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Production and Cost Assessment of a Potential Application of Surface Miners in Coal Mining in West Virginia

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**Production and Cost Assessment of a Potential Application of
Surface Miners in Coal Mining in West Virginia**

Timothy A. Nolan

**Thesis submitted to the
Benjamin M. Statler College of Engineering and Mineral Resources
At West Virginia University
in partial fulfillment of the requirements
for the degree of**

**Master of Science
in
Mining Engineering**

**Vladislav Kecojevic, Ph.D., Chair
Brijes Mishra, Ph.D.
Keith A. Heasley, Ph.D.**

Department of Mining Engineering

**Morgantown, West Virginia
2012**

**Keywords: Surface Mining; Surface Miner; Bituminous Coal;
West Virginia; Analytical Hierarchy Process;
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ABSTRACT

Production and Cost Assessment of a Potential Application of Surface Miners in Coal Mining in West Virginia

Timothy A. Nolan

The broad objective of this research was to improve current mining practices and reduce negative environmental impact of surface coal mining in West Virginia (WV). The specific objectives were to (i) build the production and cost models to determine if conventional surface mining unit operations such as drilling, blasting, digging, loading, and crushing can be replaced with one mining machine called a surface miner (SM), (ii) apply the analytical hierarchy process (AHP) to help select the optimal mining method, and (iii) determine the extent of which the SM would benefit surface coal mining in WV, particularly by reducing some of the negative environmental impacts associated with current extraction practices.

The entire production and cost models were developed in MS Excel. The design and the procedures used for achieving the objectives involved six interrelated modules. Module #1 includes rock properties of sandstone, shale, which are the predominant overburden and interburden materials in WV, and sub-bituminous coal. Module #2 considers drilling and blasting operations. Module #3 includes digging and loading of overburden material. It specifically addresses mining equipment such as electric and hydraulic shovels and front-end wheel loaders. Interburden material also includes sandstone and shale, and the two unit operations involved are ripping and pushing by bulldozers and loading by hydraulic backhoe shovel. Module #4 covers coal extraction by hydraulic backhoe shovel and the crushing operation. Module #5 focuses on the SM, while Module #6 is used to conduct comparative analyses among all mining unit operations for overburden, interburden, and coal. The resulting production rates, ownership, and operating costs were also presented. The analytical hierarchy process (AHP) was used to help select the optimal mining method based on both qualitative and quantitative factors.

Based on the results of this research, application of SM would benefit coal extraction in surface mining operations in WV. The estimated cost of coal extraction is lower than with conventional mining methods, and the machine is able to work selectively with high separation accuracy. Thin seams of coal and overburden can be mined separately, thus preventing dilution with interburden material. Rising operating costs for the SM become an issue as the quantity of material being extracted increases and higher values of unconfined compressive strength are encountered. Cutting tools on the drum wear out faster and require more changes and, at a certain point, the SM becomes unfeasible because of the time and cost involved in changing cutting tools. A significant advantage of SM, where applicable, is the elimination of the negative environmental impacts associated with blasting.

**PRODUCTION AND COST ASSESSMENT OF A POTENTIAL
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List of Symbols

AHP	Analytical Hierarchy Process
ANFO	Ammonium Nitrate + Fuel Oil
CC	Capital Cost
CI	Consistency Index
CR	Consistency Ratio
CTR	Cuttability Index
D	Cutting Depth
DR	Depreciation Rate
DV	Depreciation Value per Year
FC	Fuel Cost
FLC	Full Load Fuel Consumption
GC	Grease Cost
HCI	Hydraulic Oil Change Time Interval
HO	Hydraulic Oil Cost
INS	Insurance Cost
INSR	Insurance Rate
INT	Interest Cost
INTR	Interest Rate
LBC	Labor Cost
LC	Lube Cost
MP	Maximum Production
NCT	Non-Cutting Time
NPC	Number of Pick Changes per Row
OC	Ownership Cost
OHY	Operating Hours per Year
OP	Operating Years

OPC	Operating Cost
P	Power
PC	Purchasing Cost
PCT	Pick Changing Time
PCY	Pick Consumption per Year
PPS	Picks per Set
PQ	Practical Cutting Performance
PTC	Pick Cost
PW	Pick Wear
Q	Cutting Performance
RC	Repair Cost
RCT	Real Cutting Time
RCT%	Real Cutting Time Percent
RI	Randomly Generated Reciprocal Matrices
RL	Row Length
ROHY	Real Operating Hours per Year
RP	Required Production
RV	Residual Value
RV%	Residual Value Percent
SM	Surface Miner
SP	Spare Parts
SS	Sumping Speed
ST	Sump Time
TS	Traveling Speed
TSL	Total Sump Time
TT	Turning Time
TX	Tax Cost
TXR	Tax Rate
UP	Usable Production

V	Cutting Velocity
VHO	Volume of Hydraulic Oil Tank
VR	Value to be Recovered Through Work
W	Width
WC	Water Cost
WP	Wear Parts
WV	West Virginia

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Chapter 1

Introduction

1.1 Background

Coal mining is the backbone of West Virginia's economy. The state produced roughly 143 million tons of coal in 2011 (WV Coal, 2011). Approximately 50 million tons or 35% of total coal production came from surface mines.

The current surface mining practices in West Virginia (WV) consist of conventional mining unit operations including: drilling, blasting, digging, loading, haulage, and disposal (Figure 1.1). Draglines are used in some mines for stripping and direct disposal of overburden material into excavated areas. Coal crushing is the final conventional mining unit operation.

WV used almost 465,000 tons or 930 million lb of explosives in the mining, quarrying, and construction industries in 2008 (Apodaca, 2010). The state ranks second in the nation for explosives consumption. Almost 85% or 790.5 million lbs were used in surface coal mining. Considering an average price of blasting agents (bulk emulsions, slurry and ANFO) of \$0.52 per lb, the surface coal mines in WV spent \$411 million for explosives alone. This does not include additional blasting accessories such as detonators, boosters, detonating cords, lead lines, additional costs for labor, explosives truck-delivery, and shot services. The addition of the drilling, digging, loading, and

hauling costs for the blasted material amounts to a significant overall cost to mine operators.

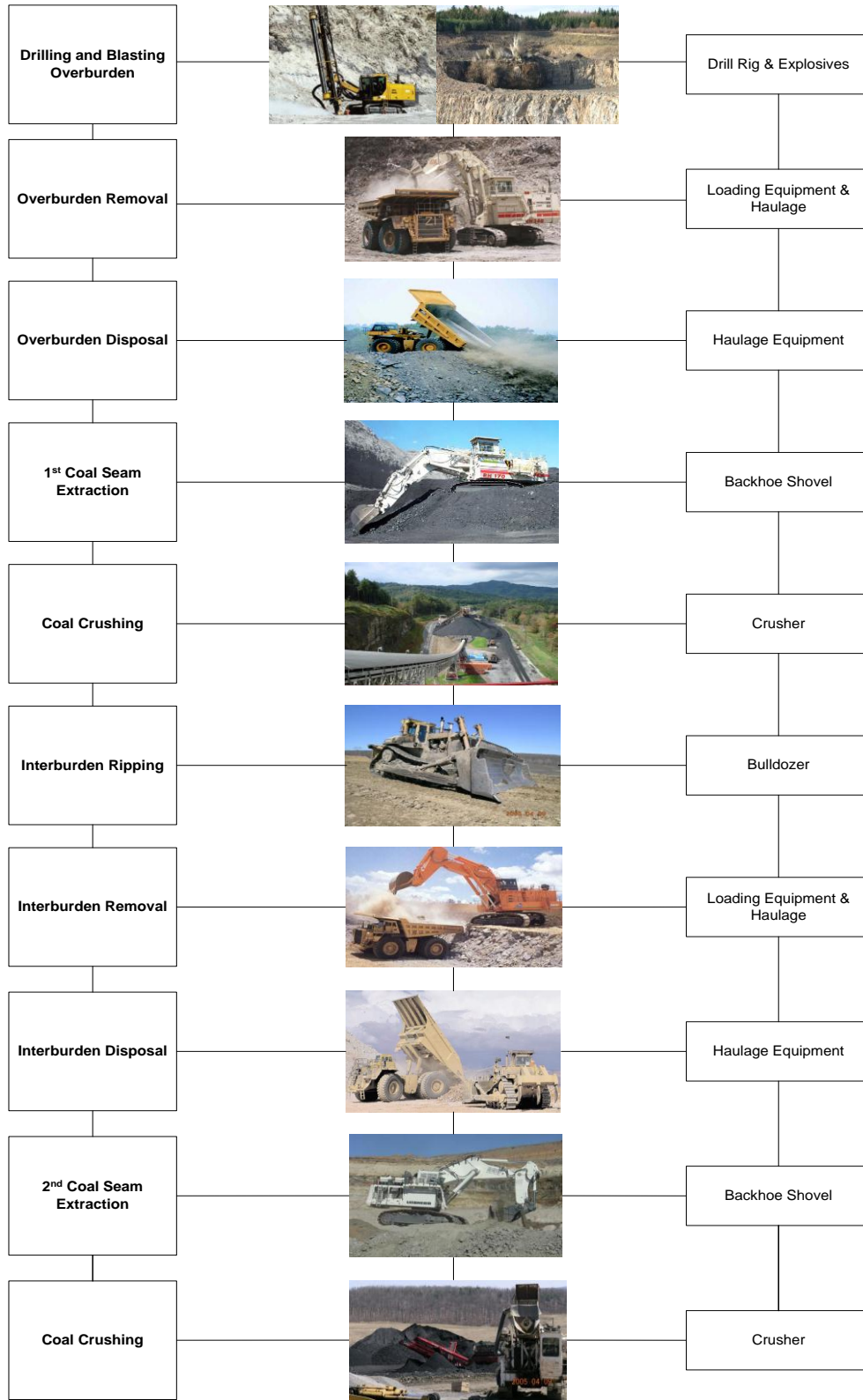


Figure 1-1 Conventional Surface Mining Process
 (Image Sources: Atlas Copco, Terex, Caterpillar, Kecojevic, Hitachi, and Liebherr)

Current blasting methods, if not properly designed and executed, can create hazards like flyrock, toxic fumes, ground vibration, and surface vibration created by air-blast. Blasting methods have been refined and are performed in a safer manner but the hazards still remain.

It is required that coal seams, overburden, and interburden be mined separately, thus preventing dilution of the coal with non-coal material. This is particularly important for the surface coal mines in WV where most of the seams are interspersed with interburden layers (Figure 1.2). The capability of the extraction systems to work selectively provides a better quality of the coal, less dilution, and higher utilization of the coal deposits.



Figure 1-2 Surface Coal Mine in WV (Kecojevic, 2006)

The Surface Miner (SM) is a multi-purpose production machine that integrates cutting, crushing, and loading (Figures 1.3 and 1.4). It is applied in numerous coal and quarry mining operations around the world. The SM is capable of continuously mining materials with compressive strengths up to 180 MPa or 26,100 psi (Wirtgen, 2010). The largest SMs have achieved a production rate of 5,300 loose yd³ per hour and can excavate layers of up to three feet of material in one step.



Figure 1-3 Surface Miner (Wirtgen, 2010)

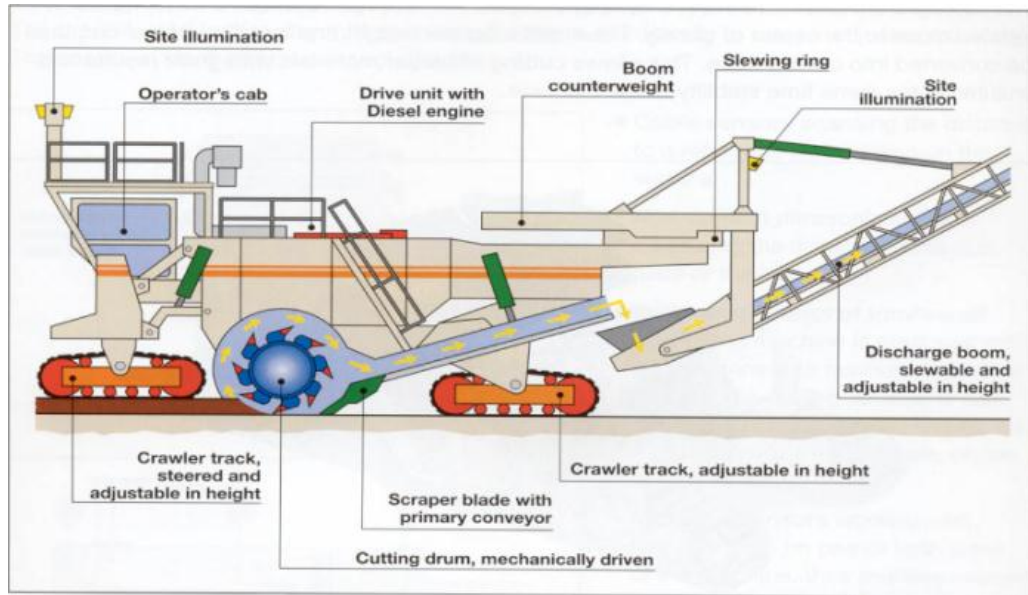


Figure 1-4 Surface Miner Diagram (Wirtgen, 2010)

The SM is mounted on four crawler tracks with an almost infinite adjustability. As the machine advances, a rotating drum studded with carbide-tipped cutting tools (Figure 1.5) cuts the material to sizes suitable for haulage. The cutting drum rotates upwards, i.e. the helically arranged cutting tools on the drum transport the material towards the center of the drum from where it is transferred by the primary belt to the secondary discharge belt.



