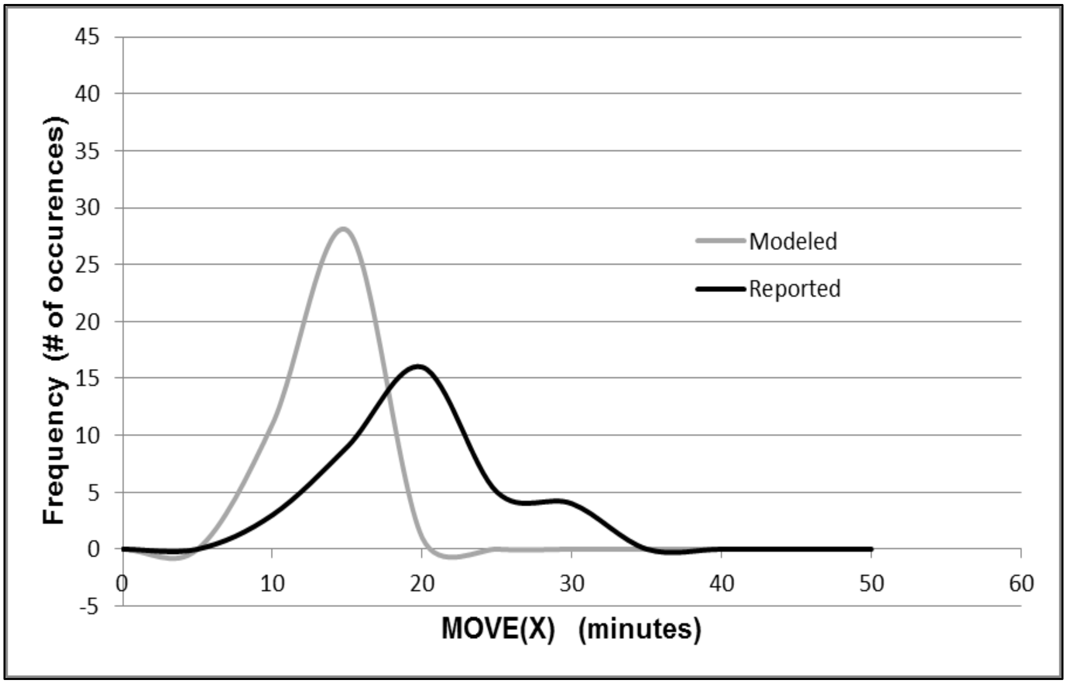

Figure 4.20. Smoothed curve fit to frequency distribution of Figure 4.16.

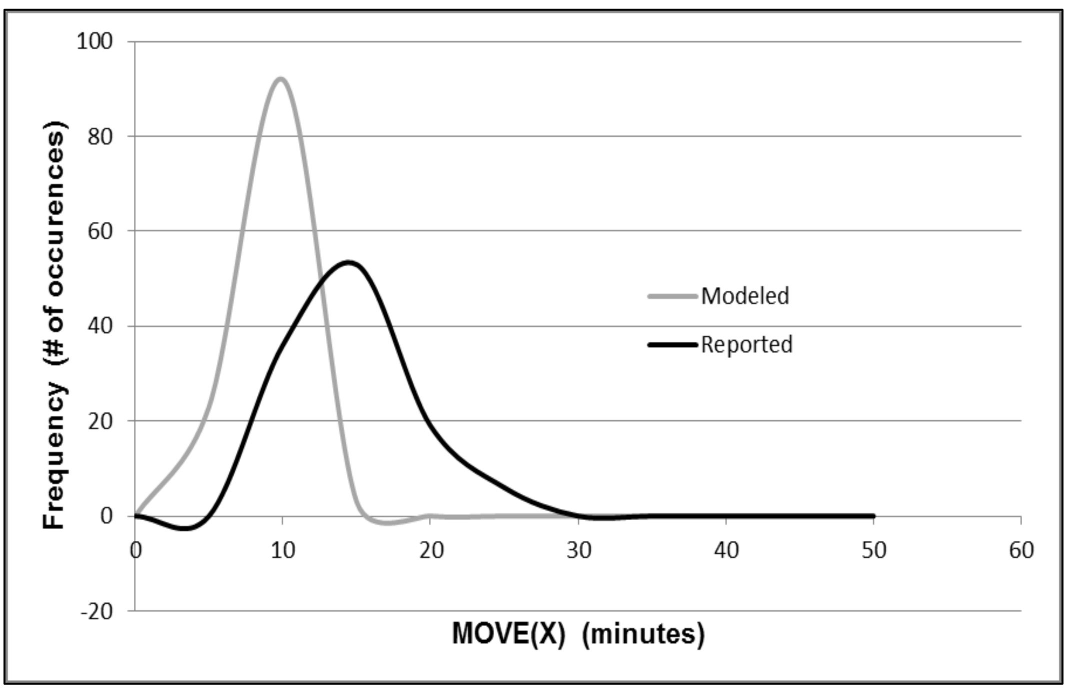

Figure 4.21. Smoothed curve fit to frequency distribution of Figure 4.17.

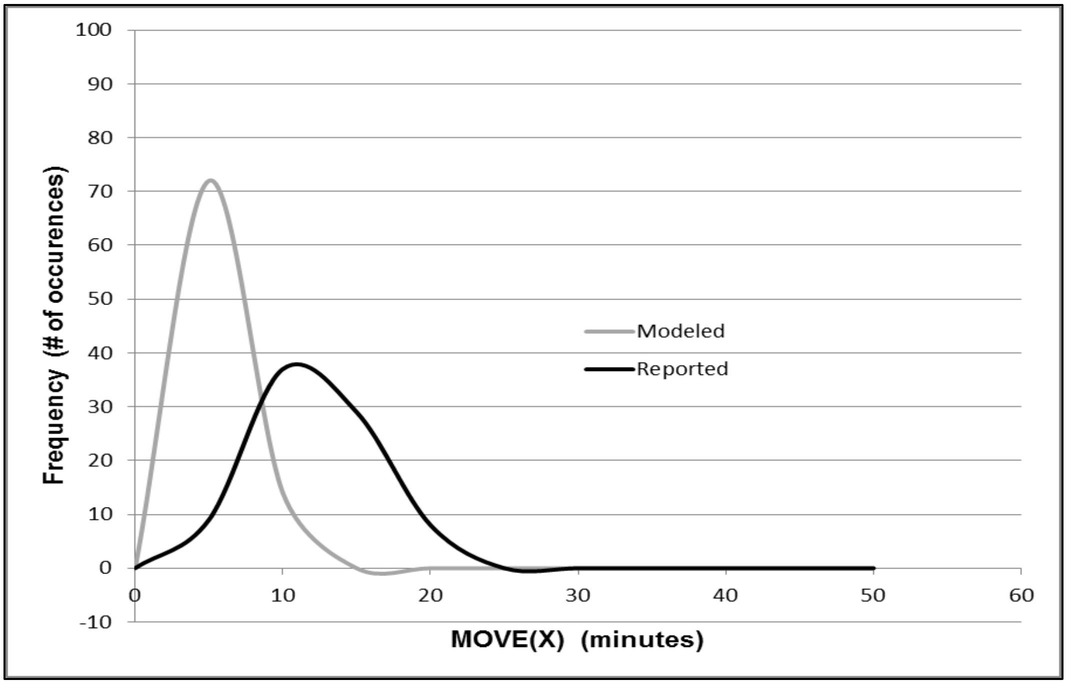

Figure 4.22. Smoothed curve fit to frequency distribution of Figure 4.18.

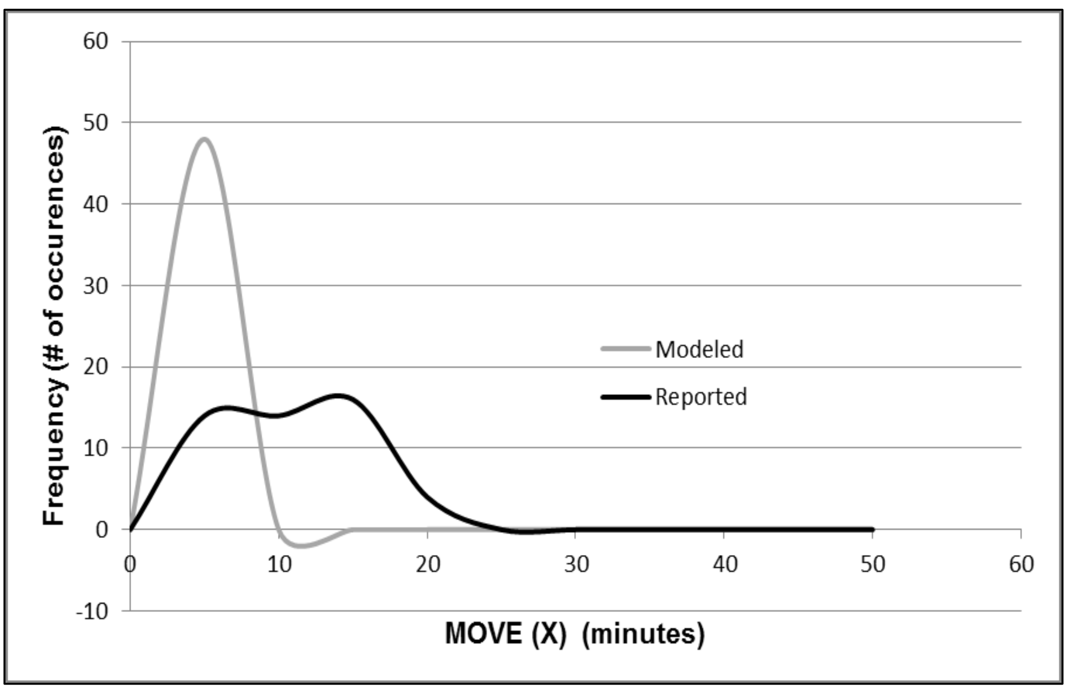

Figure 4.23. Smoothed curve fit to frequency distribution of Figure 4.19.

Table 4.14. Mean and variance comparison for place change element of cycle time (in minutes).

Category of Move	DP Model Data		Shift Report Data		Difference in Means
	$\overline{X}$	s	$\overline{X}$	s	
Move > 400’	13.48	2.03	19.20	4.83	5.72
240’ < Move < 400’	8.95	1.68	14.57	4.01	5.62
Move < 240’ (not Double)	6.01	1.40	12.13	4.02	6.12
Double Cut Move	3.88	0.53	11.10	4.78	7.23

Part of this difference was accounted for by examining differences between predicted and reported place change times as a function of tram distance. First, moves were categorized according to the type of cut being moved to, with eight possibilities defined as follows:

1) “Standard cut” is in an entry or straight where the distance from the last completed crosscut to the face is less than the crosscut spacing (71 moves),
2) “Deep cut” is in an entry where the distance from the last completed crosscut to the face is more than the crosscut spacing (41 moves),
3) First cut in a crosscut that is “turned” from an entry (15 moves),
4) First cut in a crosscut that is started “head-on” (23 moves),
5) Cut that advances a “crosscut” without completing it (24 moves),
6) “Hole through” cut that completes a crosscut (40 moves),
7) “Double cut” made in the entry from which the previous cut in a crosscut was accessed (48 moves), and
8) Cut made in the intake entry farthest from the outermost return entry immediately following a cut made in that outermost return entry requiring “rerouting” of the CM cable across the entire mining section (15 moves).

Next, moves within each category are grouped according to their respective reported place change time, i.e. groups with 5-, 10-, 15-, 20-, 25-, and 30-minute place change times. Within these groups, moves are further subdivided based on how many times the DP model indicated that the CM cable had to be handled or hung during the move. This parameter, NUMHAND in the DP algorithm, is rarely reported by mine foremen, but it is an important factor in the place change element of the DP algorithm. For modeling purposes, NUMHAND values for each move should be specified by an experienced miner familiar with CM production systems. The author's mining experience qualifies him to do that for this study.

With moves categorized, grouped, and subdivided, the difference between reported place change time and place change time predicted by the place change algorithm of the DP model was determined for each move. A continuous function plot of this difference versus distance trammed by the CM for each move of a given category, group, and subdivision describes the time variability of the place change component in the DP model's output. Continuous function plots for moves to each type of cut are provided in Figures 4.24 through 4.31. For these figures, markers of different shapes and colors distinguish between groups for each type of cut, and darker shading is used to distinguish between group subdivisions. For a given reported placed change time, e.g. 15 minutes, the shape and color of markers identifying it will be the same for all figures. Each figure shows linear regression trend lines and associated equations, where applicable, for each type of move. These were determined using the method of least squares. Some figures contain black "X" markers. These are moves with reported place change times that were not a multiple of five minutes, of which there were fourteen.