Figure 1-5 Rotating Drum and Carbide-Tipped Cutting Tools (Wirtgen, 2010)

The SM can be used for the following operations: i) cutting, crushing, and loading of overburden, interburden, and coal into haulage units such as trucks or belt conveyors; ii) cutting, crushing, and side casting; and iii) cutting, crushing, and windrowing. Therefore, one single machine can replace several conventional surface mining unit operations including drilling, blasting, digging, loading, and crushing. Since only one machine is used for several unit operations, it has a significant potential to reduce capital and operating costs, reduce manpower, and simplify coordination and planning of the mining process.

SMs are either equipped with wire rope sensors connected with side plates or sonic sensors which measures the distance from the ground and adjust the cutting depth accordingly. When a GPS receiver is connected to the automatic cutting depth controller, predefined profiles can be cut after programming the receiver. The ability to selectively

mine results in better quality of the mined coal, less dilution, and higher recovery of the coal deposits. Figure 1.6 shows the selective work of the SMs in a coal mine in the U.S.



Figure 1-6 Selective Extraction of Coal by SM (Kecojevic, 2006)

Depending on the rock properties and the desired material size, the pick configuration on the cutting drum can be modified accordingly. The pick pattern (helix) on the drum remains fundamentally the same, but variations in pick type and spacing between picks will change to achieve the desired material size. The amount of picks on the cutting drum and the spacing between picks must be defined. This value is determined based on the material properties and the desired size of processed material.

1.2 Problem Statement

Surface coal mining in WV faces many challenges. These challenges include more restrictive regulations, a negative public perception, and difficult geological formations. The industry as a whole has a negative image in the public eye due to, among other factors, blasting hazards involving flyrock, toxic fumes, ground vibration, and surface vibration created by air-blast. Although these hazards do not occur with every blast it only takes a few incidents to tarnish the reputation of the entire industry. Exploring and applying alternative safe and environmentally friendly technology provides the potential to minimize if not eliminate these hazards.

The geologic formations in WV consist of overburden and multiple layers of sandstone and shale rock types with bituminous coal seams of varying thicknesses interspersed between layers of interburden. Overburden and interburden layers must be removed before the coal is extracted. Depending on the thickness of the interburden layers, various methods are used for its removal. Layers with a thickness of generally less than five feet are ripped by a bulldozer. If the rock is ripped, additional loading is required to load and haul the material. If the interburden is too thick or too hard to rip, drilling and blasting is required to loosen the rock. The process of ripping or blasting continues with each interburden layer that is encountered in the mine. Once the coal seam is exposed, a shovel or front-end wheel loader digs and loads the coal into haulage equipment, typically dump trucks. The use of this type of loading equipment generates issues with the coal quality. Loading coal with a shovel or front-end wheel loader is conducted based on the operators' "best judgment" and "experience" attributes. This approach is the

common practices which helps distinguish between coal, overburden, and interburden material. While this approach can be effective, there is a potential for a large margin of error. Excavating coal in this manner can create quality issues, which will require additional work to clean. Technology, such as the SM, can be employed to lessen the effects created by this conventional approach to coal extraction.

More stringent governmental regulations and public pressure are forcing the mining industry to evolve and minimize its environmental impacts. Future coal mining operations will need to incorporate new design features and practices that can substantially reduce these impacts to achieve “low impact” mining. These design features and practices will be necessary to ensure that the coal industry can design, permit, build, operate, reclaim, and monitor future mines in full compliance with the increasingly stringent environmental performance standards. New mining technologies and systems can minimize environmental disturbances during overburden removal and coal extraction, while improving coal recovery.

1.3 Scope of Work

The broad objective of this research was to improve current mining practices and reduce negative environmental impacts of surface coal mining in WV. The specific objectives are:

- (i) Build the production and cost models to determine if conventional surface mining unit operations including drilling, blasting, digging, loading, and crushing can be economically replaced with a surface miner (SM).
- (ii) Apply the analytical hierarchy process (AHP) to help select the optimal mining method based on both qualitative and quantitative factors.
- (iii) Determine if the SM would benefit surface coal mining in WV, particularly by reducing the negative environmental impacts associated with the conventional mining method.

The following chapters provide a literature review of relevant studies, the methodology developed for this research, results, discussion, conclusions, and recommendations for future work.

Chapter 2

Literature Review

One of the most comprehensive studies on the application of SMs was conducted by Dey and Ghose (2008). The authors established a Cuttability Index (CTI) to explore the applicability of the SM for various values of rock properties. The rock and machine properties including point load strength, volumetric joint count, abrasivity, direction of cutting with respect to joint directions, and machine power are given numerical values, as shown in the Table 2.1.

**Table 2-1 Rating of the Parameters of New Rockmass Cuttability Classification
(Dey and Ghose, 2008)**

Class	I	II	III	IV	V
Point Load Index (I_{S50})	< 0.5	0.5 - 1.5	1.5 - 2.0	2.0 - 3.5	> 3.5
Rating (I_S)	5	10	15	20	25
Volumetric joint count (no/m ³)	> 30	30 - 10	10 - 3	3 - 1	1
Rating (J_V)	5	10	15	20	25
Abrasivity	< 0.5	0.5 - 1.0	1.0 - 2.0	2.0 - 3.0	> 3.0
Rating (A_W)	3	6	9	12	15
Direction of cutting respect to major joint direction	72 ⁰ - 90 ⁰	54 ⁰ - 72 ⁰	36 ⁰ - 54 ⁰	18 ⁰ - 36 ⁰	0 ⁰ - 18 ⁰
Rating (J_S)	3	6	9	12	15
Machine Power (kW)	> 1000	800 - 1000	600 - 800	400 - 600	< 400
Rating (M)	4	8	12	16	20

A summation of the values is performed to generate a CTI between 20 and 100. Based on this value, the applicability of the SM is ranked on the Excavatability Index from

“Very Easy Excavation,” if lower than 50, to “Surface Miner Should Not be Deployed” if the CTI is greater than 80 (Table 2.2).

Table 2-2 Applicability of Surface Miner Based on Excavatability Index (Dey and Ghose, 2008)

Excavatability Index	Possibility of Ripping
$CTI < 50$	Very Easy Excavation
$50 < CTI < 60$	Easy Excavation
$60 < CTI < 70$	Limit of Economic Excavation
$70 < CTI < 80$	Difficult Excavation, May be Not Economic
$CTI > 80$	Surface Miner Should Not be Deployed

Production rate is estimated by applying this rated machine capacity and a factor that considers influence from pick shape and pick spacing to the cuttability index.

The methodology developed by Dey and Ghose (2008) is used for an initial examination of the applicability of a SM. The production rate is calculated by considering only a few of many factors. This method does not account for operator influenced factors and will be marginally off when compared to the actual production rate. Additional factors such as time delays would add to the validity of the production rate calculated by this method.

Origliasso (2011) performed production and cost calculations for the SM based primarily on the power of the machine. Major aspects of the calculations, such as production rate, fuel consumption, and cutting speed, are determined based on the machine’s power. The equation used to determine cutting speed is verified by the case studies performed at quarry operations. The speed is determined for the hard rock applications and may require some adjustments for a coal type operation. Origliasso’s (2011) approach, with

the integration of specific energy of the SM, is fairly sophisticated. The SM calculations estimate an actual production rate as it considers delays that are encountered. The cutting time has been built into equations as well as the machine power, rock abrasivity, and the rock's unconfined compressive strength. This method produces reasonably accurate values unless any anomalies are encountered for which a more detailed analysis of cutting time would be useful.

One of the problems that have remained when considering a SM is determining what type and size is optimal for the specific application. Pradhan and Dey (2009) explored this problem and developed software that can be applied to this situation to provide a suitable SM and the optimal mode for its operation. This technique relies on the methodology developed by Dey and Ghose (2008), in which the cuttability index and production rate are found. The software provides a quick way for mine planners and engineers to evaluate the use of a SM. The Pradhan and Dey (2009) approach to the evaluation of the applicability of SMs is again based on the cuttability index created by Dey and Ghose (2008) and will provide the same results but in a nearly instantaneous timeframe. This software provides the specifics of the SM and requires a cost model to be created by a separate process.

Schimm (1997) studied the application of a Wirtgen SM in a limestone and pegmatite mining operation. This case study occurred over a four week period during which the performance and requirements of the SM were logged. The objectives of this study were to (i) determine limestone mining costs without blasting, (ii) reduce transport cost to the

nearby cement factory, and (iii) determine if selective mining of pegmatite was feasible. The properties of the limestone allowed for an average production rate of 210 tons per hr. The costs for operating and owning the machine during the time of the study were also observed. The transportation costs were determined for three scenarios, one with an in-pit mobile crusher, one with a semi-mobile crusher, and one with the Wirtgen SM. The results of this portion of the study proved the SM to be the most economical of the three scenarios. During the mining of pegmatite, the SM was able to extract the material in a highly selective manner where the rock was not rippable. For this reason, as well as the crushed material the SM produced, the SM was again the most economical method. The study concluded that when mining in medium to hard material, the SM provides an economical alternative to conventional mining methods.

The study performed by Schimm (1997) provides a detailed breakdown of the costs associated with operating and owning a Wirtgen SM. While the study relates the use of a SM to the unconfined compressive strength of the material, many other rock parameters are not considered. The inclusion of the abrasivity of the limestone and pegmatite would have been a valuable piece of information for the considering a SM. As this is a case study, values for the SM were not calculated but observed, and the abrasivity may not have been known. However, this study provides useful information that can be expected regarding similar conditions.

The application of a SM in the Nongtraï Limestone Mine in Meghalaya, India was studied by Ghose et al. (2010). This study for a possible alternative mining method was

performed in response to a Supreme Court order to stop all mining activities. The objective of this study was to determine if SMs could be used as an alternative mining method. The authors proceeded by applying eleven empirical approaches that have been developed by various mining professionals. The evaluation of each empirical approach would determine if the SM would be applicable, if the material could be cut by that type of machine, or if there was a SM that would be able to cut the rock. The results of the study indicated that there was no SM available that would be applicable to the studied limestone mine.

The study performed for the Nongtra Limestone Mine in Meghalaya, India described a situation any mine could face in light of evolving litigation. The approaches used in this study varied in complexity but provided reasonable results while focusing on applicability.

SMs are built in many sizes and have various applications based on the capability of the machine. The objectives when considering any new SM technology is to lower cost of operation, reduce coal dilution, and improve coal recovery. In his research, Schroder (2011) considered the cost comparison of the larger KSM type machine from Krupp Fordertechnik. Three different applications were explored and compared to the conventional mining method. The first was a comparison of the KSM, the hydraulic shovel, and the front-end wheel loader. In a study conducted for a coal mine in India, the author indicated that the SM was the most economical option. The second study pertained to the transportation of material. It compared a SM and truck mining unit to a

SM and belt conveyor mining unit in a Russian mine. The SM and conveyor belt combination provided the lower cost. In a third study, the comparison of the SM, hydraulic shovel, and front-end wheel loader was conducted to prepare a highwall for an auger mining operation. The SM was found to be the most economical method.

The studies by Schroder (2011) show various uses of the SM and evaluate where it is the most economical method. While the results are clearly presented, the methodology behind the calculations is not evident. The inclusion of rock parameters and mine site specifications would assist in understanding the calculations. While the data and calculations may not be stated, some of the SM's applications are. The SM does not only serve the purpose of the primary mining process but can perform supporting processes as well.

Designing and calculating costs for a mining project does not encompass all of the issues, the post-mining reclamation plans must also be designed. While economic evaluations can be performed for each scenario, the final reclamation design ultimately relies on the decision team. To aid in this group decision process Bascetin (2006) uses the Analytical Hierarchy Process (AHP) developed by Saaty (1980). In this study, the AHP was used to determine the optimal reclamation method for an open-pit coal mine in the Seyitomer region in Turkey. The AHP model developed by Bascetin (2006) considers capital cost, operating cost, natural factors, and cultural factors. Both natural and cultural factors have multiple sub-criteria that are included in the AHP. The end result of this study will be one of five different alternatives for a reclamation method.

The study performed by Bascetin (2006) was to determine if the AHP would aid in the decision process for a particular reclamation method. The AHP model developed enabled decision makers to look at each of the possible reclamation methods and decide what their weaknesses and strengths were. The model was determined to be an improvement on the team's decision making process, as well as reducing the time and effort devoted the process. This model can be used for a basis when considering multiple reclamation methods with some variations to suit a specific project.

Chapter 3

Methodology

3.1 Introduction

The objective of this research is to determine whether the Surface Miner mining method is a suitable option for surface coal mines in WV. The design and procedures used for achieving the objectives involve six interrelated modules. Module #1 includes rock properties of sandstone, shale, and bituminous coal, which are the predominant overburden, interburden, and coal types in WV. Module #2 considers drilling and blasting operations. Module #3 includes digging and loading of overburden material and specifically addresses mining equipment such as the electric and hydraulic shovels and front-end wheel loaders. Two unit operations are involved with the removal of the interburden material: ripping and pushing by bulldozers and loading by hydraulic backhoe shovel. Module #4 covers coal extraction by a hydraulic (backhoe) shovel and the coal crushing operation. Module #5 focuses on the SM and its production, ownership, and operating costs for overburden, interburden, and coal. Module #6 consists of a comparative analysis of all unit mining operations for overburden, interburden, and coal. It also presents the results for the production rates, ownership and operating costs. The production and cost models were developed in MS Excel. The analytical hierarchy process (AHP) is used for the selection of the optimal mining method based on both qualitative and quantitative factors.