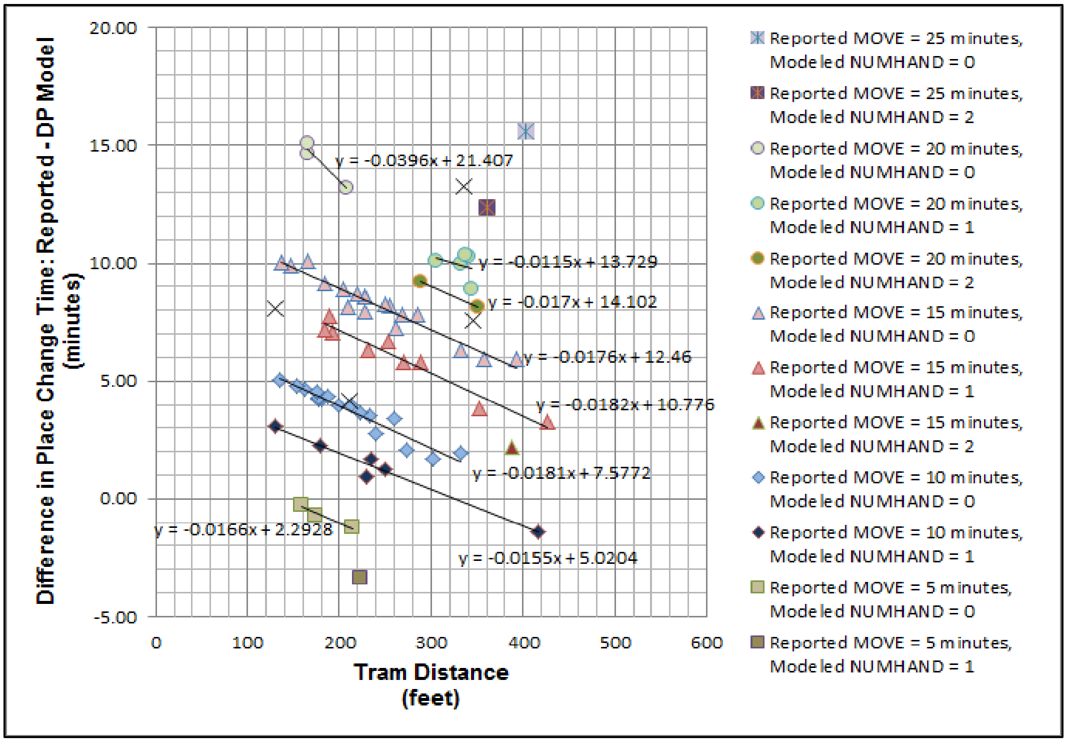

Figure 4.24. Continuous function plot for moves to “standard cuts.”

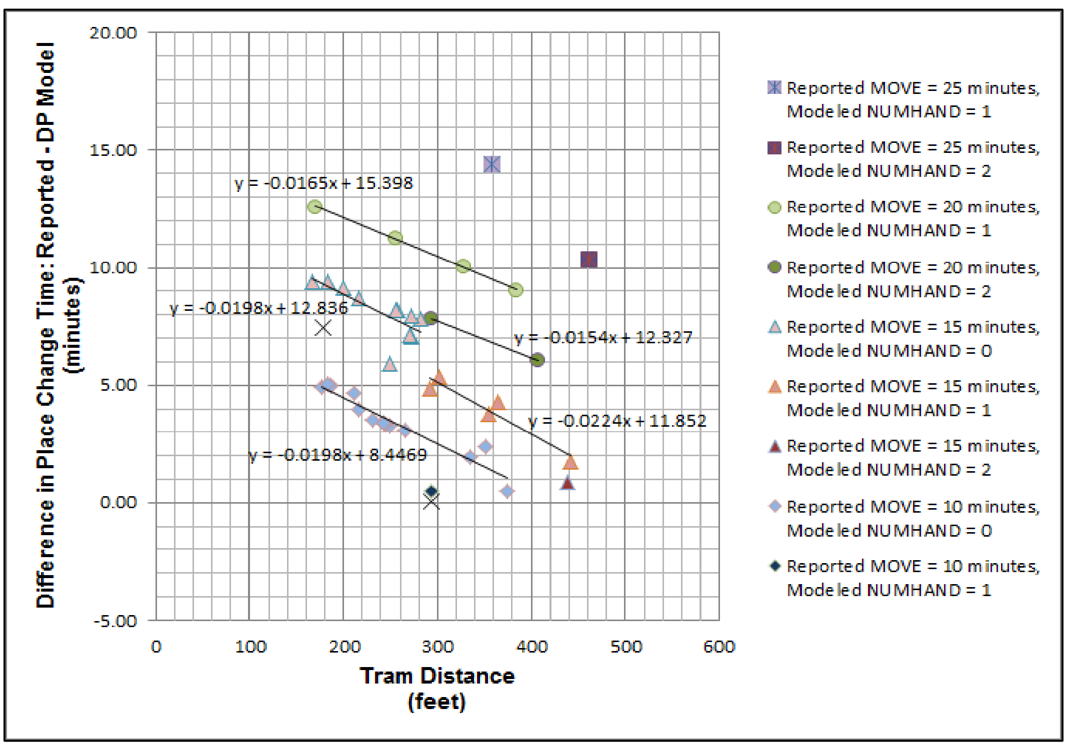

Figure 4.25. Continuous function plot for moves to “deep cuts.”

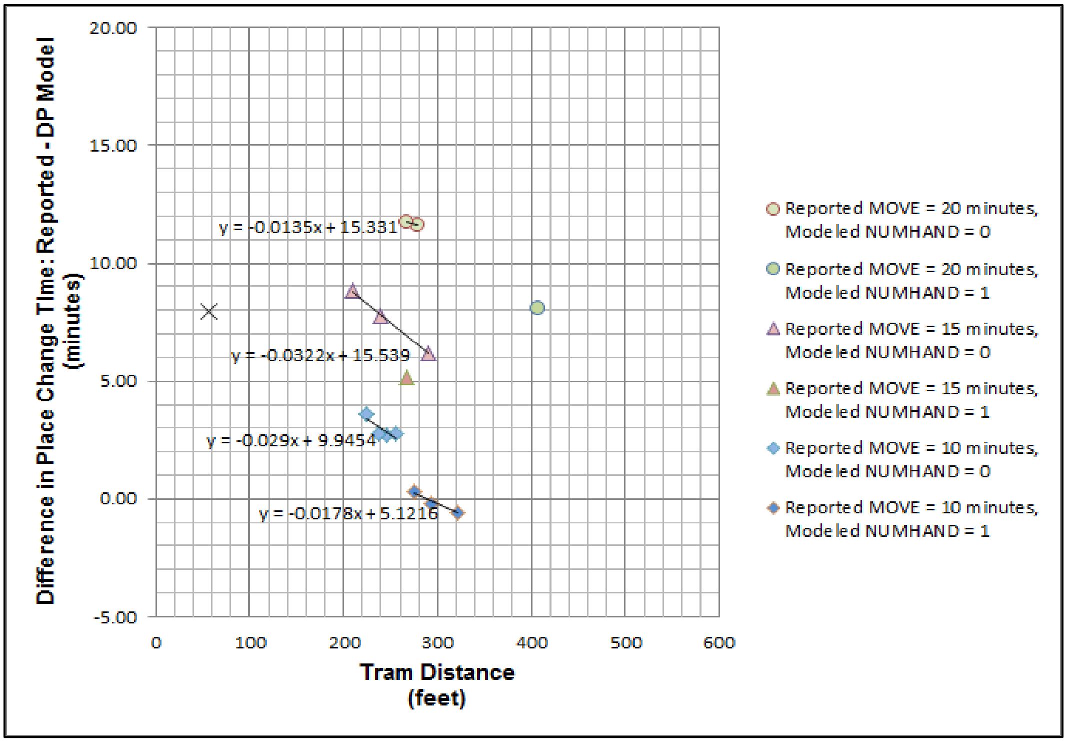

Figure 4.26. Continuous function plot for moves to “turn cuts.”

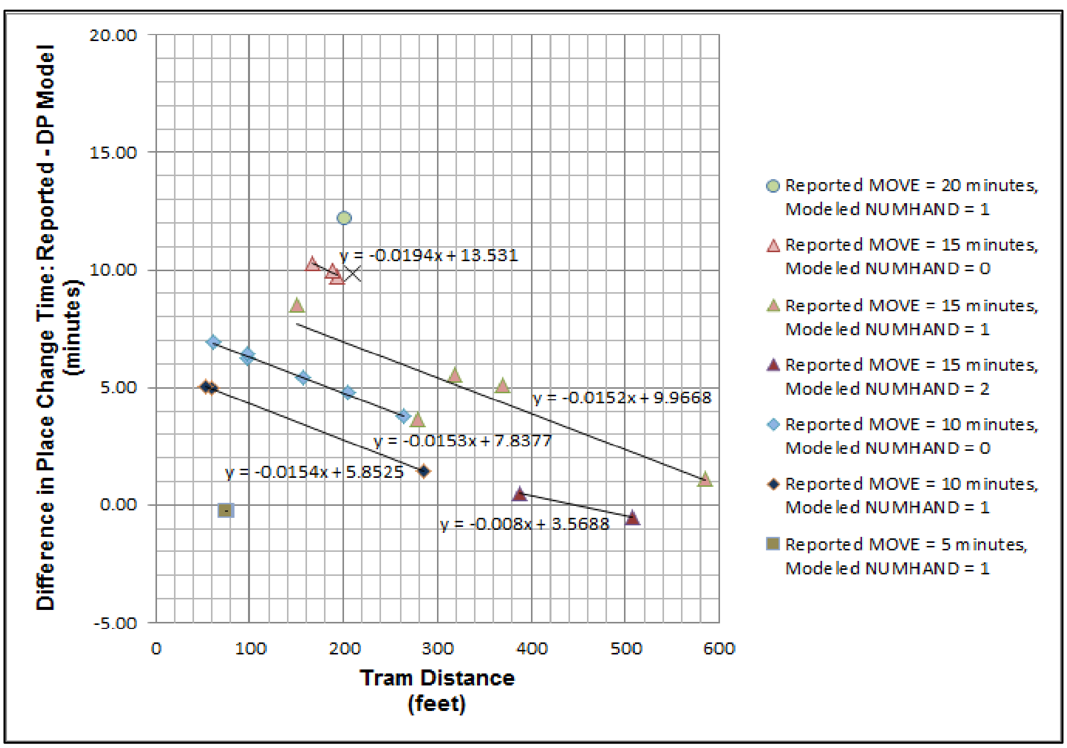

Figure 4.27. Continuous function plot for moves to “head-on cuts.”

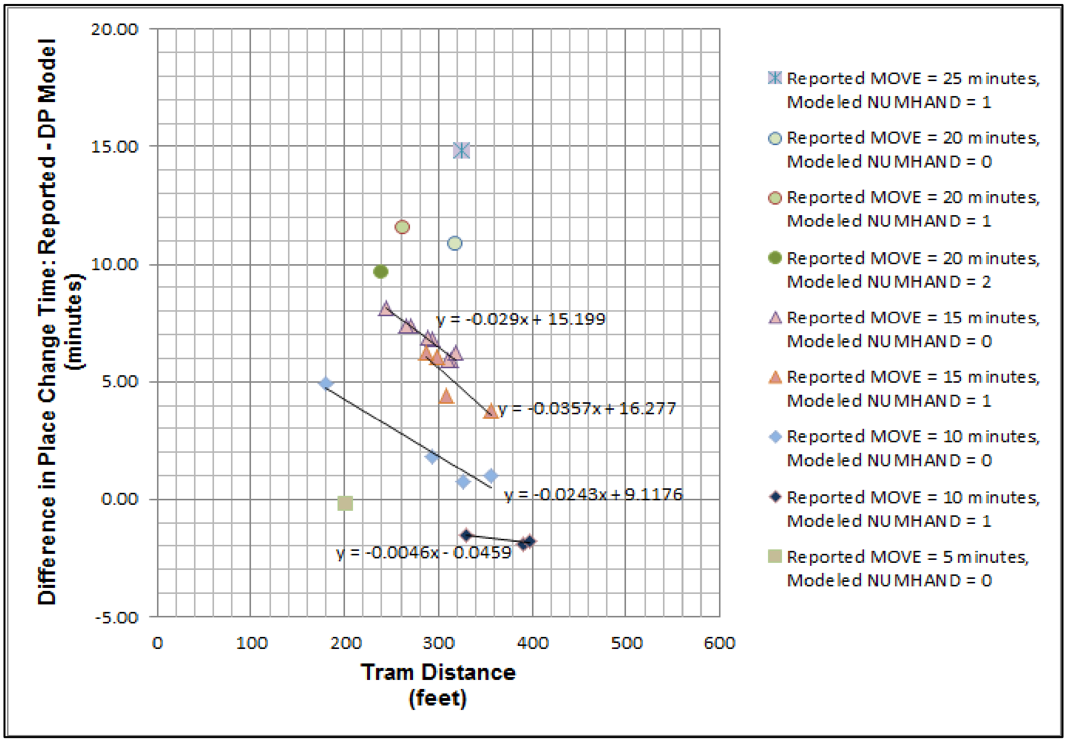

Figure 4.28. Continuous function plot for moves to “crosscut cuts.”