An algorithm was developed in order to determine the appropriate mining method for each rock unit in an organized fashion. This algorithm or one similar to it should be followed to insure all results are accurately calculated for the cost module comparison. This algorithm can be seen in Figure 3.1.

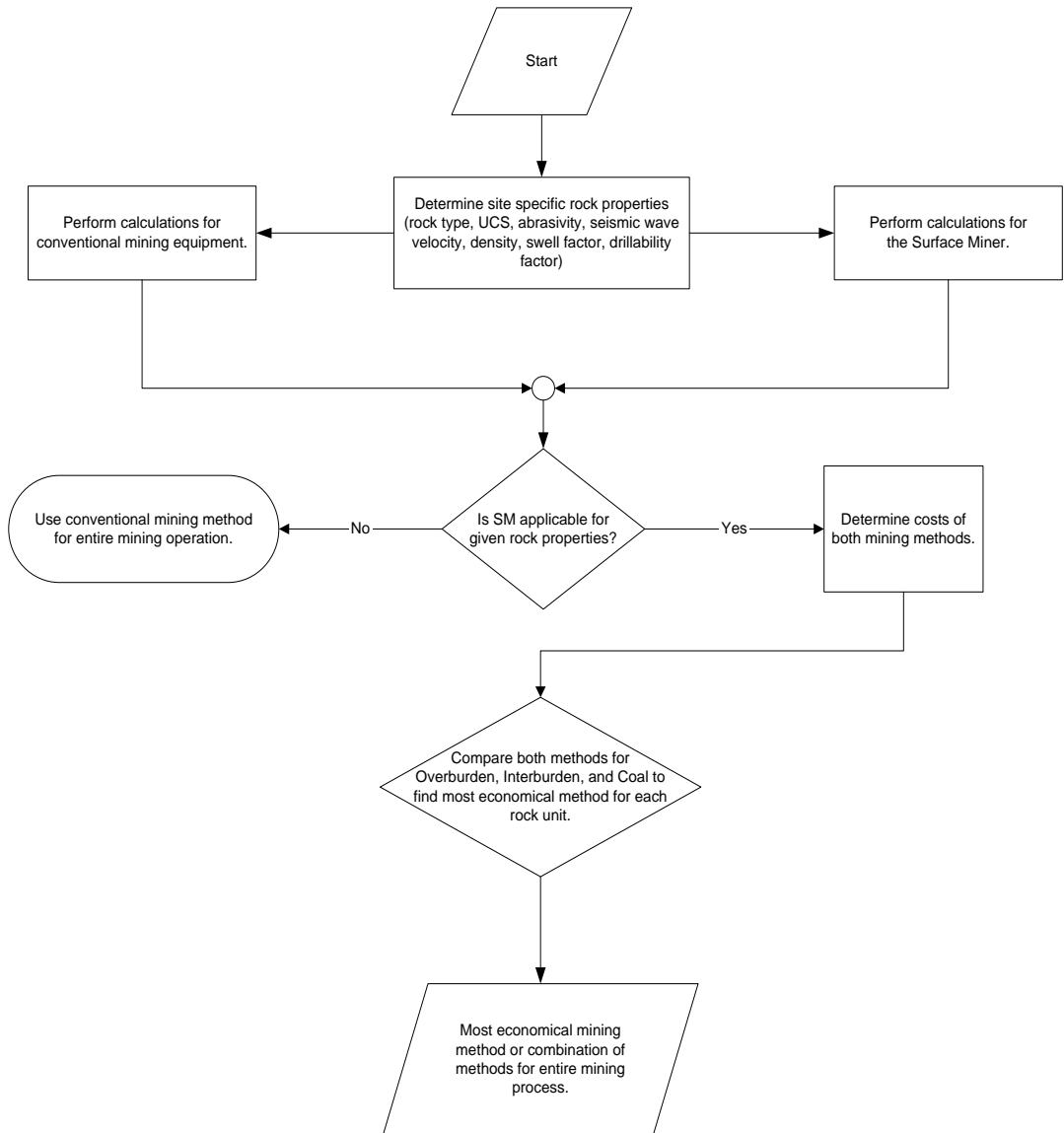


Figure 3-1 Mining Method Selection Algorithm

The cost and production models for both the conventional and SM mining methods consider a medium-size surface coal mine in WV. For this study, the medium-size mine is defined with a production rate of 10 million bank cubic yards of overburden, 1 million bank cubic yards of interburden, and 1.5 million tons of coal to be excavated annually.

3.2 Rock Properties (Module 1)

Module #1 contains the descriptions of possible rock types and properties within the state of WV. The typical rock units found in the surface coal mines of WV (generally the majority of the state excluding the eastern panhandle) are sandstone, shale, and bituminous coal (WVGES, 2011). The bituminous coal in WV and the surrounding area is found in varying thickness and in multiple seams. There may be instances where multiple seams of varied thickness exist in the same formation and are being mined or abandoned.

Properties such as bank & loose density, unconfined compressive strength, abrasivity, seismic wave velocity, quartz content, and swell factor were compiled from Caterpillar (2010), Hartman (1992), Rusnak (2000), Schubert (2007), Mavko (2011), Plinninger (2010), Ingresoll-Rand (2003), Kecojevic (2010). Average values were generated for each property and applied to the conventional mining and SM equations. Table 3.1 shows the average values for the rock and coal properties in WV that are used in this study.

Table 3-1 Average Values of Coal and Rock Properties in WV

Rock type	Bituminous coal	Sandstone	Shale
Bank density (lb/yd ³)	2,150	4,250	2,800
Loose density (lb/yd ³)	1,600	2,550	2,100
Specific gravity	1.28	2.5	1.66
Swell factor	1.35	1.6	1.45
Unconfined Compressive Strength (psi)	2,901	14,500	10,875
Cerchar Abrasive Index (CAI)	1	2	1.5
Seismic Wave Velocity (ft/sec)	-	9,020	5,900
Equivalent Quartz Content (%)	5	75	45
Drillability factor		1.8	2.0

3.3 Drilling and Blasting Operations (Module 2)

Equations developed by Atlas Copco (2006), Ingersoll-Rand (2003), Sandvik Tamrock (1999), Austin Powder (2009), Dyno Nobel (2010) and Kecojevic (2010) were used to calculate drilling and blasting parameters. These parameters include hole diameter, penetration rate, overall drilling rate, bench height, burden, spacing, sub-drilling, hole inclination, hole depth, hole length, stemming, particle size for stemming, hole charge length, hole charge concentration, total charge per hole, type of total amount of explosives, bottom and column charge concentration, volume of rock per foot of hole, volume of rock per hole, weight of rock per hole, number of required holes, specific drilling, total required drilling, powder factor, and delay times. Ammonium Nitrate + Fuel Oil (ANFO) and emulsions are the most common explosives agents used in WV surface coal mines and are used in this study.

The total drilling cost is calculated and it is expressed in $\$/\text{yd}^3$ and $\$/\text{year}$. Blasting cost related to explosives, detonators, boosters, leadline, tubes, labor, and shot service is also determined. A total blasting cost is expressed in $\$/\text{yd}^3$ and $\$/\text{year}$.

3.4 Digging and Loading (Module 3)

This module pertains to the equipment used for digging and loading of the overburden and interburden. For this study, the production rate is defined to be 10 million bank cubic yards of overburden and 1 million bank cubic yards of interburden.

3.4.1 Overburden

In most WV surface coal mines several machines are used to dig and load overburden material. This equipment includes hydraulic shovels, electric (rope) shovels, front-end wheel loaders, and draglines (which are used mainly for stripping and disposal into the excavated area). Haulage costs are not considered in this research and therefore draglines are not included in the mining method comparison.

Production rates for these machines except for the dragline are calculated and ownership and operating costs are determined. Various sources such as Caterpillar (2010), Komatsu (2011), P&H (2005), Hartman (1992), and Kecojevic (2010) are used to determine the production rates. Equations for ownership and operating costs are well documented in the sources stated previously and are used to calculate the costs in this research. Compiled cost data are obtained from InfoMine (2010). The total cost for each piece of equipment is expressed in $\$/\text{yd}^3$, $\$/\text{hr}$ and $\$/\text{year}$.

3.4.2 Interburden

The interburden material in WV is typically, but not limited to, a sandstone or shale type rock. This material is interspersed between coal seams. The thickness and material properties of the interburden determines whether the rock is ripped, excavated, and loaded or the rock is to be blasted. It is assumed that interburden less than five feet thick can be ripped by a bulldozer.

The ability to rip interburden is dependent on the properties of the rock; specifically the seismic wave velocity. A bulldozer's specific capabilities are based on the machine's power, shank characteristics, and size. To insure the widest array of applications in varying thicknesses of interburden, the Caterpillar D11T CD or similar large size bulldozer was chosen for this study. Caterpillar (2009) provides the production rates for the bulldozers.

The interburden is first ripped by a bulldozer and then it is loaded by a hydraulic backhoe shovel. Production rates for these units are calculated and ownership and operating costs are determined. Compiled cost data are obtained from InfoMine (2010). The total cost for each unit is expressed in $\$/\text{yd}^3$, $\$/\text{hr}$ and $\$/\text{year}$.

3.5 Coal Extraction & Crushing (Module 4)

Module #4 covers coal extraction by hydraulic (backhoe) shovel and the crushing operation. Once the coal is crushed it is transported to the processing facility in its fragmented form. In most cases, the coal needs to be reduced in size for its final use. The size and type of crusher is determined by the production rate, the feed and product sizes, the material characteristics such as lump factor, the number of drives, the drive efficiency, and the Bond's Work Index. The production rate and the feed and product sizes are values that will be generated by the specifications of the mine. The material lump factor is determined by the material composition, which can range from course to fine material. The drive efficiency depends on what type of motor is used in the crusher.

The ownership and operating costs are calculated based on production rate, crusher power requirement, and crusher settings. The total cost is expressed in \$/ton, \$/hr, and \$/year.

3.6 Surface Miner Method (Module 5)

This module describes the calculations of the SM in three different extraction scenarios; overburden, interburden, and coal extraction.

The first item to consider in the SM mining method is the mine design. The SM requires adequate room on both sides of the cut to turn the machine. An example of a mine design, where ramps are required on both ends of the row to allow the SM to turn around

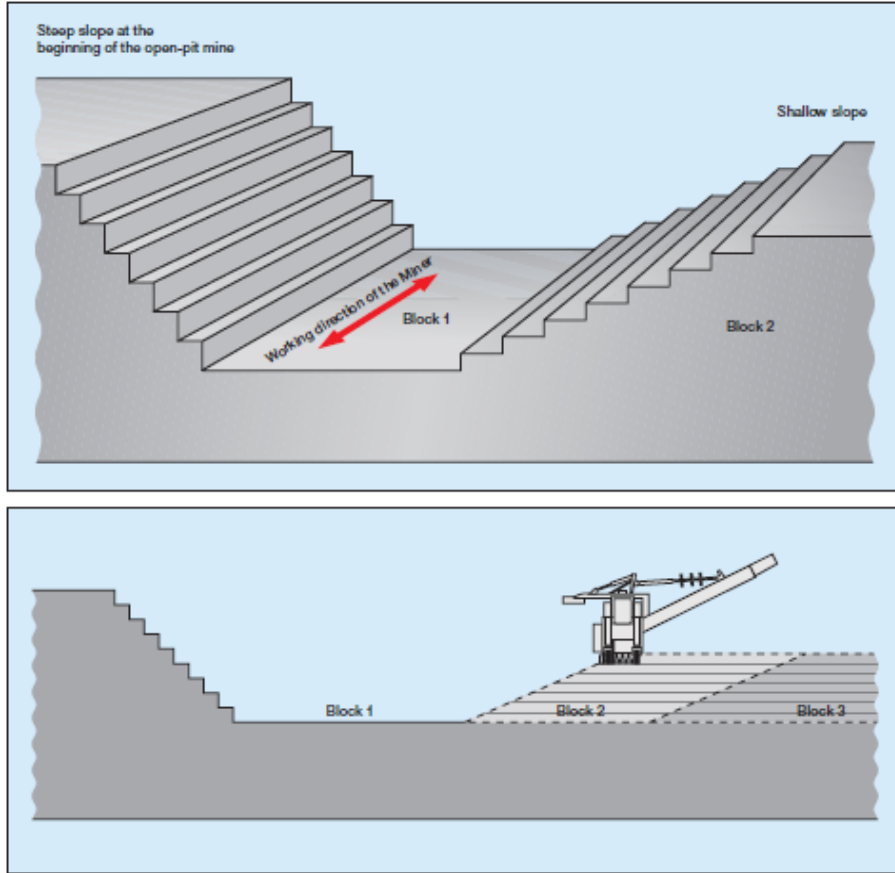


Figure 3-3 Typical Mining Sequence of a Surface Miner (Wirtgen, 2008)

A critical value that must be defined in the beginning of the SM calculations is the cutting performance, (Q) which is expressed in bank ft^3/hr . The Equation 3.1, developed by Wirtgen (2010), is used to determine the cutting performance in this thesis. The dimensions of the cut are dependent on the width, (W) in ft, of the cutting drum and an average depth, (D) in ft, of the cut. The cutting velocity, (V) in ft/min , is multiplied by the cutting dimensions to find the cutting performance as follows:

$$Q = W * D * V * 60 \quad (3.1)$$

The depth and cutting velocity are dependent on the size of SM and the material being excavated. Material with higher compressive strength and abrasivity may require a shallow cut to prevent excessive pick wear. Equation 3.1 does not account for any delays that will be encountered in the cutting process.

Essentially, according to equation 3.1, the cutting performance is based on the size of the SM and the cutting velocity. The machine also has a travel (non-cutting) speed that will be used in other equations. The traveling speed is usually designated by the manufacturer, but the velocity at which the machine cuts is typically an empirically determined value. Because an experienced value is unavailable for every desired rock property scenario, another source for this information is needed. It is necessary to use an additional equation to determine the velocity of the SM while cutting. An equation developed by Origliasso (2011) calculates the cutting velocity, (V) in m/min, and uses the machine power, (P) in kW, compressive strength, (UCS) in MPa, and cutting depth, (D) in cm, as follows.

$$V = \left(\frac{P * (59.6 - 12\ln[UCS])}{D} \right) \quad (3.2)$$

While this equation for cutting velocity is reliable, it is necessary to add an adjustment factor to accommodate for the different types of rock. The adjustment factor is generated based on the cutting velocities the SM will experience in the field for both coal and hard rock. Determination of the cutting velocity in both coal and hard rock is given by Equations 3.3 and 3.4, respectively, where the power, (P) is in kW, and cutting depth, (D)

is in cm. The adjustment factor allows the cutting velocity equation to calculate values more accurate to what will be experienced in the field. The conversion factor from meters per minute to feet per minute is 3.28.

The adjustment factor, which is determined based on cutting velocity field values, is 1.35 for coal and 2.9 for hard rock. These cutting velocity equations are written as follows:

$$V = \left(\left(\frac{P * (59.6 - 12 \ln[UCS])}{D} \right) * 3.28 \right) * (1.35) \quad (3.3)$$

$$V = \left(\left(\frac{P * (59.6 - 12 \ln[UCS])}{D} \right) * 3.28 \right) * (2.9) \quad (3.4)$$

One of the most influential variables when considering a SM is the amount of cutting tool (pick) wear that occurs during the cutting process. The reason for this high influence is not only the cost of replacement picks but also the time spent on replacing the worn picks. These additional factors affect the direct cost of operating the machine and its production capability. This is the why the SM becomes less economically feasible in the harder rock type situations.

Table 3.2 and 3.3 show the equations for the amount of pick wear expected in hard rock such as sandstone and shale, as well as coal. The equations in Table 3.2 and 3.3 were derived from pick wear data collected from a road header working in similar conditions. Best fit lines of the data were generated and the corresponding equations are displayed in

the Tables 3.2 and 3.3. As both machines exert similar forces on the picks the amount of wear of a road header is fairly transferable to that of a SM (Bauer, 2011).

Table 3-2 Pick Wear Equations for Hard Rock

	Unconfined Compressive Strength (x)			
	0-5 MPA	5-30 MPA	30-70 MPA	70 MPA +
5.0	$y = 0.0138x - 0.0026$	$y = 0.0047x^{1.4595}$	$y = 0.0326x - 0.2617$	$y = -7E-05x^2 + 0.0202x + 1.1617$
4.0	$y = 0.0002x^2 + 0.0043x + 0.0005$	$y = 0.0016x^{1.568}$	$y = 0.0281x - 0.6017$	$y = -0.0002x^2 + 0.0547x - 1.5822$
Cerchar 3.0	$y = 1E-04x^2 + 0.0017x + 0.0011$	$y = 0.0006x^{1.6334}$	$y = 0.0142x - 0.3167$	$y = 2E-05x^2 + 0.0197x - 0.9434$
Abrasivity 2.0	$y = 0.0011x + 0.001$	$y = 6E-05x^2 + 0.0004x + 0.0014$	$y = 0.0071x - 0.1713$	$y = 0.0566e^{0.0239x}$
Index 1.5	$y = 0.0006x + 0.0012$	$y = 3E-05x^2 - 8E-05x + 0.0055$	$y = 0.0119e^{0.0373x}$	$y = 0.0186e^{0.0308x}$
1.0	$y = 0.0003x + 0.0013$	$y = 8E-06x^2 + 1E-04x + 0.0024$	$y = 0.0035e^{0.041x}$	$y = 0.003e^{0.0425x}$
0.6	$y = 8E-05x + 0.0013$	$y = 4E-06x^2 - 1E-05x + 0.0018$	$y = 0.0013e^{0.0462x}$	$y = 0.001e^{0.0489x}$

Table 3-3 Pick Wear Equations for Coal

	Unconfined Compressive Strength (x)		
	0-5 MPA	5-30 MPA	30 MPA +
3.0	$y = (1E-04x^2 + 0.0017x + 0.0011) * .5038$	$y = (0.0006x^{1.6334}) * .5038$	$y = (0.0142x - 0.3167) * .5038$
Cerchar 2.0	$y = (0.0011x + 0.001) * .5038$	$y = (6E-05x^2 + 0.0004x + 0.0014) * .5038$	$y = (0.0071x - 0.1713) * .5038$
Abrasivity 1.5	$y = (0.0006x + 0.0012) * .5038$	$y = (3E-05x^2 - 8E-05x + 0.0055) * .5038$	$y = (0.0119e^{0.0373x}) * .5038$
Index 1.0	$y = (0.0003x + 0.0013) * .5038$	$y = (8E-06x^2 + 1E-04x + 0.0024) * .5038$	$y = (0.0035e^{0.041x}) * .5038$
0.6	$y = (8E-05x + 0.0013) * .5038$	$y = (4E-06x^2 - 1E-05x + 0.0018) * .5038$	$y = (0.0013e^{0.0462x}) * .5038$

The equations shown in the Table 3.2 and 3.3 are dependent on the unconfined compressive strength (UCS), measured in MPa, and the abrasivity of the rock, as measured by the Cerchar Abrasive Index, CAI. The “x” variable in each equation refers to the unconfined compressive strength of the rock, while the “y” variable is the amount of expected pick wear in picks per bank cubic meter. The pick wear, when plotted, shows an almost uniform wear as the UCS and abrasiveness increase. Because there are irregularities, equations have been derived for each value on the Cerchar Abrasive Index as well as for a range of unconfined compressive strengths.

Graphical representations of the equations in Tables 3.2 and 3.3 are displayed in the results section showing the relationship between pick wear and volume of material that can be cut until replacement is needed.

In addition to the pick wear in hard rocks, these values must also be obtained for wear in a softer material such as bituminous coal (Table 3.3). The pick wear value is determined by the characteristics, abrasivity and unconfined compressive strength, of the rock, which were described in module #1. It is necessary to have a description of the softer bituminous coal because the unconfined compressive strength and abrasivity are not the only factors affecting the amount of pick wear. Rock properties, such as rock structure (solid, fractured, macro/micro seamy), and grain size are a few factors that cause vastly different pick wear values at the same unconfined compressive strengths and abrasiveness.

The pick wear equations as stated previously are derived from the application of a road header in similar rock types. This among other factors cause the equations to generate pick wear values that are higher than what may actually be experienced. These values are acceptable but conservative.

With the known pick consumption per bank cubic meter, the amount of pick changes per row cut can be calculated. This value is needed to calculate the real cutting time of the SM. In the case of this research, the values for pick wear (PW_{SI}) generated by the equations in Table 3.2 and 3.3, are stated in picks per bank cubic meter. To avoid conversions in all equations using pick wear, a simple conversion factor of 35.32 ft³ per m³ will be used. The pick wear is represented in picks per bank cubic foot as follows:

$$PW = PW_{SI} * 35.32 \quad (3.5)$$

In addition to calculating pick wear in terms of picks per bank cubic foot, it is good practice to find the pick wear per bank cubic yards and tons. The equations used to determine the various pick wear values are as follows:

$$PW_{yd3} = PW * 27 \quad (3.6)$$

$$PW_{ton} = \frac{PW}{Density} \quad (3.7)$$

The number of times the picks need to be changed is found by dividing the volume of material in the row length, (RL) in ft, by the pick wear (PW). It is common practice to

replace a minimum of five picks per change sequence. With the implementation of a pick changing tool, the time to change one pick has been reduced to approximately one minute. The equation for the number of pick changes per row (NPC) is as follows:

$$NPC = \frac{RL * D * W}{PW * 5} \quad (3.8)$$

The pick wear information in this research is provided in picks per bank cubic meter, which may not always be the case. Pick wear can also be measured in forward distance of the machine. For instance, the pick wear may be measured in the amount of picks per foot traveled while cutting. Equation 3.8 is slightly modified to accommodate the calculation of pick wear (PW_{ft}) in feet traveled while cutting. Equation 3.9 represents the number of pick changes (NPC_L) required when pick wear is calculated in cutting distance.

$$NPC_L = \left(\frac{RL}{PW_{ft} * 5} \right) \quad (3.9)$$

When the cutting performance has been determined, delays can be factored in to provide a practical cutting performance value. Delays such as time spent on sumping in and out of the cut (ST), turning (TT), pick changing (PCT), and other non-cutting time (NCT) lower the real cutting time. Truck haulage adds additional delay time in the calculations based on the time to change trucks after loading. The total time taken to process one row length, including all delays, is known as the total time per row (TTR).

The first variable needed to find the total time per row is the time the SM is cutting material. The cutting time per row (CTR) is expressed in minutes as follows:

$$CTR = \left(\frac{RL}{V} \right) \quad (3.10)$$

The second variable in the total time equation is the time spent on changing picks. With the number of changes per row obtained from Equation 3.8, the time spent on pick changing, (PCT) in minutes, can be calculated as follows:

$$PCT = NPC * 5 \quad (3.11)$$

The remaining delay time components are calculated in Equations 3.12, 3.13, and 3.14. When the SM starts a cut, there is a span of distance in which the cutting drum is being lowered until the appropriate depth is reached. This distance also exists at the end of a cut when the machine raises the cutting drum. When these two distances are combined the total sump length, (TSL) in ft, is determined. The sumping segments are cut at a designated sumping speed, (SS) in ft/min. The sumping time is determined in minutes as follows:

$$ST = \left(\frac{TSL}{SS} * \# \text{ of Sumps per row} \right) \quad (3.12)$$

The SM travels at higher rates of speed while not cutting. The travel speed, (TS) in ft/min, is applied to delays that do not involve cutting, such as turning time, (TT) in minutes, and other non-cutting time, (NCT) in minutes. The equations for these delays, calculated in minutes, are as follows:

$$TT = \left(\frac{\text{Turning Radius}}{TS} \right) \quad (3.13)$$

$$NCT = \left(\frac{\text{Non Cutting Length}}{TS} \right) \quad (3.14)$$

When all of the delay components have been determined, the total time per row (TTR) is calculated in hours as follows:

$$TTR = (CTR + PCT + ST + TT + NCT)/60 \quad (3.15)$$

The Equation 3.1 must account for the delays in the cutting sequence to achieve an accurate cutting performance. The total operating hours that the SM is cutting material is known as the real cutting time (RCT), represented in hrs/year, which is a percentage (RCT%) of the total operating hours per year (OHY). The RCT% is a result of the amount of time spent cutting a row divided by the total time spent on cutting and non-cutting time for that row as shown by equation 3.16.

$$RCT\% = \frac{CTR}{TTR} \quad (3.16)$$

$$RCT = RCT\% * OHY \quad (3.17)$$

The cutting performance value with applied delays is called the practical cutting performance, (PQ) in bft³/hr, and is achieved by multiplying the cutting performance by the percentage of real cutting time as follows:

$$PQ = RCT\% * Q \quad (3.18)$$

This practical cutting performance value can be multiplied by the operating hours per year (OHY) to show the maximum production (MP) of the SM in a specific scenario and is calculated in bank yd³/year as follows:

$$MP = PQ * OHY * 27 \quad (3.19)$$

The MP is a useful value when considering what size of SM and the number of SMs required to meet production goals. If more than one SM is needed to meet production requirements, the equations can be modified to calculate the number of required SMs. Multiple SMs are needed if the maximum production of the SM is less than the required production RP, i.e.

$$\text{True or False } RP > MP \quad (3.20)$$

If more than one machine is needed, the required production will be divided equally among all SMs. This defines a new required production per SM.

When the maximum production of the SM exceeds the required production, a variable called the real operating hours per year (ROHY) needs to be considered. To accurately compare both mining methods, their annual production must be the same. Therefore, if the ROHY is less than the OHY, the ROHY will be used for any calculations using the OHY.

There is an alternative option to calculate the practical cutting performance and it is used as a check to verify its validity. The practical cutting performance method is verified by multiplying the theoretical cutting performance, (TQ) in ft/hr, by the cutting dimensions. The TQ can be calculated without including the SM's cutting dimensions. It is measured in feet per hour instead of cubic feet per hour. This is done based on the time spent cutting. The TQ is calculated by dividing the RL by the TTR as follows:

$$TQ = \frac{RL}{TTR} \quad (3.21)$$

The theoretical cutting performance can then be multiplied by the operating hours per year to find the total distance cut per year. Although this is a theoretical value, the delay times experienced during the cutting process are considered because they are included in the total time per row variable. The distance cut per year is determined by:

$$TQ_Y = TQ * OHY \quad (3.22)$$

The theoretical cutting performance can be used to verify the practical cutting performance as stated previously. To do this, the theoretical cutting performance is multiplied by the dimensions of the cutting drum and the cutting depth to obtain the PQ.

$$PQ = TQ * W * D \quad (3.23)$$

An additional validation of the practical cutting performance is the maximum production, (MP) per year in byd³/year. This is the yearly theoretical cutting performance multiplied by the cutting dimensions and a conversion factor, 27 ft³/yd³, as follows:

$$MP = TQ_Y * W * D * 27 \quad (3.24)$$

The TQ may also be necessary when pick wear is described in picks per distance cut.