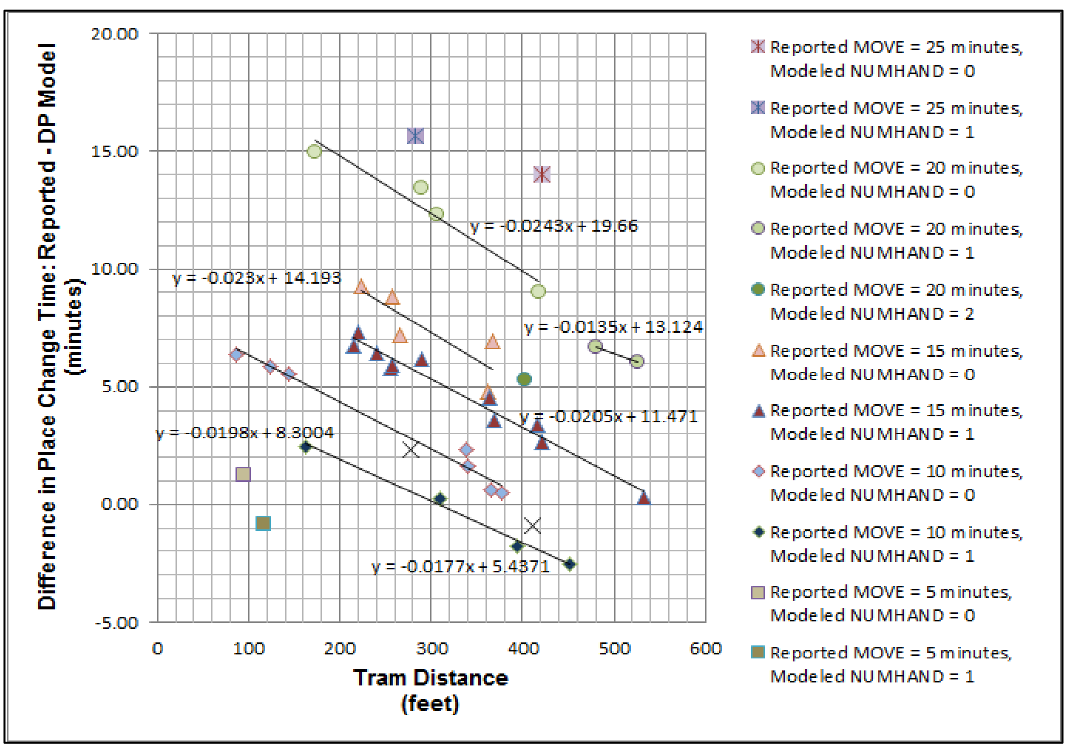

Figure 4.29. Continuous function plot for moves to “hole through cuts.”

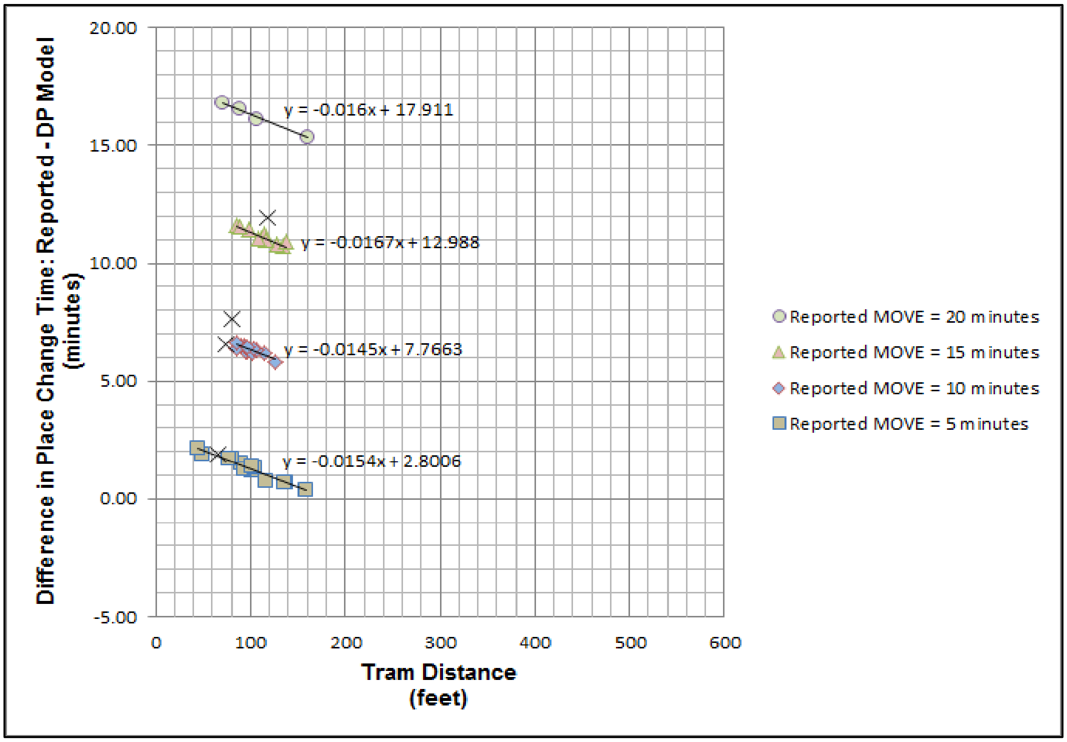

Figure 4.30. Continuous function plot for moves to "double cuts."

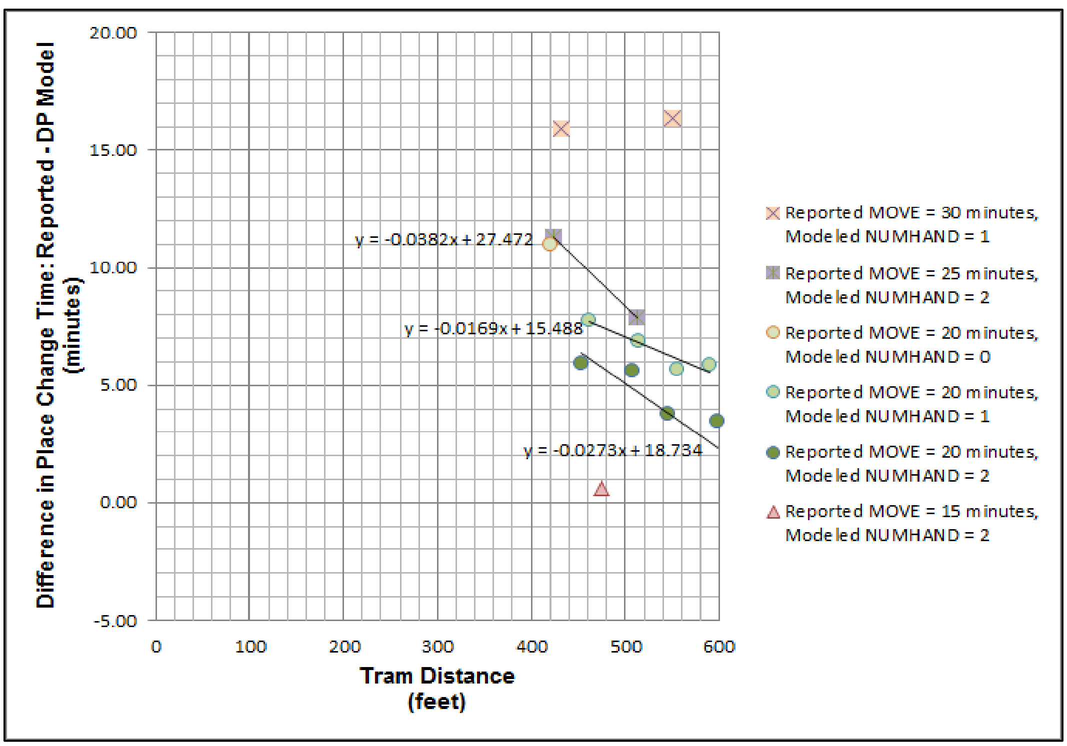

Figure 4.31. Continuous function plot for "reroute" moves.

Trend lines in each of the above plots confirm the importance of tram distance, $TD^{i}_{i-1}(X)$, and CM tram speed, SPD_{CM}, in validating the place change element of cycle time. The DP algorithm is capable of predicting move times to within seconds, whereas move times used to validate the model were reported in 5-minute increments (or to within ± 2.5-minute accuracy). Given this arrangement and assuming no cornering, cable handling, or any other adjustments for "human factors" are required, a continuous function plot with a slope of $-1/SPD_{CM}$ (-0.013 for SPD_{CM} = 74.23 feet per minute from Table 4.5) would result. Slopes of trend lines shown in Figures 4.24 through 4.31 vary slightly from that due to adjustments made for cornering and cable handling; however, given that the variance is slight and that most of the trends on any one figure and even between figures exhibit a parallel relationship, it is reasonable to conclude that the place change component of the DP algorithm has validity.

Because double cutting moves require the least amount of adjustment, Figure 4.30 provides the most uniform set of trend lines. The average slope of the four trend lines shown in that figure is -0.0155. Applying that slope to the range of tram distances for the four categories of moves described in Table 4.14, accounts for as much as half of the variance between predicted and reported move times, as shown in Table 4.15.

Table 4.15. Mean and variance comparison for place change element.

Category of Move	TD Range (feet)		Time Difference (minutes)	Mean Difference (minutes)	Variance Due to TD
	Min	Max			
Move > 400’	401	602	2.71	5.72	47%
240 < Move < 400’	241	400	2.14	5.62	38%
Move < 240’ (not Double)	54	240	2.51	6.12	41%
Double Cut Move	44	161	1.58	7.23	22%

4.4 Chapter Summary

In the initial time study phase of model validation, one model parameter had to be redefined and one new parameter was added. First, the wait-no-car adjustment factor, $WOC_i(X)$, was changed from a haul distance/change-out distance ratio, which did not allow for instances when only one haulage unit was in use, to the more accurate mathematical expression for "wait-no-car" used in the SSP Model. Second, the clean-up car adjustment factor was added to the loading time element to account for the haulage unit that takes the last load from a cut having to reset when the CM makes clean-up passes on both sides of the cut. With those corrections made, it was possible to validate that the DP algorithm provides an accurate assessment of production and place change elements of the CM coal production system.

The need for a "human factor adjustment" to the place change element is suggested by the significant variance between DP model predictions and reported values. Human factors contributing to place change time include but are not limited to breaks for food and water, communications with supervision and co-workers, CM inspection and upkeep, establishing and maintaining ventilation controls, and crossing paths with the roof bolter. These factors occur intermittently but when they do occur, the amount of time added to a place change is significant. Including a human factor would skew predicted place change times for the many cuts where they do not occur. Time study data can quantify human factors allowing mine operators to develop management controls for minimizing their effect on productivity.

CHAPTER 5

CASE STUDY APPLICATION OF THE DP MODEL

5.1 Introduction

The concluding task of this study is a case study application of the developed DP model. Optimal sequences generated by the model are compared with actual mining sequences completed on a CM super-section at the cooperating mine from which validation data was collected. Number of cuts, feet of advance, and cut-cycle time are the productivity measures evaluated in this comparison.

The application case study was completed in two parts. First, the DP model was used to identify optimal mining sequences for the first and last days of the two-week study period. Four different scenarios (left and right side of the super-section constitute different scenarios) are examined by comparing optimal mining sequences (OMS) predicted by the DP model with actual mining sequences (AMS) completed by the mine. Second, the DP model was used to identify an optimal mining sequence for advancing the entire section by three crosscuts, which is compared with the actual mining sequence followed. Three crosscuts of advance is the amount required to complete belt and power moves, which are also a natural cycle in underground room-and-pillar coal mining.

5.2 One-Day, Two-Shift Analysis

Mine characteristics for DP model set-up. The mining section at the cooperating mine is an 11-entry super-section with left- and right-side operations as shown in Figure 5.1. Three intake entries (#7, #8, and #9) bring fresh ventilation air to the face where it is channeled into

Entry #7 at the last open crosscut (LOXC) using tight curtains in Entries #8 and #9. There the intake air splits into two streams, one for each side of the super-section. Intake air on the left side moves right to left from Entry #6 to Entry #11. Intake air on the right side moves left to right from Entry #7 to Entry #11. After sweeping the face area, ventilation air exits the section in return entries on either side (Entries #1, #2, and #3 on the left side; Entries #10 and #11 on the right side). Two travel entries (Entries #4 and #5) provide access to the face for mobile mining equipment.

The left-side CM (LSCM) is configured with the scrubber on the left side of the machine requiring ventilation air to be supplied from the right side of the machine. Thus, the LSCM mines Entry #1 through Entry #6 and crosscuts connecting those entries. The right-side CM (RSCM) is configured with the scrubber on the right side of the machine requiring ventilation air to be supplied from the left side of the machine. Thus, the RSCM mines Entry #7 through Entry #11 and crosscuts connecting those entries. The RSCM also mines crosscuts connecting Entry #6 and Entry #7. While flexibility exists for CMs to go beyond these boundaries, they are maintained constant during this study.

Three haulage units are assigned to each side of the super-section. They discharge onto the same conveyor belt, which is located in the center entry (#6) of the super-section. Because battery-powered ramcars are used for haulage, operators have the flexibility of moving from side to side as needed to enhance productivity. For example, cuts with a long haul distance (HD) and a short change-out distance (COD) have greater “wait-no-car” delays (as defined in Chapter 3), which can be reduced or eliminated by adding haulage units into the haulage cycle. During this case study, the number of haulage units on each side is maintained at a constant level of three.

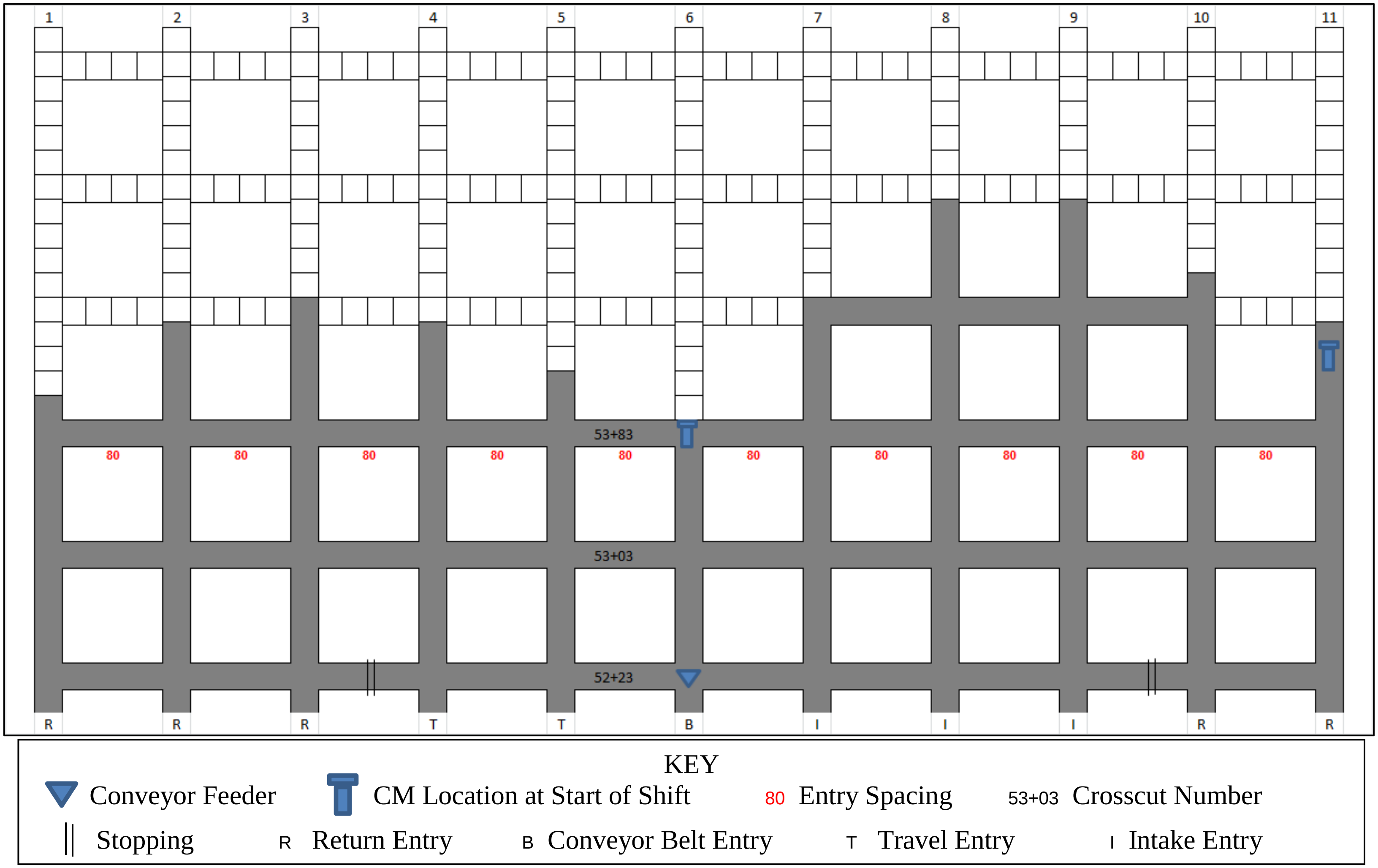

Figure 5.1. Face locations for Day 1 scenario with left-side section consisting of Entries 1-6 and right-side section consisting of Entries 7-11.

For the case study, a primary and two secondary constraints were applied to the DP algorithm. The primary constraint is a regulatory bolting constraint (see constraint matrix in Table 3.1) imposed by federal safety regulations and enforced by the Mine Safety and Health Administration (MSHA). It prohibits any cut inby unsupported roof from being mined. Examples include a cut that can only be accessed through the previous cut just completed and not yet bolted, a cut in an entry inby the first cut in a crosscut started from that entry or the last cut in a crosscut holed into that entry, either of which was the previous cut just completed and not yet bolted. Secondary constraints are operational and can be either ventilation or bolting constraints. The secondary ventilation constraint prohibits mining a cut if it is in an entry that is deep enough to start or complete a crosscut. This gives preference to starting and finishing crosscuts as soon as it becomes possible to mine them in keeping with the guiding policies described in Chapter 3. The secondary bolting constraint prohibits mining a cut that was constrained by the primary bolting constraint on the previous cut. This maintains a buffer between CM and roof bolter minimizing congestion in the face area and delays due to the CM having to wait on the roof bolter to support an area before a cut can be mined. Both secondary constraints are used at the author's discretion and should not be assumed to be standard operating procedure at the mine.

Day 1 scenario. Face positions on the first day of the study period (Day 1) are shown in Figure 5.1. The section conveyor belt had just been extended such that the belt feeder (indicated by the inverted triangle) was within two crosscuts of the face or two crosscuts outby the LOXC. Mine geology and equipment data identified by time studies and used in the validation task as described in Tables 4.1, 4.2, and 4.5 also pertain to the case study application.

Running the DP model consists of a series of iterations using a spreadsheet format as depicted in Table 5.1, which shows the first two iterations for the LSCM on Day 1. During each iteration, the cut with the minimum value for cut-cycle time (CCT), the optimal value function defined in Chapter 3, is selected for mining as indicated by cells with bold borders in Table 5.1. The DP model is not meant to be concerned with delays, expected or unexpected, due to equipment or infrastructure breakdowns, safety issues, or other downtime. Thus, the target recursion value is the total time for two nine-hour shifts (with one-hour overlap) less travel time to the face for the first shift and from the face for the second shift as well as all delays that interrupted production and were accounted for on shift reports. This value is considered to be the time available for production and is the total cut-cycle time for a one-day scenario. Iterations continue until the recursion value reaches the total cut-cycle time achieved by the mine on the day being examined and selection of cuts forms an optimal path or sequence for the CM to follow. This sequence can be used for strategic planning tool by entering it as input to any of the production models described in Chapter 2. It can also be a tactical tool for use by the mine foreman as a daily operations plan.

As described in Section 3.4, CCT values within an upper limit of the minimum are also evaluated as separate branches of a globally optimal sequence. That upper limit, MOVEDIF, is the time it takes the CM to tram the entry spacing distance. Most often these separate paths converge into the same sequence at which point, only the path with the lowest recursion value at the point of convergence is evaluated further. Alternate paths for the LSCM on Day 1 showing CCT and recursion values for each iteration are illustrated in Table 5.2. Only the optimal path is pursued after the twelfth iteration because five of the eight paths had converged. Cycle times (from the validation study) and footages mined following actual mining sequences (AMS) are compared with the same data for optimal mining sequences (OMS) predicted by the DP model in Tables 5.3 and 5.4 for LSCM and RSCM, respectively.

Table 5.1. DP model 1st iteration for the LSCM and Day 1 scenario.

	#1 Entry	#2 Entry	#3 Entry		#4 Entry	#5 Entry	#6 Entry
Cuts Available for 1st Cut	E1.1	E2.1	E3.1	C34.1	E4.1	E5.1	E6.1
Constraint - bolt, vent, both			vent				
Corner/Curtain Factor	1	1.5		1.5	1.5	1.5	1
HD	480	480		400	320	240	240
COD	99	84		89	76	43	92
WOC Factor	0.84	3.04		-0.10	-0.90	-1.49	-2.62
DEPTH	10	30		20	10	17	12
OSD	30	6		6	36	18	18
TRIPS - number of loads	10	21		14	11	14	10
CO Segment	7.87	20.69		12.30	8.57	7.37	6.67
Loading Segment	9.50	18.62		19.13	10.10	13.17	9.70
Production Element	17.37	39.31		31.43	18.67	20.54	16.37
Number of Corners	2	2		3	2	2	0
Direction Changes	2	0		0	0	0	0
Cable Hook-ons	5	5		4	3	2	0
Cable Handling	0	0		0	0	0	0
TD	419	404		329	236	123	12
Place Change Element	10.80	7.85		7.28	5.15	3.41	0.16
DP Model CCT Value	28.17	47.15		38.71	23.82	23.95	16.53
Cuts Available for 2nd Cut	E1.1	E2.1	E3.1	C34.1	E4.1	E5.1	E6.2
Constraint - bolt, vent, both			vent				bolt
Corner/Curtain Factor	1	1.5		1.5	1.5	1.5	
HD	480	480		400	320	240	
COD	99	84		89	76	43	
WOC Factor	0.84	3.04		-0.10	-0.90	-1.49	
DEPTH	10	30		20	10	17	
OSD	30	6		6	36	18	
TRIPS - number of loads	10	21		14	11	14	
CO Segment	7.87	20.69		12.30	8.57	7.37	
Loading Segment	9.50	18.62		19.13	10.10	13.17	
Production Element	17.37	39.31		31.43	18.67	20.54	
Number of Corners	2	2		3	2	2	
Direction Changes	1	1		1	1	1	
Cable Hook-ons	6	6		5	4	3	
Cable Handling	1	1		1	1	1	
TD	443	428		353	260	147	
Place Change Element	11.97	11.76		11.20	9.07	7.33	
DP Model CCT Value	29.34	51.07		42.63	27.74	27.87	

Table 5.2. Paths evaluated for LSCM and Day 1 scenario.

Path										
1st Cut Options					E6.1					
Optimal value $CCT_i(X)$					16.53					
2nd Cut Options		E4.1						E5.1		
Optimal value $CCT_i(X)$		28.99						28.65		
$f_i(X)$		45.51						45.18		
3rd Cut Options	E1.1		E5.1					E4.1		
Optimal value $CCT_i(X)$	28.55		29.49					26.18		
$f_i(X)$	74.07		75.00					71.36		
4th Cut Options	E6.2		E6.2			E1.1				E6.2
Optimal value $CCT_i(X)$	26.05		26.81			28.55				28.24
$f_i(X)$	100.12		101.81			99.91				99.60
5th Cut Options	E5.1		E1.1		E5.2		E6.2			E5.2
Optimal value $CCT_i(X)$	28.79		32.44		26.40		26.05			21.99
$f_i(X)$	128.91		134.25		126.31		125.96			121.59
6th Cut Options	C45.1		E5.2		E6.2		E5.2			E1.1
Optimal value $CCT_i(X)$	32.46		26.40		26.94		20.59			29.63
$f_i(X)$	161.36		160.65		153.26		146.55			151.22
7th Cut Options	E6.3		C45.1		C45.1		C45.1			E6.3
Optimal value $CCT_i(X)$	36.70		32.59		35.40		32.59			33.47
$f_i(X)$	198.07		193.25		188.66		179.14			184.68
8th Cut Options	E5.2		E6.3		E5.3		E6.3			C45.1
Optimal value $CCT_i(X)$	22.83		36.70		39.01		36.70			35.65
$f_i(X)$	220.90		229.95		227.67		215.85			220.33
9th Cut Options	C34.1		E5.3		E6.3		E5.3			E5.3
Optimal value $CCT_i(X)$	43.06		37.71		34.63		37.71			39.01
$f_i(X)$	263.95		267.67		262.29		253.56			259.34
10th Cut Options	E6.4		C34.1		C34.1		C34.1			E6.4
Optimal value $CCT_i(X)$	32.79		43.32		46.11		43.32			29.29
$f_i(X)$	296.74		310.99		308.40		296.88			288.62
11th Cut Options	E5.3		E6.4	E1.2		C45.2	E6.4			C34.1
Optimal value $CCT_i(X)$	37.88		32.79	49.39		49.69	32.79			46.27
$f_i(X)$	334.62		343.78	357.79		358.09	329.68			334.89
12th Cut Options	E1.2	C45.2	C45.2	E6.4		E6.4	C45.2		C45.2	E1.2
Optimal value $CCT_i(X)$	46.06	46.77	49.72	28.39		26.10	49.72		49.69	49.39
$f_i(X)$	380.67	381.39	393.50	386.18		384.19	379.39		384.58	384.28
13th Cut Options							C34.2			
Optimal value $CCT_i(X)$							49.36			
$f_i(X)$							428.75			
14th Cut Options							E6.5			
Optimal value $CCT_i(X)$							32.60			
$f_i(X)$							461.35			
15th Cut Options							C45.3			
Optimal value $CCT_i(X)$							52.88			
$f_i(X)$							514.24			

Table 5.3. Productivity comparison of AMS and OMS for LSCM on Day 1.