The SM ownership and operating costs are calculated once the specifics and capabilities of the SM have been determined. The ownership cost equations come from Caterpillar (2009) and are used for every piece of equipment in this research. Variables used in these calculations are machine purchasing cost (PC) in dollars, ownership period (OP) in yrs, operating hours per year (OHY) in hrs, depreciation rate (DR) as a percentage, interest rate (INTR) as a percentage, insurance rate (INSR) as a percentage, tax rate (TXR) as a percentage, and the residual value rate (RV) as a percentage. The equations

for depreciation value per year (DV) in \$/yr, the value to be recovered through work (VR) in dollars, capital cost (CC) in \$/hr, interest cost (INT) in \$/hr, insurance cost INS in \$/hr, and tax cost (TX) in \$/hr are as follows:

$$DV = DR * PC \quad (3.25)$$

$$RV = RV\% * PC \quad (3.26)$$

$$VR = PC - RV \quad (3.27)$$

$$CC = (PC - RV)/(OP * OHY) \quad (3.28)$$

$$INT = \frac{\frac{OP + 1}{2 * OP} * PC * \frac{INTR}{100}}{OHY} \quad (3.29)$$

$$INS = \frac{\frac{OP + 1}{2 * OP} * PC * \frac{INSR}{100}}{OHY} \quad (3.30)$$

$$TX = \frac{\frac{OP + 1}{2 * OP} * PC * \frac{TXR}{100}}{OHY} \quad (3.31)$$

The ownership costs (OC) are calculated after all previous cost values have been determined. The ownership cost is the addition of all costs represented in \$/hr as follows:

$$OC = CC + INT + INS + TX \quad (3.32)$$

The operating cost (OPC) is the summation of the repair (RC), fuel (FC), grease (GC), hydraulic oil (HO), water (WC), picks (PTC), and labor (LBC) stated \$/hr.

$$OPC = RC + FC + GC + HO + WC + PTC + LBC \quad (3.33)$$

The costs for every element except for the cost of picks can be seen in Equations 3.34-3.38, Caterpillar (2010) and Kecojevic (2010). The RC uses the machine purchasing cost and OHY as follows:

$$RC = (8\% * PC) / OHY \quad (3.34)$$

The fuel consumption rate, typically provided by the manufacturer, is used to calculate the fuel cost. The manufacturer provided fuel consumption rate generally describes the motor under full load capacity (FLC). The machine will not likely operate at full load capacity for long periods of time because of the risk of pushing the machine to its failure point. Therefore, a load coefficient factor (LCF) is used to estimate the average load on the motor. In the calculations for this research, a value of 70 percent is used for the LCF (Wirtgen, 2010). The equations used to calculate fuel and grease cost are as follows:

$$FC = FLC * LCF * Fuel Cost \quad (3.35)$$

$$GC = 8\% * FC \quad (3.36)$$

The cost of hydraulic oil, (HO) in \$/hr, is calculated by the replacement time intervals in hrs of the oil (HCI) divided by the capacity of the hydraulic oil tank (VHO). The cost of water, (WC) in \$/hr, is determined by the consumption rate multiplied by the water cost. These equations are as follows:

$$HO = \frac{VHO}{HCI} * Hydraulic\ Oil\ Cost \quad (3.37)$$

$$WC = Consumption\ Rate * Water\ Cost \quad (3.38)$$

The pick cost is dependent on how the pick wear is described; per volume or distance cut. Before the cost of replacement picks can be found, the number of picks consumed yearly (PCY) must be calculated. Equation 3.24 calculates the SM's maximum production. As stated when explaining the ROHY, only the costs for achieving the exact required production needs to be calculated. This usable production (UP) value will be equal to the required production unless more than one SM is used to meet the requirement. If one SM is used, then Equation 3.39 is true; if multiple SMs are used, then Equation 3.40 is true.

$$UP = RP \quad (3.39)$$

$$UP = \frac{RP}{\#\ of\ Surface\ Miners} \quad (3.40)$$

It is useful to define the amount of picks per set (PPS) for purchasing reasons. This value depends on the width of the cutting drum and the spacing between the picks on the drum. The spacing calculation is in SI units. The cutting drum width, (W) in meters, is based on the size of the SM and the pick spacing, (PS) in millimeters, is determined by the type of rock and the desired size of the product.

$$PPS = \left(\frac{W * 1000}{PS * 1.1} \right) \quad (3.41)$$

The amount of picks used to extract the exact amount of material is calculated after the usable production is determined, (the pick wear in this research is expressed as picks per volume). Time per set or per pick can also be calculated from the value found in Equation 3.42 by using simple arithmetic. The amount of picks consumed per year (PCY) is calculated by dividing the usable production (UP) by the pick wear. The conversion factor $27 \text{ ft}^3/\text{yd}^3$ is applied to the equation for PCY and is as follows:

$$PCY = \frac{UP}{PW * 27} \quad (3.42)$$

Multiplication of the values calculated in Equations 3.43 and 3.44 by the amount of material will yield the same result as calculated in Equation 3.42. The cost of pick consumption (PTC) is calculated from the amount of picks consumed per year. The equation can be varied to represent this value in the desired units, but it is calculated in \$/hr to follow the previous cost calculations.

$$PTC = \frac{PCY * Pick Cost}{OHY} \quad (3.43)$$

The theoretical values are also used to verify the pick consumption and cost calculations. The pick wear, (PW_L) in pick/ft, is used with the theoretical cutting performance to find the pick consumption per year.

$$PCY = TQ_Y * PW_L \quad (3.44)$$

The value for pick consumption is based on the distance the SM cuts per year. The practical and theoretical pick consumption values are compared to check the validity of the pick consumption calculation. The pick consumption is correct if these two values are the same.

The calculations for the SM are broken down into three excavation scenarios; the overburden, interburden, and coal extraction. The same cost equations are used for the overburden, interburden, and coal extraction except for the different values for the pick wear, operating hours, and material characteristics. The costs for all three scenarios are calculated in \$/hr and are converted to \$/yd³ for overburden and interburden and \$/ton for coal for the mining method comparison. The SM(s) unit operation costs are compared to the conventional mining method costs to reveal the most economical mining method.

3.7 Comparison of Mining Methods (Module 6)

In this module, the conventional mining method is compared to the SM mining method. By comparing the overburden, interburden, and coal mining unit operations, the most economical method is determined. To insure both methods are comparable, all of the final costs are represented in dollars per cubic yard for overburden and interburden and in dollars per ton for coal. The comparison of methods uses the values derived by the calculations based on the material properties and characteristics described in module one. The results of the comparison module should not be taken as a rule-of-thumb for the entire state of WV. This comparison will provide results based on the average rock and material properties. In order to ultimately determine what method is the most

economical, the site specific material properties must be known and then site specific calculations can be performed and provide a location specific result.

3.7.1 Analytical Hierarchy Process for the Selection of the Optimal Mining Method

This research uses the analytical hierarchy process (AHP), developed by Saaty (1980), to evaluate conventional mining methods with that of a surface miner. The AHP is suitable for this application because of its ability to evaluate the qualitative and quantitative values derived in this research. The AHP in this research breaks the problem of selecting the optimal mining method into four levels; goals, strategic issues, criterion, and alternatives. The AHP model developed for the selection of the optimal mining method can be seen in Figure 3.4. The problem is broken into levels based on the level of importance of a specific criterion. By defining the hierarchy of criteria the problem is analyzed in smaller more manageable problems.

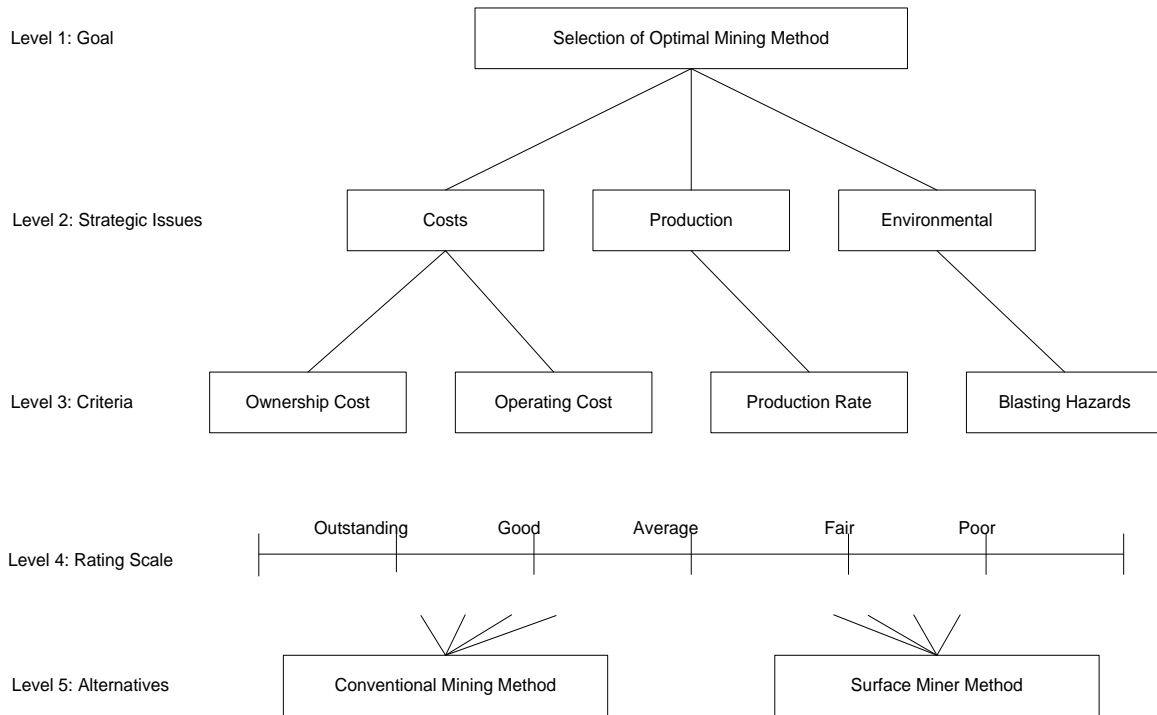


Figure 3-4 AHP Model for Selection of Optimal Mining Method

The optimal mining method can be selected when the levels have been defined and arranged into pairwise comparison matrices. The matrices are constructed based on the number of candidate requirements. For the model developed in this research, it uses two “ $n \times n$ ” matrices; one “ 3×3 ” matrix for the level two strategic issues and a “ 2×2 ” matrix for the level three criteria. No comparison matrix is needed for the production and environmental strategic issues because they only contain one criterion. With the matrices constructed, pairwise comparisons of all the requirements are performed. To assign a numerical value to each pair of requirements in the matrix, the scale created by Saaty (2008) is used to and can be seen in Table 3.4.

Table 3-4 The Fundamental Scale of Absolute Numbers (Saaty, 2008)

<i>Intensity of Importance</i>	<i>Definition</i>	<i>Explanation</i>
1	Equal Importance	Two activities contribute equally to the objective
2	Weak or Slight	
3	Moderate Importance	Experience and judgement slightly favour one activity over another
4	Moderate Plus	
5	Strong Importance	Experience and judgement strongly favour one activity over another
6	Strong Plus	
7	Very Strong or Demonstrated Importance	An activity is favoured very strongly over another; its dominance demonstrated in practice
8	Very, Very Strong	
9	Extreme Importance	The evidence favouring one activity over another is of the highest possible order of
Reciprocals of above	If activity i has one of the above non-zero numbers assigned to it when compared with activity j , then j has the reciprocal value when compared with i	
1.1-1.9	If the activities are very close	May be difficult to assign the best value but when compared with other contrasting activities the size of the small numbers would not be too noticeable, yet they can still indicate the relative importance of the activities.

Eigenvalues, a.k.a. the priority matrix, are generated from the comparison matrix when each criteria comparison has been given a value. The eigenvalues, or priority matrix, represent the weights of each criteria considered for the comparison matrix. The higher the weight of a criterion, the higher the influence it will have on the final selection.

The priority matrix represents the weight of each criterion. This provides weights for each specific criterion but does not account for the weights of the other level's criteria for the same alternative. The last level of criteria/sub-criteria/etc. in an AHP model have Global Weights (GW) assigned to them. The GW considers each criterion, and its weight, from any previous level. For example, if Figure 3.5 would be the AHP model a GW would be assigned to the sub-criteria.

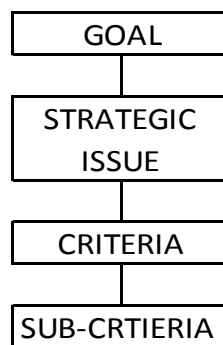


Figure 3-5 Example AHP Model

The GW calculated for the sub-criteria would account for the local weights of the strategic issue, the criteria, and the sub-criteria. A realistic AHP model will have more than one for the categories in Figure 3.5. Each alternative combines of all GW associated with it and finally the alternative with the highest value will be the best alternative.

To consider the differences between mining methods, it is necessary to rank the criteria for each mining method (level 4). A ranking system developed by Liberatore et al. (1992) uses five ratings: Outstanding (O), Good (G), Average (A), Fair (F), and Poor (P).

This system was used by Bascetin (2006) in formulating a pairwise comparison judgment matrix to assign values to the five different ratings as seen in Table 3.5.

Table 3-5 Rank Values

Rank	Numerical Value
Outstanding (O)	0.513
Good (G)	0.261
Average (A)	0.129
Fair (F)	0.063
Poor (P)	0.034

A rating and score are assigned at the author's discretion to each of the criteria. By multiplying the score and the global weight and summing all these values for each mining method, a total score for the mining method is achieved. Finally, after normalizing each total score, the higher score is the determined to be the optimal mining method.