	AMS	OMS	Difference
Cumulative Distance Mined (feet)	276	252	-24
Cuts Mined	17	18	+1
Cumulative Cycle Time (minutes)	590	596	+6
Cumulative Loading Time (minutes)	232	219	-13
Cumulative Change-out Time (minutes)	239	209	-30
Cumulative Place Change Time (minutes)	120	168	+48

Table 5.4. Productivity comparison of AMS and OMS for RSCM on Day 1.

	AMS	OMS	Difference
Cumulative Distance Mined (feet)	314	348	+34
Cuts Mined	12	14	+2
Cumulative Cycle Time (minutes)	544	556	+12
Cumulative Loading Time (minutes)	216	235	+19
Cumulative Change-out Time (minutes)	245	225	-20
Cumulative Place Change Time (minutes)	82	97	+15

At the cooperating mine, limestone in the immediate roof strata provides a stable mine opening that is easily supported. When the limestone thins or disappears, there is a corresponding increase in thickness of the immediate roof rock known as Energy shale, which is a very weak formation that typically falls before it can be supported. When such conditions are encountered, the mine plan is to make shorter cuts than the standard 32-ft cut. Shorter cuts vary from as little as 5-ft depth to as much as 25-ft depth with the majority being 10 feet deep. On Day 1, the LSCM encountered challenging mining conditions due to no limestone in the immediate roof strata. Since the DP model gives preference to short cut with their inherently shorter production cycle times resulting from fewer change-outs due to fewer haulage units loaded, the OMS predicted by the DP model for LSCM on Day 1 focuses on short cuts. As seen in Table 5.3, the OMS has one more cut than the AMS, but mined footage was less. Pursuing shorter cuts leads to greater place change times and smaller change-out times. For the RSCM, conditions were good except for Entry #7, which improved during the course of the day, and the DP model predicts an OMS that provides greater productivity than the AMS. The OMS included two additional cuts and an additional 34 feet in the same amount of cycle time.

Both AMS and OMS were entered as input in the SSP Model described in Chapter 2 with results compared in Tables 5.5 and 5.6. Figures 5.2 and 5.3 provide a visual comparison of actual and optimal mining sequences for the Day 1 scenario.

Table 5.5. LSCM SSP Model output for Day 1 AMS and OMS.

	AMS	OMS	Difference
Cumulative Distance Mined (feet)	276	252	-24
Cumulative Cycle Time (minutes)	494	474	-20
Cumulative Loading Time (minutes)	209	191	-18
Cumulative Change-out Time (minutes)	124	103	-21
Expected Mining Rate (tons/minute)	3.49	3.32	

Table 5.6. RSCM SSP Model output for Day 1 AMS and OMS.

	AMS	OMS	Difference
Cumulative Distance Mined (feet)	314	348	+34
Cumulative Cycle Time (minutes)	501	539	+38
Cumulative Loading Time (minutes)	201	249	+48
Cumulative Change-out Time (minutes)	193	161	-32
Expected Mining Rate (tons/minute)	3.92	4.04	

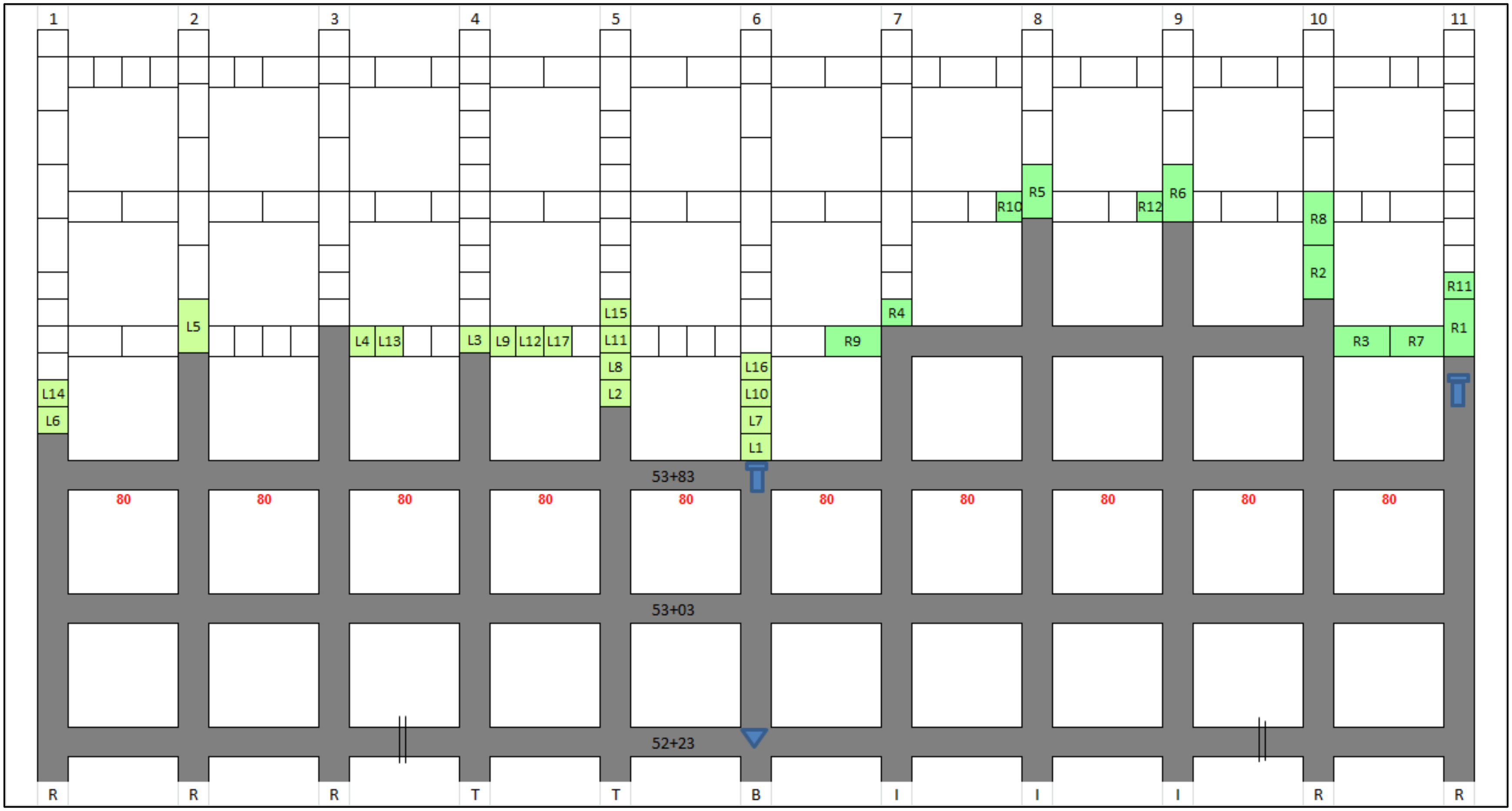

Figure 5.2. Actual mining sequence for Day 1 scenario with left-side section consisting of Entries 1-6 and right-side section consisting of Entries 7-11.

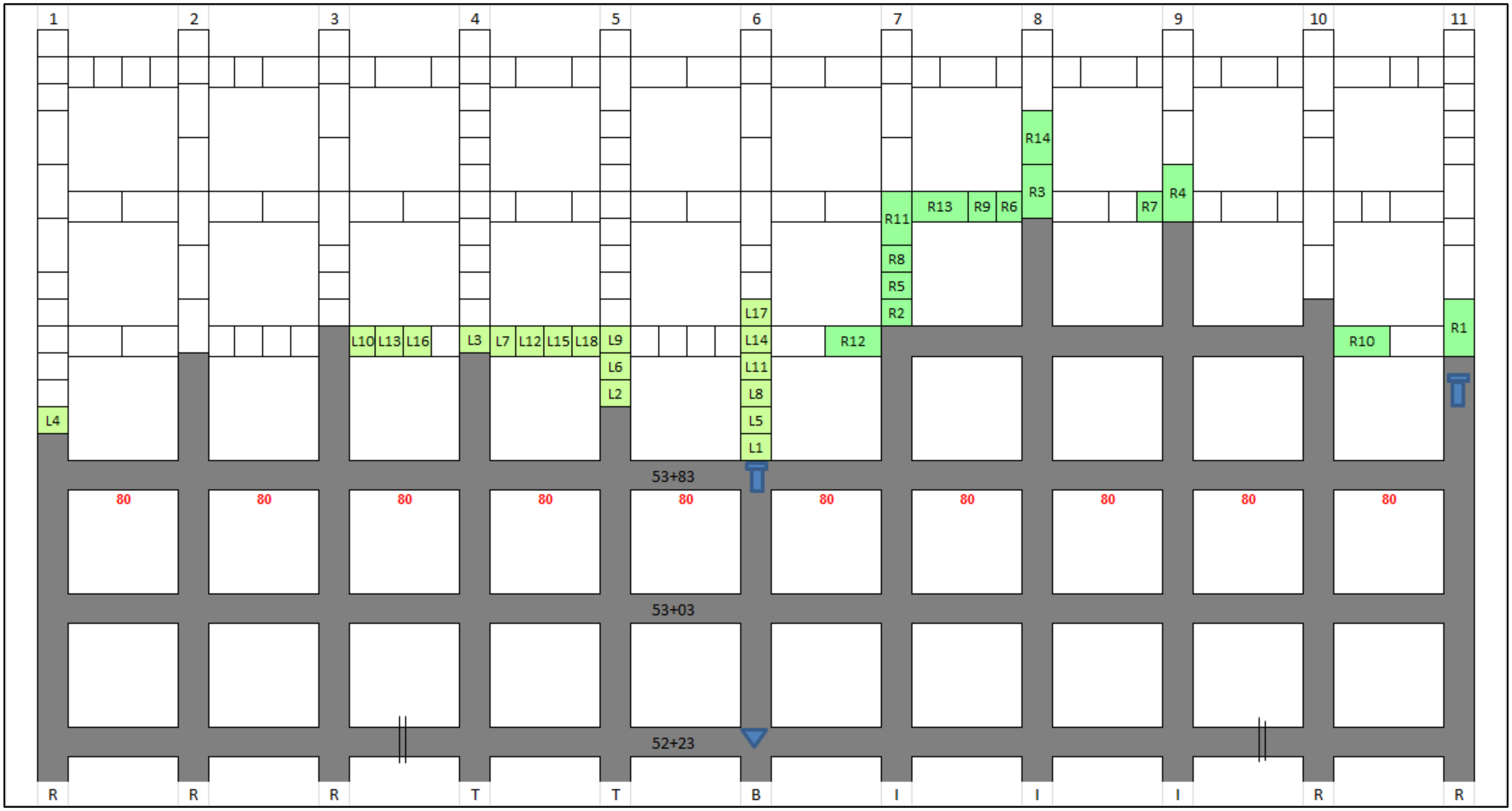

Figure 5.3. Optimal mining sequence suggested by DP model for Day 1 scenario with left-side section consisting of Entries 1-6 and right-side section consisting of Entries 7-11.

Day 12 scenario. Time study data collected on Day 12 provides the most accurate information for comparison. Mining conditions included both good and bad, as was the case for every day of the study. On Day 12, the LSCM faced poor roof conditions in Entries #1 and #4; the RSCM faced poor roof conditions in Entry #10.

Cycle times and footages mined following the actual mining sequences are compared with the same data for optimal mining sequences in Tables 5.7 and 5.8 for LSCM and RSCM, respectively. SSP Model results of AMS and OMS for both sides are compared in Tables 5.9 and 5.10 for LSCM and RSCM, respectively. Figures 5.4 and 5.5 provide a visual comparison of actual and optimal mining sequences for the Day 12 scenario.

Day 12 results are similar to Day 1 results in that the DP model once again showed preference for shorter cuts leading to an OMS for the LSCM that appears to be suboptimal when compared with the AMS; however, differences between OMS and AMS productivity measures are less than 5%, which confirms validation efforts described in Chapter 4. The author was present with the RSCM crew for the first half of Day 12 and offered suggestions on the mining sequence to be followed that day, which may explain why there is little difference between the AMS and the OMS for the RSCM.

Table 5.7. Productivity comparison of AMS and OMS for LSCM on Day 12.

	AMS	OMS	Difference
Cumulative Distance Mined (feet)	371	357	-14
Cuts Mined	15	16	+1
Cumulative Cycle Time (minutes)	653	653	0
Cumulative Loading Time (minutes)	252	255	3
Cumulative Change-out Time (minutes)	282	286	4
Cumulative Place Change Time (minutes)	119	112	-7

Table 5.8. Productivity comparison of AMS and OMS for RSCM on Day 12.

	AMS	OMS	Difference
Cumulative Distance Mined (feet)	370	376	+6
Cuts Mined	16	16	0
Cumulative Cycle Time (minutes)	696	687	-9
Cumulative Loading Time (minutes)	253	259	+6
Cumulative Change-out Time (minutes)	326	313	-13
Cumulative Place Change Time (minutes)	116	115	-1

Table 5.9. LSCM SSP Model output for Day 12 AMS and OMS.

	AMS	OMS	Difference
Cumulative Distance Mined (feet)	371	357	-14
Cumulative Cycle Time (minutes)	589	572	-17
Cumulative Loading Time (minutes)	260	257	-3
Cumulative Change-out Time (minutes)	184	171	-13
Expected Mining Rate (tons/minute)	3.94	3.9	

Table 5.10. RSCM SSP Model output for Day 12 AMS and OMS.

	AMS	OMS	Difference
Cumulative Distance Mined (feet)	370	376	+6
Cumulative Cycle Time (minutes)	631	641	+10
Cumulative Loading Time (minutes)	264	282	18
Cumulative Change-out Time (minutes)	225	212	-13
Expected Mining Rate (tons/minute)	3.66	3.67	

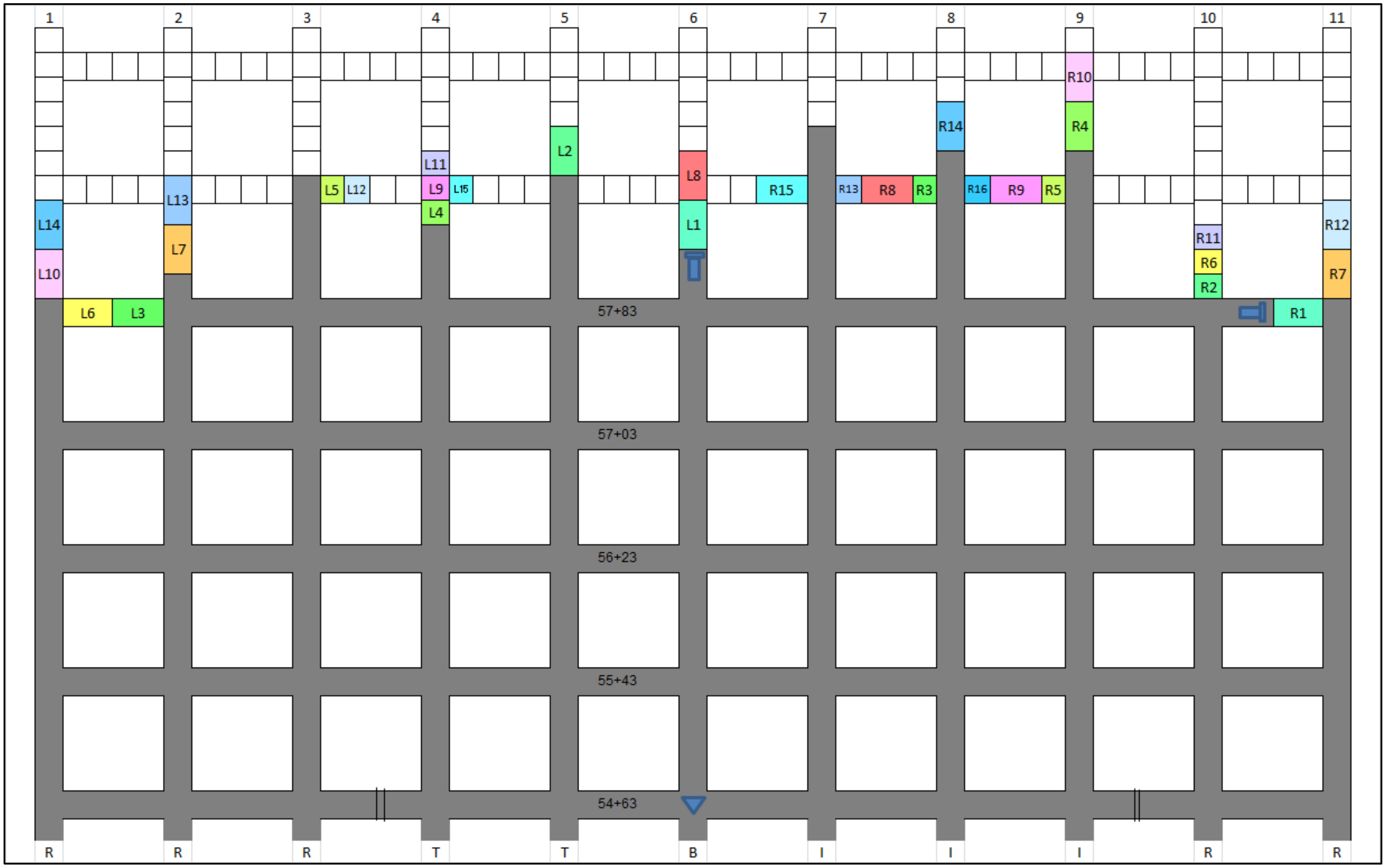

Figure 5.4. Actual mining sequence for Day 12 scenario with left-side section consisting of Entries 1-6 and right-side section consisting of Entries 7-11.

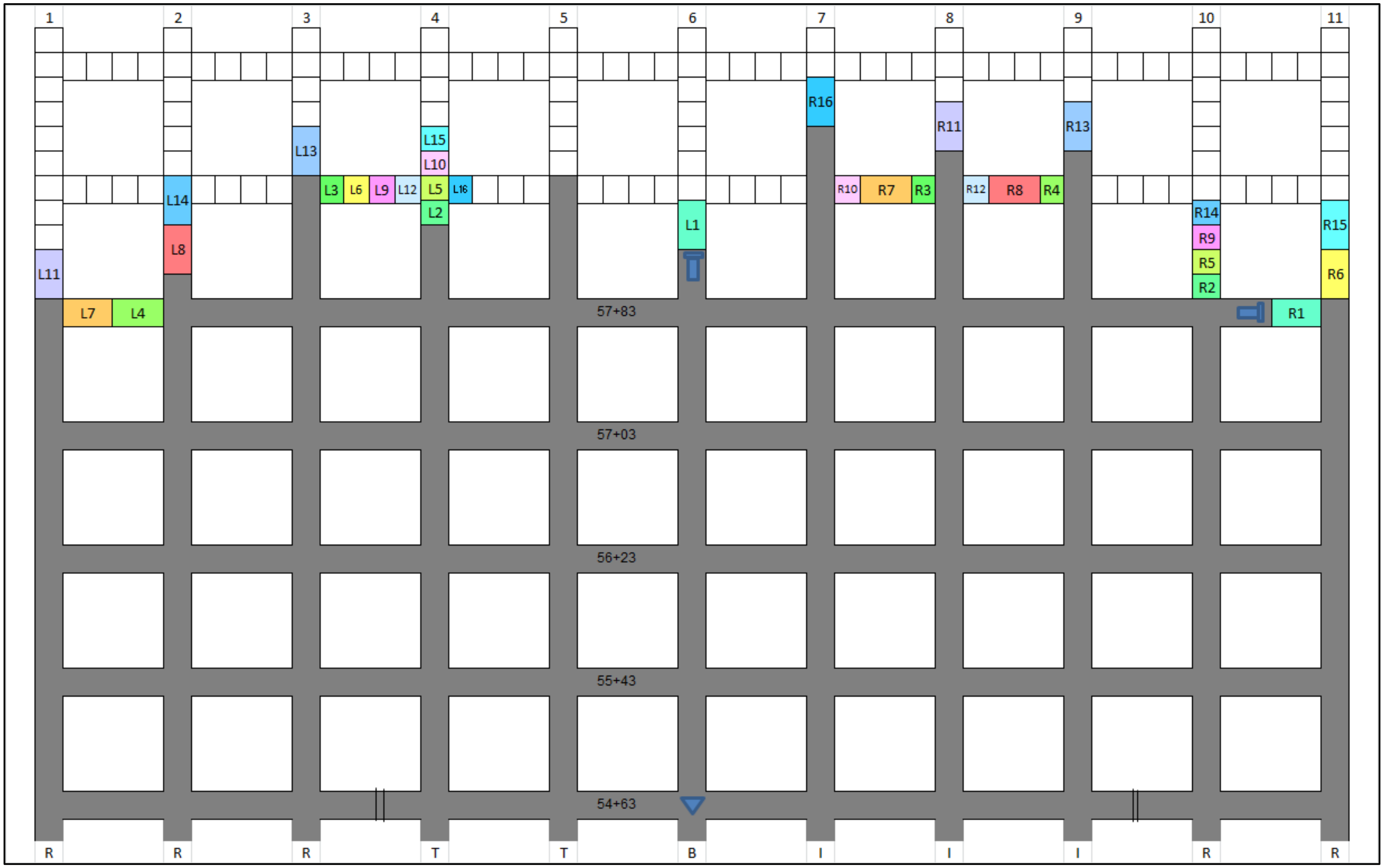

Figure 5.5. Optimal mining sequence suggested by DP model for Day 12 scenario with left-side section consisting of Entries 1-6 and right-side section consisting of Entries 7-11.

5.3 Seven-Day, Three-Crosscut Analysis

Identifying an OMS for one day or one shift is a tactical application of the DP model. The model can also be used for strategic planning. One such opportunity is to identify an OMS between belt and power moves. As room-and-pillar mining sections advance, infrastructure must be moved to support mining. This includes the section conveyor belt. Conveyor belt is typically packaged in 500-ft lengths. Since there is a carrying and return side to the conveyor, a 500-ft roll will advance the belt by 250 feet. With crosscuts mined on 80-ft centers, the conveyor has to be advanced every three crosscuts. Advancing the section conveyor is called a "belt move." Day 1 of the mine study was the first production day immediately following a belt move. The conveyor belt was advanced again after Day 7 of the study. Thus, the DP model was used to determine an OMS for seven days of mining.

Because of challenging roof conditions already described, the standard cut depth of 32 feet could not be uniformly followed throughout the section. To provide an accurate comparison between AMS and OMS, a cut depth for every cut to be made during three crosscuts of advance was determined using shift report data. Thus, each cut in the OMS, no matter when it was mined, had the same depth as that cut in the AMS.

The mine operates with two production shifts and an idle shift. "Hot seat" shift changes, where the afternoon shift crew arrives at the working place before the day shift crew leaves, enable mining to continue with little interruption during shift change; however, during the idle night shift, CMs are often used to load out gob or grade road bottoms. The night shift crew then "spots" the CM at a cut for the beginning of day shift. DP modeling did not incorporate such an option but rather progressed without interruption from start to finish.

Results for the seven-day, three-crosscut analysis are shown in Tables 5.11 and 5.12 for the LSCM and RSCM, respectively, with color coding to match the modeled AMS and the predicted OMS shown on Figures 5.6 and 5.7, respectively.

Table 5.11. Productivity comparison of AMS and OMS for LSCM during three crosscuts of advance over seven days.

Day	AMS			OMS			Difference		
	Cuts	Feet Mined	CCT (min)	Cuts	Feet Mined	CCT (min)	Cuts	Feet Mined	CCT (min)
1	17	276	590	18	252	596	+1	-24	+6
2	15	221	508	17	229	503	+2	+8	-5
3	17	292	521	14	285	538	-3	-7	+17
4	15	375	662	15	384	624	0	+9	-38
5	12	343	600	12	375	598	0	+32	-2
6	13	384	612	13	357	610	0	-27	-2
7	13	394	651	14	402	675	+1	+8	+24
Totals	102	2285	4144	103	2284	4144	+1	-1	0

Table 5.12. Productivity comparison of AMS and OMS for RSCM during three crosscuts of advance over seven days.