The Consistency Index (CI) is calculated to measure the errors in judgment by the user. This can be explained as the consistency of the user's judgments for each comparison in the matrix. An example of a user with consistent judgments would be as follows:

- A is deemed more important than B
- B is deemed more important than C
- A is therefore more important than C

This type of input into a comparison matrix would display a consistent judgment making process and will likely provide valid results. An inconsistent judgment for the previous example would rate C more important than A.

To determine the CI, the maximum principle eigenvalue (λ_{\max}) of the comparison matrix is needed. The equation developed by Saaty (1980) for the CI is dependent on the λ_{\max} and the number or candidate requirements in the matrix “ n ” as shown in Equation 3.45.

$$CI = \frac{\lambda_{\max} - n}{n - 1} \quad (3.45)$$

Finally, Saaty (1980) proposed a method to determine if the matrix is acceptable, by the calculation of the Consistency Ratio (CR). The CR is based on the CI and the consistency indices of Randomly Generated Reciprocal Matrices (RI) developed by Saaty (1980). The RI values can be seen in Table 3.6.

Table 3-6 RI Values vs. "n" (Bascetin, 2006)

n	1	2	3	4	5	6	7	8	9	10
RI	0.00	0.00	0.58	0.90	1.12	1.24	1.32	1.41	1.45	1.49

The CR is calculated based on the “ n ” value equivalent to the comparison matrix and its corresponding RI value displayed in Table 3.6. The equation for the CR is as follows:

$$CR = \frac{CI}{RI[n]} \quad (3.46)$$

A general rule for an acceptable matrix is a CR value less than or equal to 0.10. The AHP can be determined quicker and more accurately by using software such as Expert Choice (2012), which uses the principles described here. This software is used to carry out the AHP in this research.

Chapter 4

Results and Analysis

4.1 Global Input Parameters

Throughout this study, variables such as fuel price, hydraulic oil price, electricity price, insurance rate, interest rate, and tax rate will remain unchanged and therefore are considered to be global parameters. The values compiled from InfoMine (2011) and Kecojevic (2010) are displayed in Table 4.1 as global parameters.

Table 4-1 Global Parameters

Parameters	Value	Units
Fuel Price	3.20	\$/gal
Hydraulic Oil Price	11.00	\$/gal
Electricity Price	0.07	\$/kWh
Insurance Rate	6.00	%
Interest Rate	1.00	%
Tax Rate	1.00	%

4.2 Results and Analysis

In this section, the results of the methodology described in chapter three is presented.

4.2.1 Drilling and Blasting

The cost assessment of the drilling and blasting operation was conducted for an annual production of 10 million bank yd³ of overburden material. Drilling and blasting

parameters are shown in Table 4.2. The drilling and blasting cost analysis are shown in Tables 4.3 and 4.4 respectively.

Table 4-2 Drilling and Blasting Parameters

Parameters	Sandstone	Shale	Units
Required production (RP)	10,000,000	10,000,000	bank yd ³ /year
Operating Hours per Year (OHY)	4,600	4,600	hrs
Bench height	50	50	ft
Hole diameter	6	6	in
Drillability Factor	1.8	2.0	
Overall Drilling Factor	0.70	0.70	
Explosive	ANFO	ANFO	
Detonation pressure	31	31	kbars
Density of explosive	0.82	0.82	g/cc

Table 4-3 Drilling Cost Analysis

Parameters	Sandstone	Shale	Units
Burden	12.90	14.93	ft
Drill Hole Spacing	16.78	19.41	ft
Sub-drilling	3.87	4.48	ft
Hole Length	54.70	55.32	ft
Stemming	12.90	14.93	ft
Particle Size for Stemming	0.50	0.50	in
Hole charge length	41.80	40.39	ft
Penetration Rate	89.59	98.99	ft/hr
Overall Drilling Rate	62.71	69.30	ft/hr
Specific drilling	0.14	0.10	ft/yd ³
Total Required Drilling	1,364,499	1,031,196	ft
Total drilling time	22,903	15,664	hrs
Number of required drills	5	4	
Machine Purchasing Cost (PC)	598,200	598,200	\$
Depreciation Value per Year (DV)	89,730	89,730	\$
Value to be Recovered Through Work (VR)	538,380	538,380	\$
Residual Value (RV)	59,820	59,820	\$
Capital Cost (CC)	19.51	19.51	\$/hr
Interest Cost (INT)	4.55	4.55	\$/hr
Insurance Cost (INSR)	0.76	0.76	\$/hr
Tax Cost (TX)	0.76	0.76	\$/hr
Ownership cost (OC)	25.58	25.58	\$/hr
Fuel Cost (FC)	33.60	33.60	\$/hr
Lube Cost (LC)	4.62	4.62	\$/hr
Maintenance Cost	23.93	23.93	\$/hr
Drill Accessory Cost	18.81	13.86	\$/hr
Operating cost (OPC)	100.96	96.01	\$/hr
Total drilling cost for all drills	632.69	486.33	\$/hr
Total Drilling cost	13,765,809	7,237,047	\$/year
Total Drilling cost	1.38	0.72	\$/bank yd³

Table 4-4 Blasting Cost Analysis

Parameters	Sandstone	Shale	Units
Hole charge concentration	10.04	10.04	lb/ft
Total charge per hole	419.52	405.40	lb
Volume of rock per ft of hole	8.02	10.73	yd ³
Volume of rock per hole	401	536	yd ³
Weight of Rock per Hole	849.93	751.04	ton
Number of required holes	24,944	18,641	
Total amount of explosive	10,464,391	7,556,998	lb
Powder factor	1.05	0.76	lb/yd ³
Hole to Hole Delay Time	50	50	ms
Row to Row Delay Time	125	150	ms
Explosive cost	5,441,483	3,929,639	\$
Detonator Cost	254,179	189,952	\$
Booster Cost	73,086	54,618	\$
Leadline & Tube Cost	80,458	69,682	\$
Time to Charge all Drill Holes	968.93	699.72	hrs
Blaster Labor	74,607	53,879	\$
Delivery Driver	59,686	43,103	\$
Travel, delivery and shot service	42,000	42,000	\$
Total blasting cost per year	6,025,500	4,382,872	\$/year
Total blasting cost	0.60	0.44	\$/bank yd³
Total drilling and blasting cost	1.98	1.16	\$/bank yd³

Drilling costs for sandstone and shale are \$1.38 and \$0.72 per bank yd³, respectively, while blasting costs are \$0.60 and \$0.44 per bank yd³. Total drilling and blasting costs for sandstone and shale are \$1.98 and \$1.16 per bank yd³, respectively.

4.2.2 Digging and Loading

Overburden

The handling of sandstone and shale overburden material is considered for the following digging and loading equipment: hydraulic shovel, electric (rope) shovel and front-end

wheel loader. The selection of the overburden removal machine is based on an annual production rate of 10 million bank yd³.

The parameters and cost analysis for the overburden hydraulic shovel is given in Tables 4.5 and 4.6.

Table 4-5 Overburden Hydraulic Shovel Parameters

<u>Parameters</u>	<u>Sandstone</u>	<u>Shale</u>	<u>Units</u>
Required production per year (RP)	10,000,000	10,000,000	bank yd ³ /year
Operating Hours per Year (OHY)	6,000	6,000	hrs
Number of Cycles	122	122	
Availability	90.00	90.00	%
Operating Efficiency	83.00	83.00	%
Fill Factor	85.00	90.00	%
Machine Purchasing Cost (PC)	7,000,000	6,100,000	\$
Horsepower	2,000	1,944	HP
Labor	35.00	35.00	\$/hr

Table 4-6 Overburden Hydraulic Shovel Cost Analysis

Parameters	Sandstone	Shale	Units
Hourly production	1,666.67	1,666.67	bank yd ³ /hr
Bucket volume	34.42	29.46	yd ³
Depreciation Value per Year (DV)	787,500	686,250	\$
Value to be Recovered Through Work (VR)	6,300,000	5,490,000	\$
Residual Value (RV)	700,000	610,000	\$
Capital Cost (CC)	131.25	114.38	\$/hr
Interest Cost (INT)	39.38	34.31	\$/hr
Insurance Cost (INSR)	6.56	5.72	\$/hr
Tax Cost (TX)	6.56	5.72	\$/hr
Ownership Cost (OC)	183.75	160.13	\$/hr
Fuel Cost (FC)	243.20	236.39	\$/hr
Maintenance, Wear Parts & Labor	175.00	152.50	\$/hr
Operating Cost (OPC)	453.20	423.89	\$/hr
Total Digging and Loading Cost	636.95	584.02	\$/hr
Total Digging and Loading Cost	3,821,700	3,504,092	\$/year
Total Digging and Loading Cost	0.38	0.35	\$/bank yd³

The total loading costs for sandstone and shale overburden material are \$636.95 and \$584.02 per hour, respectively, or \$3.82-million and \$3.50-million on an annual basis, respectively. The costs of digging and loading per bank yd³ are \$0.38 for sandstone and \$0.35 for shale material.

The parameters and cost analysis for the electric (rope) shovel is given in Tables 4.7 and 4.8.

Table 4-7 Overburden Electric Shovel Parameters

Parameters	Sandstone	Shale	Units
Required production per year (RP)	10,000,000	10,000,000	bank yd ³ /year
Operating Hours per Year (OHY)	6,000	6,000	hrs
Number of Cycles	120	120	
Availability	90.00	90.00	%
Operating Efficiency	83.00	83.00	%
Fill Factor	90.00	95.00	%
Machine Purchasing Cost (PC)	8,802,000	8,580,000	\$
Horsepower	3,000	2,700	HP
Average Electric Draw	32.6	34.6	%
Labor	35.00	35.00	\$/hr

Table 4-8 Overburden Electric Shovel Cost Analysis

Parameters	Sandstone	Shale	Units
Hourly production	1,666.67	1,666.67	bank yd ³ /hr
Bucket volume	33.05	28.38	yd ³
Depreciation Value per Year (DV)	352,080	343,200	\$
Value to be Recovered Through Work (VR)	7,041,600	6,864,000	\$
Residual Value (RV)	1,760,400	1,716,000	\$
Capital Cost (CC)	58.68	57.20	\$/hr
Interest Cost (INT)	46.21	45.05	\$/hr
Insurance Cost (INSR)	7.70	7.51	\$/hr
Tax Cost (TX)	7.70	7.51	\$/hr
Ownership Cost (OC)	120.29	117.26	\$/hr
Electricity Cost	51.07	48.78	\$/hr
Maintenance, Wear Parts & Labor	220.05	214.50	\$/hr
Operating Cost (OPC)	306.12	298.28	\$/hr
Total Digging and Loading Cost	426.42	415.54	\$/hr
Total Digging and Loading Cost	2,558,491	2,493,264	\$/year
Total Digging and Loading Cost	0.26	0.25	\$/bank yd³

The total loading costs for sandstone and shale overburden material are \$426.42 and \$415.54 per hour, respectively, or approximately \$2.56-million and \$2.49-million on an annual basis, respectively. The costs of digging and loading per bank yd³ are \$0.26 for sandstone and \$0.25 for shale material.

The parameters and cost analysis for the front-end wheel loader is given in Tables 4.9 and 4.10.

Table 4-9 Overburden Front-End Wheel Loader Parameters

<u>Parameters</u>	<u>Sandstone</u>	<u>Shale</u>	<u>Units</u>
Required production per year (RP)	10,000,000	10,000,000	bank yd ³ /year
Operating Hours per Year (OHY)	6,000	6,000	hrs
Number of Cycles	90	90	
Availability	90.00	90.00	%
Operating Efficiency	83.00	83.00	%
Fill Factor	85.00	90.00	%
Machine Purchasing Cost (PC)	6,700,000	6,500,000	\$
Fuel Consumption Rate	50.00	44.00	gal/hr
Labor	35.00	35.00	\$/hr

Table 4-10 Overburden Front-End Wheel Loader Cost Analysis

<u>Parameters</u>	<u>Sandstone</u>	<u>Shale</u>	<u>Units</u>
Hourly production	1,666.67	1,666.67	bank yd ³ /hr
Bucket volume	46.66	39.94	yd ³
Depreciation Value per Year (DV)	1,005,000	975,000	\$
Value to be Recovered Through Work (VR)	6,700,000	5,850,000	\$
Residual Value (RV)	670,000	650,000	\$
Capital Cost (CC)	167.50	162.50	\$/hr
Interest Cost (INT)	39.08	37.92	\$/hr
Insurance Cost (INSR)	6.51	6.32	\$/hr
Tax Cost (TX)	6.51	6.32	\$/hr
Ownership Cost (OC)	219.61	213.06	\$/hr
Fuel Cost (FC)	160.00	140.80	\$/hr
Tire Cost	66.00	60.00	\$/hr
Maintenance, Wear Parts & Labor	178.67	162.50	\$/hr
Operating Cost (OPC)	439.67	398.30	\$/hr
Total Digging and Loading Cost	659.28	611.36	\$/hr
Total Digging and Loading Cost	3,955,667	3,668,133	\$/year
Total Digging and Loading Cost	0.40	0.37	\$/bank yd³

The total loading costs for sandstone and shale overburden material are \$659.28 and \$611.36 per hour, respectively, or \$3.96-million and \$3.67-million on an annual basis, respectively. The costs of digging and loading per bank yd³ are \$0.40 for sandstone and \$0.37 for shale material.

Interburden

The bulldozer is designated for ripping and pushing interburden material. The bulldozer selected for this application is a Caterpillar D11T CD with a single shank or an equivalent machine of similar size and capability. The production rate is determined by referring to the production rate chart in Caterpillar (2009), which bases production on the seismic wave velocity of the material. If the thickness of interburden is larger than five feet or the seismic wave velocity increases, other methods for fracturing the rock, such as drilling and blasting, will be required.