Day	AMS			OMS			Difference		
	Cuts	Feet Mined	CCT (min)	Cuts	Feet Mined	CCT (min)	Cuts	Feet Mined	CCT (min)
1	12	314	544	14	348	556	+2	+34	+12
2	13	353	601	13	365	606	0	+12	+5
3	12	336	554	12	297	533	0	-39	-21
4	14	329	638	13	355	635	-1	+26	-3
5	9	245	419	8	214	415	-1	-31	-4
6	12	292	544	12	282	552	0	-10	+8
7	15	273	590	15	255	590	0	-18	0
Totals	87	2142	3890	87	2116	3887	0	-26	-3

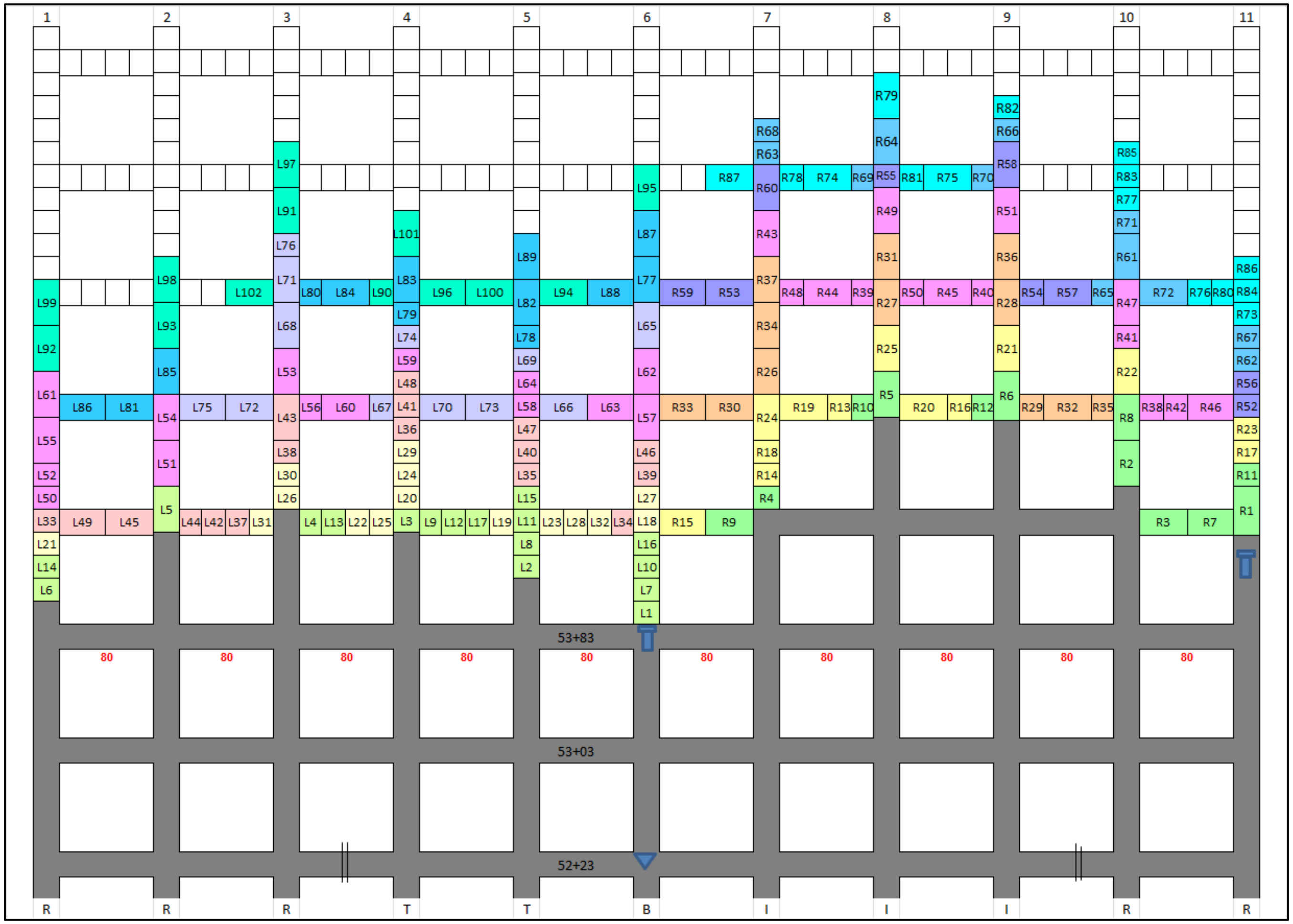

Figure 5.6. Actual mining sequence for three crosscuts of advance with left-side section consisting of Entries 1-6 and right-side section consisting of Entries 7-11.

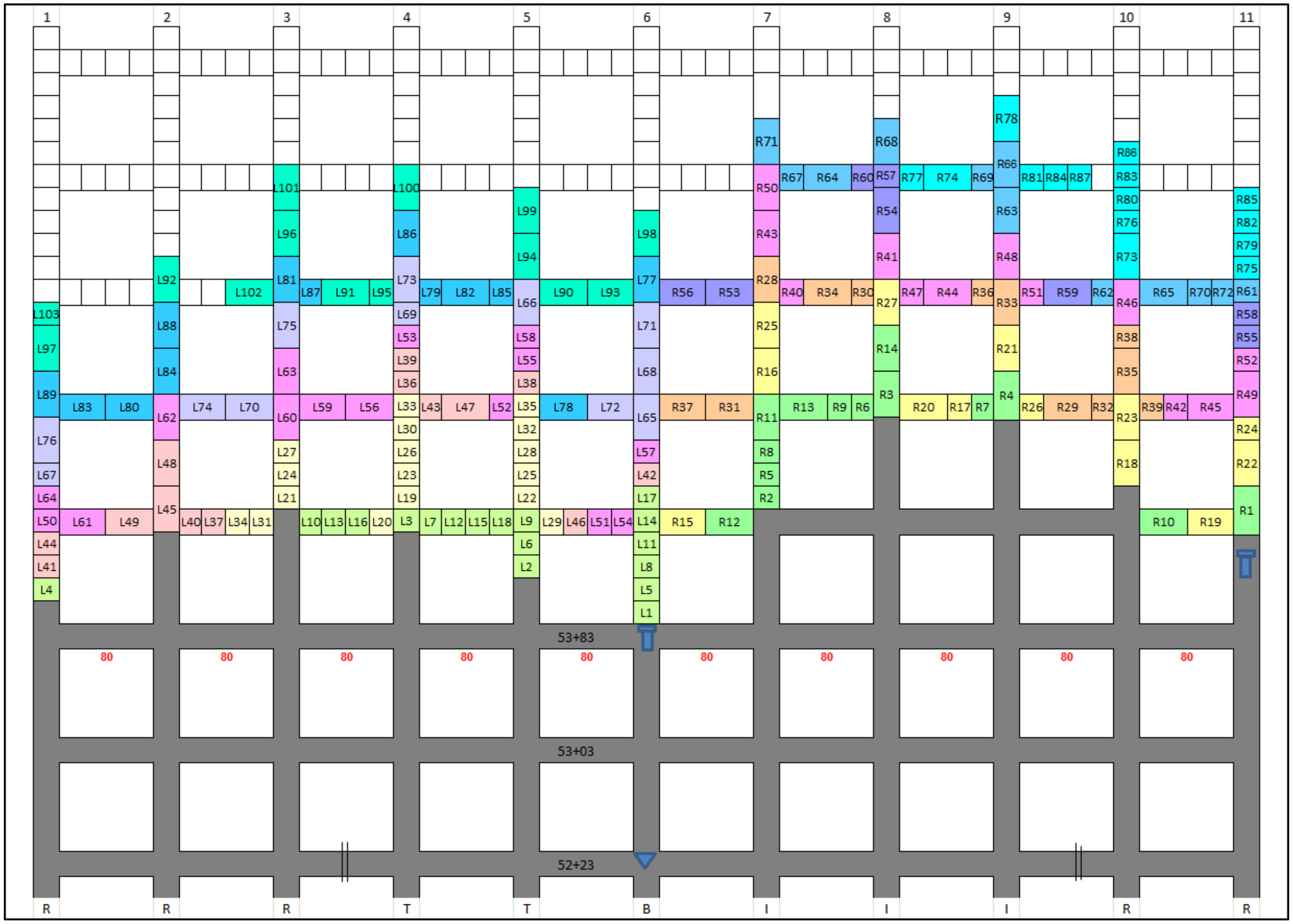

Figure 5.7. Mining sequence predicted by DP model for three crosscuts of advance with left-side section consisting of Entries 1-6 and right-side section consisting of Entries 7-11.

Productivity differences between AMS and OMS are related to the number of short cuts made on a given day. On Day 1, in the AMS mined by the LSCM included one 32-ft cut (L5) and 16 shorter cuts of varying depths while the DP model suggested a cut sequence composed entirely of short cuts resulting in OMS having one more cut but mining 24 feet less. On Day 2, both the AMS and the OMS were composed entirely of short cuts; however, the OMS stayed within the three center entries while the AMS was spread out across the section resulting in a favorable OMS with two more cuts and eight more feet mined in five less minutes than the AMS. On Day 4, the AMS mined by the LSCM included four short cuts in the 12 cuts mined that day whereas the DP model predicted an OMS that included only two short cuts resulting in a difference between the two sequences of 32 feet with the OMS being more favorable. The situation was reversed on Day 5. For the RSCM, total feet mined by the OMS were less than the AMS because the OMS, with its tendency to give preference to short cuts, became confined to Entries #10 and #11 where poor roof conditions prevailed.

If the OMS for three crosscuts of advance is examined from the perspective of two of the guiding policies and practices described in Chapter 3, it can be observed that the predicted OMS provided no improvements in the number of crosscuts holed through against the ventilation air current while it did incorporate more double cuts. The OMS did reveal that crosscuts that are turned do not have to always be the same crosscut. The mine consistently turns the crosscut between Entries #3 and #4 but the OMS showed where that crosscut could be mined head-on (see Cuts L56 and L59 in Figure 5.7) The OMS included two more double cuts than the AMS for the LSCM while the number of double cuts was the same for the RSCM.

5.4 Chapter Summary

Overall, results of the case study application described in this chapter appear to provide a second confirmation of the validation effort described in Chapter 4 suggesting that the DP algorithm accurately describes the CM production system in mathematical terms. It was hoped that the DP model would predict OMS options that bettered in every respect the AMS that was reported and observed. While this did not happen, the DP model does provide a tool for identifying reasonable cut sequences for use in production modeling. It also provides a means for evaluating productivity improvement potential for cut sequence scheduling.

The case study application was a time consuming, tedious process of manipulating spreadsheet data in an iterative process all done "by hand." For the model to see any real application in the industry, the ability to use it in a time effective fashion requires creating computer programming to integrate data input, modeling calculations, and sensitivity analysis opportunities.

CHAPTER 6

SUMMARY, CONCLUSIONS, AND RECOMMENDATIONS

6.1 Research Summary and Specific Accomplishments

The underground coal mine is an atypical yet fascinating laboratory. The industry has a storied past that underscores its continued relevance as a critical global energy supplier. The continuous miner (CM) is the centerpiece of the most prevalent production system used in underground coal mines in the US. Their introduction more than half a century ago spurred a concentrated industrial engineering effort in the mining engineering community producing several production models and other engineering tools that contributed to tremendous growth in mine productivity as was shown in Figure 1.1. With a reverse in mine productivity trends during the past decade, there is renewed interest in reapplying proven methods and developing new techniques for enhancing mine productivity. This research study is only part of a greater effort being conducted by Southern Illinois University (SIU) research teams working on productivity optimization concepts as well as mine safety and health improvements for underground coal mines in the Illinois Basin that can be obviously extended to the industry worldwide.

The CM production system follows a mining sequence composed of multiple repetitive cycles. The mining sequence itself is routinely repeated and, due to the tremendous flexibility of the system providing numerous sequencing options, it is easy to become complacent with mediocre results without making a conscientious effort toward optimization. Without diligently focusing on keeping the CM at the face cutting and loading coal, productivity can be needlessly sacrificed. To assist mine operators with maintaining that focus, the overall objective of this dissertation was to utilize a known dynamic programming (DP) optimization technique to develop an algorithm for identifying optimal mining sequences for room-and-pillar mining.

To accomplish this objective, the first step was developing a mathematical description of the production process for use as an algorithm in a DP model. In cooperation with industry professionals, six priority policies and practices were identified to guide development. The algorithm developed to satisfy these guiding policies predicts cycle times for feasible cuts which are evaluated based on a selection criterion to minimize cut-cycle time. This time-based optimal value function was selected over what might be considered the more attractive option of production output because it relies on time study data, which is easier to obtain and more accurate than production data for individual mining sections. The optimal value function developed in this study is comprised of production and place change time elements with the production element being separated into loading and change-out time components. As an application of the DP concept, the algorithm developed in this study is unique in that it is based on both path-specific and state-specific parameters rather than just state-specific parameters. To the best of the author's knowledge, it is the first known application of DP to underground room-and-pillar mining sequence optimization.

The reasonableness and accuracy of the mathematical model were verified by modeling an actual mining sequence completed over a two-week period of study and comparing model output with cycle times measured by means of time studies conducted at a cooperating mine as well as with cycle times reported by mine foremen at the same mine. The validity of the model was shown by charting frequency distributions of production cycle times for more than 300 cuts classified by cut depth and change-out distance and by developing continuous function plots of place change cycle times for a similar number of moves between cuts. Those parameters for which mine-specific data must be collected in order to use the DP model were identified in the time study process. In addition to verifying the reasonableness and accuracy of the DP model, the validation effort also led to some fine-tuning of the mathematical model.

Finally, in a case study application of the DP model, it was used to predict one-day and seven-day sequences for the same two-week time period that was studied at the cooperating mine. These predicted sequences were evaluated for optimality by comparison with actual mining sequences followed. The case study application attempted to demonstrate the usefulness of the DP model as both a strategic planning tool for identifying optimal sequencing patterns that may be used to specify standard operating procedures and a tactical planning tool for providing a day-to-day plan to assist mine foremen with decision making.

6.2 Conclusions

The following conclusions can be made regarding results and outcomes of this study:

1) Production cycle times (ranging from 26 minutes for cuts less than 15 feet to 50 minutes for cuts greater than 28 feet) predicted by the DP model are within 30 seconds of reported data for all categories of cuts. Coefficient of variation values are consistently uniform at about 20% indicating that the model provides an accurate mathematical description of the production element of total cut-cycle time for all categories of cuts based on cut depth.
2) Place change cycle times predicted by the DP model (ranging from four minutes for double cut moves to 13 minutes for moves greater than 400 feet) portray much greater consistency than reported place change cycle times (ranging from 11 minutes for double cut moves to 19 minutes for moves greater than 400 feet). Taking into account tram distance accounts for as much as half of the difference between predicted and reported

values. It is hypothesized that the influence of human factors accounts for the remainder of the variability observed when comparing model output with reported data for the place change element of total cut-cycle time.

3) For each cut, the overall production cycle time consists of a number of repetitive loading and change-out cycles. The ability to separate the production cycle time element into these components increases accuracy and decreases variability. The place change element is subject to greater variability because it is difficult to break it down into smaller components.

4) While operating under much more constrained conditions for ventilation and bolting, the DP model predicted mining sequences with as good as or better productivity than that reported for sequences actually followed by mine operators during the two-week time period of this study.

5) The DP model in its present state shows a clear preference for shorter cuts. This is because shorter cuts require fewer haulage units to be loaded meaning fewer change-outs occur.

6) Changing conditions at the mine cooperating with this study presented the challenge of having cuts of various depths, which made it difficult to tactically predict optimal mining sequences for a single shift; however, over longer time periods of a full day or a period of days such as between belt moves, the DP model did provide reasonable predictions for mining sequences that were as productive as the actual mining sequence followed during the study period.

7) Perhaps the greatest value of the DP optimization technique for identifying CM production system mining sequences is as an educational and training tool. The DP

model presented in this study is simple and provides a concise and accurate mathematical representation of the production process that can aid the user in identifying, quantifying, and manipulating critical parameters that affect mine productivity.

6.3 Recommendations for Future Work

Interest from industry shown during the course of developing the DP algorithm and model has motivated the author to continue this work. The effort was born out of industry recommendations to eliminate the heuristic nature of cut sequence evaluation and streamline the process of entering cut sequences as input to production models. Such a model has been developed but in its current form as a spreadsheet requiring tedious manipulation, it would be of little use to the typical mine engineer. For it to become an effective tool, a person skilled in Excel® programming will have to build an optimal mining sequence (OMS) module for integration with the SSP Model or some other production modeling software.

The case study application exercise identified the DP model's sensitivity to cut depth. When cut depth is a function of mining conditions, with shorter cuts required to maintain the integrity of mine openings in areas where poor geology is encountered, there is some merit to the argument that selecting shorter cuts is an optimal policy because it promotes advancing through the difficult area as quickly as possible. This is difficult to quantify with typical productivity measures such as feet of advance, number of cuts, and cycle time. The author believes that additional research is needed to refine the DP algorithm such that it is less dependent on any single parameter.

The real value of mine production modeling is that it requires thinking through the process as part of model development. This is particularly true when it comes to the problem of mining sequences. While it is hoped that the optimization tool presented in this study will be developed further to a more useful state, the author advocates that users of the DP model take the time to understand the mathematics of the model and how the algorithm works. Doing so will help to insure that strategic or tactical planning information generated by the model is applied properly.

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APPENDICES

APPENDIX A

SAMPLE SHIFT REPORT

xxxxxxxx indicates information withheld to protect company identity

Daily Mine Summary

12-19-11

Everyone worked safe today

Production:	7-4 Shift	831’
	3-12 Shift	765’
	Daily Total	1596’

7-4 Shift

Mine Manager:	xxxxxxxxxxx	
Manpower:	Safety	xxx
	Maintenance	xxx
	Operations	xxx
Safety:	Everyone worked safe this shift	
	No accidents or injuries reported	
Safety Observation:	xxxxxxxxxxx	
Safety Observation:	xxxxxxxxxxx	
Safety Observation:	xxxxxxxxxxx	
Inspectors:	xxxxxxxxxxx	

Section #1: xxxxxxxxxxxx

L/S Report

Foreman: xxxxxxxxxxx

Time Period	Location	Cut Depth	Line Curtain Air	Description
(7:35-7:50)				Pre-ops
7:50-8:45	#6	33’	8116 cfm	
(8:45-9:00)				knot in water line, move to E5
9:00-9:45	#5	32’	8062 cfm	
10:00-10:45	#2L	32’	48710 cfm	
11:00-11:35	#4	17’	8566 cfm	
11:45-12:10	#3R	5’	9202 cfm	
12:25-1:10	#2L	30’	8018 cfm	
1:25-2:05	#2	32’	9254 cfm	
2:30-3:15	#6	32’	8180 cfm	
(3:15-EOS)				move to #4, work on ventilation

Miscellaneous:
Put up 2 boards
Watered roads at 12:00

Start Footages:	End Footages:
#6: 58+30	
#5: 58+72	59+05
#4: 58+35	58+52
#2: 58+12	58+44

Bolted: #6, #2L, #5, #4, #2L
Unbolted: #2, #3R, #6

Downtime:
(7:35-7:50) Pre-ops
(8:45-9:00) Knot in water line, move to E5

Section #1: xxxxxxxxxxx

R/S Report
Foreman: xxxxxxxxxxx

Time Period	Location	Cut Depth	Line Curtain Air	Description
(7:30-7:40)				Pre-ops, walk & hang cable, put up sights
7:40-8:10	#10R	25’	8652 cfm	
8:23-8:50	#10	10’	12250 cfm	
9:05-9:45	#8L	20’	8410 cfm	
9:53-10:43	#9	32’	8718 cfm	
10:56-11:38	#9L	20’	8520 cfm	
11:49-12:05	#10	10’	9160 cfm	
12:18-1:10	#11	25’	12650 cfm	
1:20-2:15	#8L	32’		(1 RC)
(2:15-2:30)				WOB, pushing gob, dusting
2:40-3:25	#9L	20’	8250	

Miscellaneous:
Moved 3 cars of gob to EN7
Changed battery and serviced RC301 twice
Change battery on RC 302, 303, 304
Started building stopping @56+23 – ¾ sealed
Put up 6 boards
Watered roads twice
Pushed feeder at EOS

Start Footages:	End Footages:
#7: 59+03	59+39
#8: 58+99	
#9: 58+79	59+13
#10: 57+93	58+32
#11: 58+03	

Bolted: #10, #10R, #7L, #8L, #10, #8L
Unbolted: #11 – 25’, #9L – 30’, #8L – 10’

Downtime:
(7:30-7:40) Pre-ops, walk & hang cable, put up sights
(2:15-2:30) WOB, pushing gob, dusting

Section Summary:

Equipment:
L/S: CM203, RB402, RB403, RC303, RC305, RC306, CS505
R/S: CM204, RB404, RC301, RC303, RC304, CS503, Minitrac 701

Geology:
L/S: losing 2'-3' of rock in #4, all other areas good, floor good
R/S: top adverse in #10 & #11, floor good

Air Readings:

L/S:	Intake 31612	Return 50370
R/S:	Intake 28910	Return 37250

L/S:	**Coal Cut: 214'**	**Bolted: 145'**	**Unbolted: 69'**
RS:	**Coal Cut: 204'**	**Bolted: 159'**	**Unbolted: 65'**

APPENDIX B

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Address of Licensee:	5776 Coal Drive, Suite 200, Carterville, IL 62918
Name of Licensee Contact Person:	Joseph Hirschi
Contact Person’s E-mail Address:	jhirschi@icci.org
Contact Person’s Office Telephone and Fax:	618-985-3500, fax 618-985-6166

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JOY TECHNOLOGIES INC., D/B/A JOY MINING MACHINERY

By: Cynthia A. Hayes
Name Printed: Cynthia A. Hayes
Title: Photographer
Date: 3/26/12

VITA

Graduate School

Southern Illinois University

Joseph Christian Hirschi

jhirschi@icci.org

University of Utah

Bachelor of Science, Mining Engineering, June 1983

University of Utah

Master of Science, Mining Engineering, June 1985

Thesis Title: The Principles of Mine Management

Major Professor: Howard M. Wells

University of Utah

Master of Business Administration, June 1985

Southern Illinois University

Doctor of Philosophy, Engineering Science, August 2012

Dissertation Title: A Dynamic Programming Approach to Identifying Optimal Mining Sequences for Continuous Miner Coal Production Systems

Major Professor: Yoginder Paul Chugh

Special Honors and Awards:

Outstanding Graduate Student, 2001-2002, Department of Mining and Mineral Resources Engineering, Southern Illinois University at Carbondale.

Related Publications:

Hirschi, J.C., Y.P. Chugh, A. Saha, and M. Mohanty, 2002. "Evaluating the Use of Surfactants to Enhance Dust Control Efficiency of Wet Scrubbers for Illinois Coal Seams." Mine Ventilation, Proceedings of the 9th North America/US Mine Ventilation Symposium, E. DeSouza, ed., A.A. Balkema Publishers, Lisse, The Netherlands, pp. 601-606.

Hirschi, J.C., M. McGolden, T. Cushman, and Y.P. Chugh, 2004. "Optimizing Unit Shift Productivity of Continuous Miner Coal Production Systems." In Proceedings of the 29th International Technical Conference on Coal Utilization and Fuel Systems, Clearwater, FL, Apr. 18-22, Paper #138, pp. 1323-1334.

Hirschi, J.C., Y.P. Chugh, and A. Moharana, 2006. "Development of Alternate Mining Geometry Concepts for Increased Extraction, Ground Stability and Productivity at a Shallow Depth Illinois Coal Mine." In Proceedings of the 2006 International Symposium on Mine Planning and Equipment Selection, M. Cardu, R. Ciccu, and E. Michelotti, eds., pp. 1118-1123.

Hirschi, J.C., 2007. "Review of Computer Modeling in Underground Coal Mining and Its Application in Optimizing Continuous Miner Production Systems." Transactions of the Society for Mining, Metallurgy and Exploration, Inc., Littleton, CO, Vol. 320, Sec. 3, pp. 142-148.

Related Presentations:

Hirschi, J.C., Y.P. Chugh, A. Saha, and M. Mohanty, 2002. "Evaluating the Use of Surfactants to Enhance Dust Control Efficiency of Wet Scrubbers for Illinois Coal Seams." 9th North America/US Mine Ventilation Symposium, Kingston, Ontario, Canada, Jun. 8-12.

Hirschi, J.C., M. McGolden, T. Cushman, and Y.P. Chugh, 2004. "Optimizing Unit Shift Productivity of Continuous Miner Coal Production Systems." 29th International Technical Conference on Coal Utilization and Fuel Systems, Clearwater, FL, Apr. 18-22.

Hirschi, J.C., 2005. "A Review of Computer Modeling in Underground Coal Mining and Its Application in Optimizing Continuous Miner Production Systems." Annual Meeting of the Society for Mining, Metallurgy, and Exploration, Salt Lake City, UT, Feb. 28-Mar. 2.

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Hirschi, J.C., 2011. "Optimizing Continuous Miner Cut Sequences for Improved Productivity and Worker Health and Safety." Annual Meeting of the Illinois Mining Institute, Marion, IL, August 31.

Hirschi, J.C. and Y.P. Chugh, 2012. "Optimizing Continuous Miner Cut Sequences for Improved Productivity and Worker Health and Safety." Annual Meeting of the Society for Mining, Metallurgy, and Exploration, Seattle, WA, Feb. 19-22.