The parameters and cost analysis for the bulldozer is given in Tables 4.11 and 4.12.

Table 4-11 Bulldozer Parameters

Parameters	Sandstone	Shale	Units
Required production per year (RP)	1,000,000	1,000,000	bank yd ³ /year
Operating Hours per Year (OHY)	3,000	3,000	hrs
Machine Purchasing Cost (PC)	1,870,000	1,870,000	\$
Seismic Wave Velocity	9,020	5,900	ft/sec
Labor	35.00	35.00	\$/hr

Table 4-12 Bulldozer Cost Analysis

Parameters	Sandstone	Shale	Units
Hourly production	334	334	bank yd ³ /hr
Depreciation Value per Year (DV)	336,600	336,600	\$
Value to be Recovered Through Work (VR)	1,683,000	1,683,000	\$
Residual Value (RV)	187,000	187,000	\$
Capital Cost (CC)	112.20	112.20	\$/hr
Interest Cost (INT)	22.44	22.44	\$/hr
Insurance Cost (INSR)	3.74	3.74	\$/hr
Tax Cost (TX)	3.74	3.74	\$/hr
Ownership Cost (OC)	142.12	142.12	\$/hr
Operating Cost (OPC)	179.00	160.00	\$/hr
Total Ripping and Pushing Cost	321.12	302.12	\$/hr
Total Ripping and Pushing Cost	963,360	906,360	\$/year
Total Ripping and Pushing Cost	0.96	0.91	\$/bank yd³

The total ripping and pushing costs for sandstone and shale interburden material are \$321.12 and \$302.12 per hour, respectively, or \$963,360 and \$906,360 on an annual basis, respectively. The total cost per yd³ is \$0.96 and \$0.91 per yd³, respectively.

After the material has been ripped and pushed, a backhoe shovel is used for loading the material into haul trucks. The selection of a hydraulic backhoe is based on an hourly production rate of 334 bank yd³ of interburden material. The parameters and cost analysis for the backhoe shovel is given in Tables 4.13 and 4.14.

Table 4-13 Interburden Backhoe Parameters

Parameters	Sandstone	Shale	Units
Required production per year (RP)	1,000,000	1,000,000	bank yd ³ /year
Operating Hours per Year (OHY)	3,000	3,000	hrs
Number of Cycles	122	122	
Availability	85.00	85.00	%
Operating Efficiency	83.00	83.00	%
Fill Factor	85.00	90.00	%
Machine Purchasing Cost (PC)	1,600,000	1,400,000	\$
Horsepower	760	730	HP
Labor	35.00	35.00	\$/hr

Table 4-14 Interburden Backhoe Cost Analysis

Parameters	Sandstone	Shale	Units
Hourly production	334	334	bank yd ³ /hr
Bucket volume	7.30	6.24	yd ³
Depreciation Value per Year (DV)	240,000	210,000	\$
Value to be Recovered Through Work (VR)	1,440,000	1,260,000	\$
Residual Value (RV)	160,000	140,000	\$
Capital Cost (CC)	80.00	70.00	\$/hr
Interest Cost (INT)	18.67	16.33	\$/hr
Insurance Cost (INSR)	3.11	2.72	\$/hr
Tax Cost (TX)	3.11	2.72	\$/hr
Ownership Cost (OC)	104.89	91.78	\$/hr
Fuel Cost (FC)	92.42	88.77	\$/hr
Maintenance, Wear Parts & Labor	80.00	70.00	\$/hr
Operating Cost (OPC)	207.42	193.77	\$/hr
Total Digging and Loading Cost	312.30	285.55	\$/hr
Total Digging and Loading Cost	936,915	856,637	\$/year
Total Digging and Loading Cost	0.94	0.86	\$/bank yd³

The total loading costs for sandstone and shale interburden material are \$312.30 and \$285.55 per hour, respectively, or \$936,915 and \$856,637 on an annual basis, respectively. The cost of loading per bank yd³ is \$0.94 for sandstone and \$0.86 for shale material.

4.2.3 Conventional Coal Extraction and Crushing

A hydraulic backhoe shovel is considered as the coal digging and loading equipment. An annual production of 1.5 million tons of bituminous coal is considered in this study. An overview of the parameters and cost assessment results for the hydraulic backhoe shovel is given in Tables 4.15 and 4.16.

Table 4-15 Coal Extraction Parameters

Parameters	Coal	Units
Required production per year (RP)	1,500,000	tons/year
Operating Hours per Year (OHY)	3,000	hrs
Number of Cycles	115	
Availability	85.00	%
Operating Efficiency	83.00	%
Fill Factor	85.00	%
Machine Purchasing Cost (PC)	1,800,000	\$
Horsepower	760	HP
Labor	35.00	\$/hr

Table 4-16 Coal Extraction Cost Analysis

Parameters	Coal	Units
Hourly production	500	tons/hr
Bucket volume	9.06	yd ³
Depreciation Value per Year (DV)	231,429	\$
Value to be Recovered Through Work (VR)	1,620,000	\$
Residual Value (RV)	180,000	\$
Capital Cost (CC)	77.14	\$/hr
Interest Cost (INT)	20.57	\$/hr
Insurance Cost (INSR)	3.43	\$/hr
Tax Cost (TX)	3.43	\$/hr
Ownership Cost (OC)	104.57	\$/hr
Fuel Cost (FC)	92.42	\$/hr
Maintenance, Wear Parts & Labor	96.00	\$/hr
Operating Cost (OPC)	223.42	\$/hr
Total Digging and Loading Cost	327.99	\$/hr
Total Digging and Loading Cost	983,962	\$/year
Total Digging and Loading Cost	0.66	\$/ton

The total digging and loading cost for bituminous coal is \$327.99 per hour or \$936,962 on an annual basis. The cost of digging and loading is \$0.66 per ton.

The selection of a crusher was based on the hourly production rate of 500 tons of coal being extracted by the backhoe shovel. The parameters and cost assessment results for the crusher is given in Tables 4.17 and 4.18.

Table 4-17 Crushing Parameters

Parameters	Coal	Units
Required production per year (RP)	1,500,000	tons/year
Operating Hours per Year (OHY)	3,000	hrs
Feed Size	15.0	in
Product Size	4.0	in
Material Lump Factor	2.0	
Bond's Work Index	11.37	kW/tonne
Drive Efficiency	0.85	
Machine Purchasing Cost (PC)	260,000	\$
Labor	25.00	\$/hr

Table 4-18 Crushing Cost Analysis

Parameters	Coal	Units
Hourly production	500	tons/hr
Reduction Ratio	3.8	
Bond's Work per Tonne	0.173	kWh/tonne
Power	240	HP
Depreciation Value per Year (DV)	21,273	\$
Value to be Recovered Through Work (VR)	234,000	\$
Residual Value (RV)	26,000	\$
Capital Cost (CC)	7.88	\$/hr
Interest Cost (INT)	2.84	\$/hr
Insurance Cost (INSR)	0.47	\$/hr
Tax Cost (TX)	0.47	\$/hr
Ownership Cost (OC)	11.66	\$/hr
Electricity Cost	12.89	\$/hr
Wear Parts	3.03	\$/hr
Spare Parts	2.60	\$/hr
Operating Cost (OPC)	43.52	\$/hr
Total Crushing Cost	55.18	\$/hr
Total Crushing Cost	165,537	\$/year
Total Crushing Cost	0.11	\$/ton

The total crushing cost for bituminous coal is \$55.18 per hour or \$165,537 million on an annual basis, and \$0.11 per ton.

4.2.4 Surface Miner Method

Two important parameters that need to be considered in the process of selection of SM are the unconfined compressive strength (UCS) and the abrasiveness of the rocks. The former is expressed in MPa or psi while the later by the Cerchar Abrasive Index (CAI). The CAI is used to describe the abrasiveness of a rock. The Cerchar Index was created in the 1970's by the Cerchar Institute in France (Rostami et al., 2005). The test consisted of scratching a rock surface with a steel pin and measuring the amount of wear, which defined the Cerchar Index. The amount of wear a pick experienced at varying values of abrasivity and UCS for both hard rock and coal can be seen in the Figure 4.1 and 4.2, respectively. These figures were obtained by plotting the pick wear equations described in chapter three. The equations and consequently the graphs are based on the pick wear data collected from a road header, which operated in similar conditions. Graphing the pick wear equations displays the amount of pick wear that can be expected based on the characteristics of the material. Referring to the pick wear graphs can provide an initial estimate of the cost of using the SM. It should be noted that the amount of wear caused by the hard rock and coal as calculated in Figures 4.1 and 4.2 may be higher than seen in the field. As expected, the more abrasive and the higher the compressive strength of the rock, the higher wear on the pick.

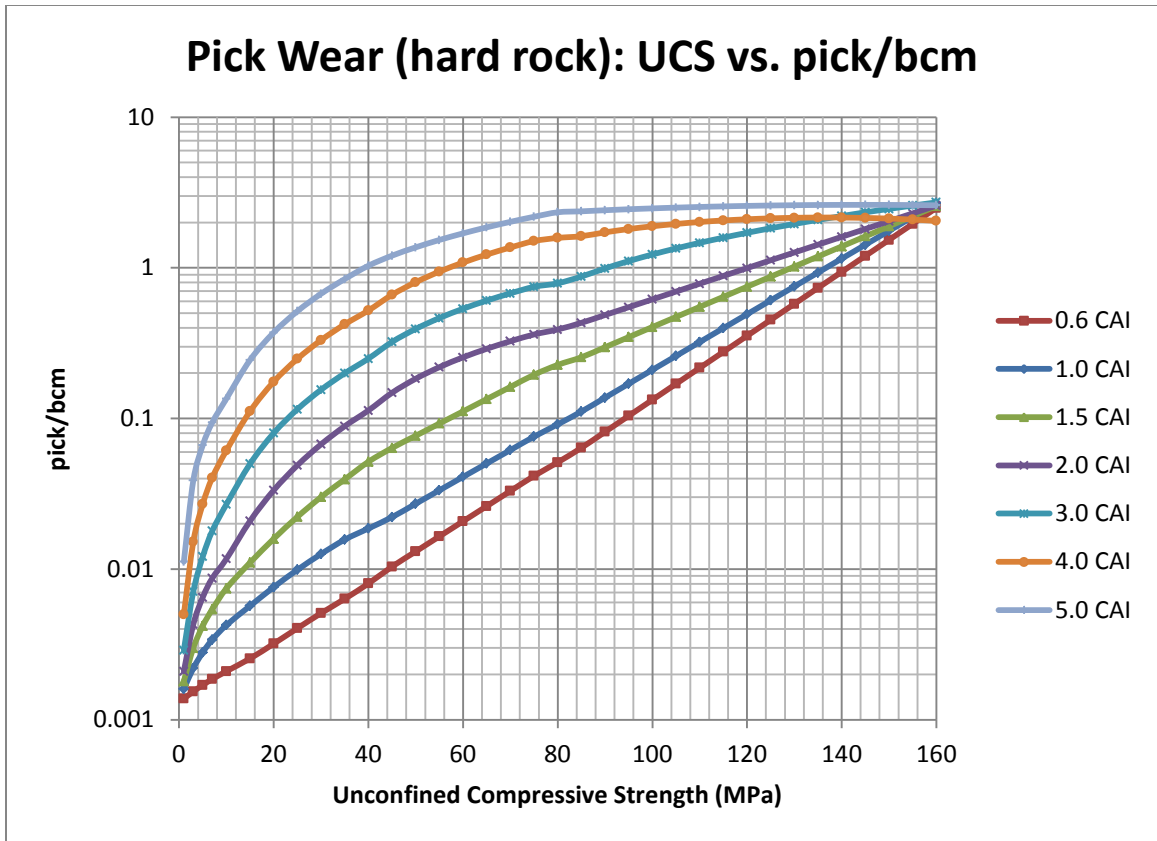


Figure 4-1 Pick Wear in Hard Rock

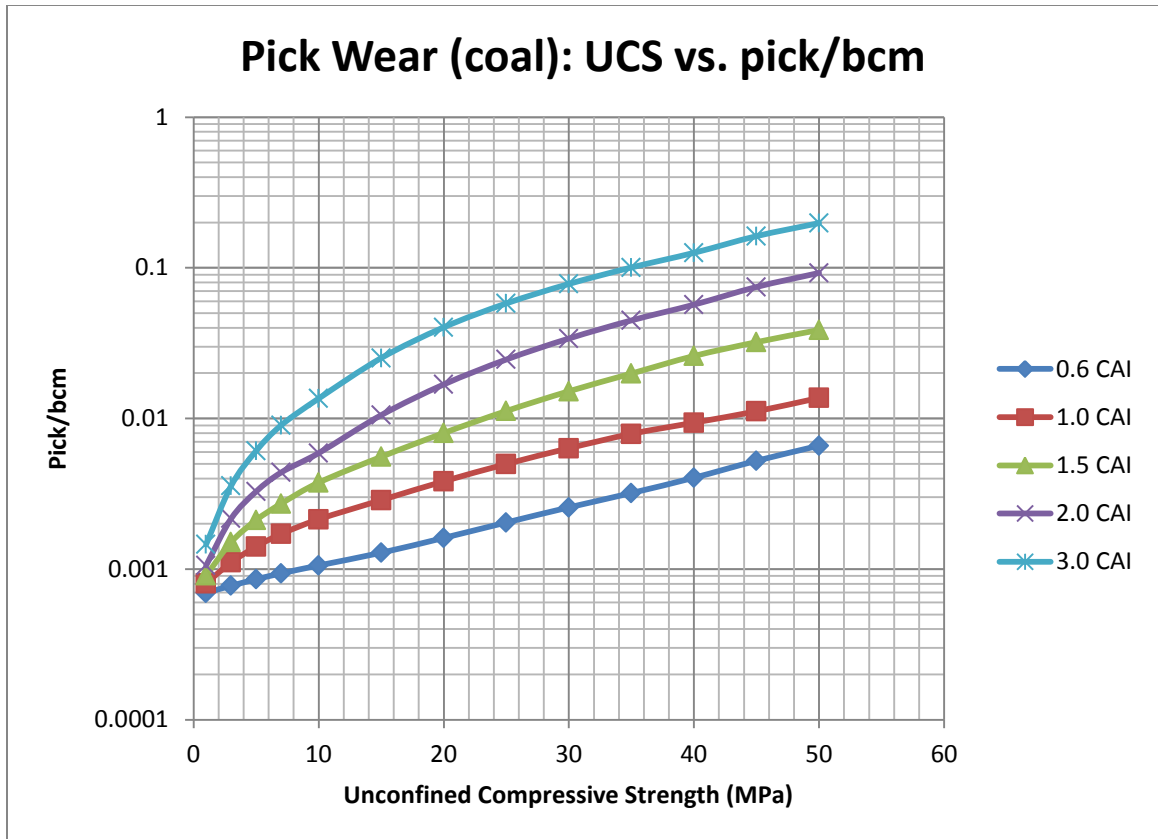


Figure 4-2 Pick Wear in Coal

Variations of practical cutting performance of the SM are given in Tables 4.19, 4.20, and 4.21 based on the compressive strength and abrasivity. These rates are given for the SM with the following properties: 8.2 ft cutting drum and an engine output approximately 780 kW. The practical cutting performance rates were calculated by using equation 3.18 described in chapter 3. The figures are color-coded to show the ranges of production rates for this particular SM.

Table 4-19 Surface Miner Performance in Sandstone (bank yd³/hr)

	Sandstone (MPa)						
	60	70	80	90	100	110	120
CAI=.6	630	495	383	287	207	143	93
CAI=1	542	419	320	238	172	120	80
CAI=1.5	364	273	206	160	120	89	63
CAI=2	219	174	144	116	91	70	53
CAI=3	123	98	83	66	53	44	36
CAI=4	66	53	45	41	37	33	30
CAI=5	44	37	32	30	29	27	26

Practical Cutting Performance (bank yd ³ /hr)	> 350	350-250	250-150	150-50	<50
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Table 4-20 Surface Miner Performance in Shale (bank yd³/hr)

	Shale (MPa)				
	60	70	80	90	100
CAI=.6	630	495	383	287	207
CAI=1	542	419	320	238	172
CAI=1.5	364	273	206	160	120
CAI=2	219	174	144	116	91
CAI=3	123	98	83	66	53
CAI=4	66	53	45	41	37

Practical Cutting Performance (bank yd ³ /hr)	> 350	350-250	250-150	150-50	<50
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Table 4-21 Surface Miner Performance in Coal (tons/hr)

	Coal (MPa)				
	10	20	30	40	50
CAI=.6	1126	840	668	544	447
CAI=1	1110	821	648	526	430
CAI=1.5	1087	789	607	477	382
CAI=2	1058	729	535	406	307
CAI=3	965	606	418	305	222

Practical Cutting Performance (ton/hr)	> 800	800-600	600-400	<400
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The parameters and cost analysis for the SM in overburden material is given in Tables 4.22 and 4.23.

Table 4-22 Surface Miner Parameters for Overburden

<u>Parameters</u>	<u>Sandstone</u>	<u>Shale</u>	<u>Units</u>
Required production per year (RP)	10,000,000	10,000,000	bank yd ³ /year
Operating Hours per Year (OHY)	6,000	6,000	hrs
Cutting Depth (CD)	1.48	1.48	ft
Cutting Width (CW)	8.20	8.20	ft
Cutting Velocity (V)	11.97	21.49	ft/min
Pick Wear (PW)	0.017489	0.005527	pick/ bank ft ³
Machine Purchase Cost (PC)	2,800,000	2,800,000	\$
Fuel Consumption at Full Load (FLC)	50.6	50.6	gal/hr
Load Coefficient Factor (LCF)	70.00	70.00	%
Cutting Pick Cost	17.04	17.04	\$/pick
Labor (LBC)	35.00	35.00	\$/hr

Table 4-23 Surface Miner Cost Analysis for Overburden

Parameters	Sandstone	Shale	Units
Cutting Performance (Q)	321.84	577.97	bank yd ³ /hr
Pick Changes per Row (NPC)	84.67	26.76	
Pick Changing Time (PCT)	423.34	133.78	min
Sump Time (ST)	1.00	1.00	min
Turning Time (TT)	0.92	0.92	min
Non-Cutting Time (NCT)	0.00	0.00	min
Real Cutting Time (RCT)	1,693	2,441	hrs
Practical Cutting Performance (PQ)	90.80	235.14	bank yd ³ /hr
Maximum Production (MP)	544,827	1,410,828	bank yd ³ /year
Number of Required Surface Miners	19	8	
Usable Production (UP)	526,316	1,250,000	bank yd ³ /year
Real Operating Hours per Year (ROHY)	5,796	5,316	hrs
Number of Picks Consumed (PCY)	248,525	186,526	picks/year
Depreciation Value per Year (DV)	252,000	252,000	\$
Value to be Recovered Through Work (VR)	2,520,000	2,520,000	\$
Residual Value (RV)	280,000	280,000	\$
Capital Cost (CC)	43.48	47.40	\$/hr
Interest Cost (INT)	15.94	17.38	\$/hr
Insurance Cost (INS)	2.66	2.90	\$/hr
Tax Cost (TX)	2.66	2.90	\$/hr
Ownership Cost (OC)	64.73	70.58	\$/hr
Repair Cost (RC)	38.65	42.14	\$/hr
Fuel Cost (FC)	113.32	113.32	\$/hr
Grease Cost (GC)	9.07	9.07	\$/hr
Hydraulic Oil Cost (HO)	1.45	1.45	\$/hr
Water Cost (WC)	0.00	0.00	\$/hr
Pick Cost (PTC)	730.63	597.89	\$/hr
Operating Cost (OPC)	928.12	804.71	\$/hr
Total Cost per Surface Miner	992.85	869.44	\$/hr
Total Cost per Surface Miner	5,754,720	4,621,994	\$/year
Total Surface Miner Cost	10.93	3.70	\$/bank yd³

The total cost per SM for sandstone and shale are \$992.85 and \$869.44 per hour, or \$5,754,720 and \$4,621,991 on an annual basis, respectively. The overall total cost per bank yd³ is \$10.93 for sandstone and \$3.71 for shale material.

The parameters and cost analysis for the SM in interburden material is given in Tables 4.24 and 4.25.

Table 4-24 Surface Miner Parameters for Interburden

<u>Parameters</u>	<u>Sandstone</u>	<u>Shale</u>	<u>Units</u>
Required production per year (RP)	1,000,000	1,000,000	bank yd ³ /year
Operating Hours per Year (OHY)	3,000	3,000	hrs
Cutting Depth (CD)	1.48	1.48	ft
Cutting Width (CW)	8.20	8.20	ft
Cutting Velocity (V)	11.97	21.49	ft/min
Pick Wear (PW)	0.017489	0.005527	pick/ bank ft ³
Machine Purchase Cost (PC)	2,800,000	2,800,000	\$
Fuel Consumption at Full Load (FLC)	50.6	50.6	gal/hr
Load Coefficient Factor (LCF)	70.00	70.00	%
Cutting Pick Cost	17.04	17.04	\$/pick
Labor (LBC)	35.00	35.00	\$/hr

Table 4-25 Surface Miner Cost Analysis for Interburden

Parameters	Sandstone	Shale	Units
Cutting Performance (Q)	321.84	577.97	bank yd ³ /hr
Pick Changes per Row (NPC)	84.67	26.76	
Pick Changing Time (PCT)	423.34	133.78	min
Sump Time (ST)	1.00	1.00	min
Turning Time (TT)	0.92	0.92	min
Non-Cutting Time (NCT)	0.00	0.00	min
Real Cutting Time (RCT)	846	1,221	hrs
Practical Cutting Performance (PQ)	90.80	235.14	bank yd ³ /hr
Maximum Production (MP)	272,413	705,414	bank yd ³ /year
Number of Required Surface Miners	4	2	
Usable Production (UP)	250,000	500,000	bank yd ³ /year
Real Operating Hours per Year (ROHY)	2,753	2,126	hrs
Number of Picks Consumed (PCY)	118,049	74,610	picks/year
Depreciation Value per Year (DV)	252,000	252,000	\$
Value to be Recovered Through Work (VR)	2,520,000	2,520,000	\$
Residual Value (RV)	280,000	280,000	\$
Capital Cost (CC)	91.53	118.51	\$/hr
Interest Cost (INT)	33.56	43.45	\$/hr
Insurance Cost (INS)	5.59	7.24	\$/hr
Tax Cost (TX)	5.59	7.24	\$/hr
Ownership Cost (OC)	136.28	176.45	\$/hr
Repair Cost (RC)	81.36	105.34	\$/hr
Fuel Cost (FC)	113.32	113.32	\$/hr
Grease Cost (GC)	9.07	9.07	\$/hr
Hydraulic Oil Cost (HO)	1.45	1.45	\$/hr
Water Cost (WC)	0.00	0.00	\$/hr
Pick Cost (PTC)	730.63	597.89	\$/hr
Operating Cost (OPC)	1,042.38	973.79	\$/hr
Total Cost per Surface Miner	1,107.11	1,038.52	\$/hr
Total Cost per Surface Miner	3,048,072	2,208,317	\$/year
Total Surface Miner Cost	12.19	4.42	\$/bank yd³

The total cost per SM for sandstone and shale are \$1,107 and \$1,039 per hour, or \$2,048,072 and \$2,208,317 on an annual basis, respectively. The overall total cost per bank yd³ is \$12.19 for sandstone and \$4.42 for shale material.

The parameters and cost analysis for the SM in bituminous coal is given in Tables 4.26 and 4.27.

Table 4-26 Surface Miner Parameters for Coal

Parameters	Coal	Units
Required production per year (RP)	1,500,000	tons/year
Operating Hours per Year (OHY)	3,000	hrs
Cutting Depth (CD)	1.48	ft
Cutting Width (CW)	8.20	ft
Cutting Velocity (V)	21.49	ft/min
Pick Wear (PW)	0.000108	pick/ bank ft ³
Machine Purchase Cost (PC)	2,800,000	\$
Fuel Consumption at Full Load (FLC)	50.6	gal/hr
Load Coefficient Factor (LCF)	70.00	%
Cutting Pick Cost	17.04	\$/pick
Labor (LBC)	35.00	\$/hr

Table 4-27 Surface Miner Cost Analysis for Coal

Parameters	Coal	Units
Cutting Performance (Q)	878.12	tons/hr
Pick Changes per Row (NPC)	0.52	
Pick Changing Time (PCT)	2.62	min
Sump Time (ST)	1.00	min
Turning Time (TT)	0.92	min
Non-Cutting Time (NCT)	0.00	min
Real Cutting Time (RCT)	2,807	hrs
Practical Cutting Performance (PQ)	821.49	bank yd ³ /hr
Maximum Production (MP)	2,464,477	tons/year
Number of Required Surface Miners	1	
Usable Production (UP)	1,500,000	tons/year
Real Operating Hours per Year (ROHY)	1,826	hrs
Number of Picks Consumed (PCY)	4,084	picks/year
Depreciation Value per Year (DV)	252,000	\$
Value to be Recovered Through Work (V)	2,520,000	\$
Residual Value (RV)	280,000	\$
Capital Cost (CC)	138.01	\$/hr
Interest Cost (INT)	50.60	\$/hr
Insurance Cost (INS)	8.43	\$/hr
Tax Cost (TX)	8.43	\$/hr
Ownership Cost (OC)	205.48	\$/hr
Repair Cost (RC)	122.68	\$/hr
Fuel Cost (FC)	113.32	\$/hr
Grease Cost (GC)	9.07	\$/hr
Hydraulic Oil Cost (HO)	1.45	\$/hr
Water Cost (WC)	0.00	\$/hr
Pick Cost (PTC)	38.11	\$/hr
Operating Cost (OPC)	460.38	\$/hr
Total Cost per Surface Miner	525.11	\$/hr
Total Cost per Surface Miner	958,825	\$/year
Total Surface Miner Cost	0.64	\$/ton

The total cost for bituminous coal is \$525.11 per hour or \$958,825 on annual basis. The cost per ton of bituminous coal is \$0.64. The combination of the values determined in this section form a benchmark for the each of the mining methods to be compared in the following section.

4.2.5 Comparison of Mining Methods

Figure 4.3 shows the cost of the extraction of sandstone overburden (UCS: 100 MPa, CAI: 2) for four different scenarios: drilling, blasting, and hydraulic shovel (scenario #1); drilling, blasting, and electric shovel (scenario #2); drilling, blasting, and front-end loader (scenario #3); and SM (scenario #4). Scenario #2 yields the lowest cost of \$2.24 per bank yd³, followed by, in increasing order, scenario #1 with the cost of \$2.36 per bank yd³, scenario #3 with the cost of \$2.38 per bank yd³, and scenario #4 with the cost of \$10.93 per bank cubic yard.

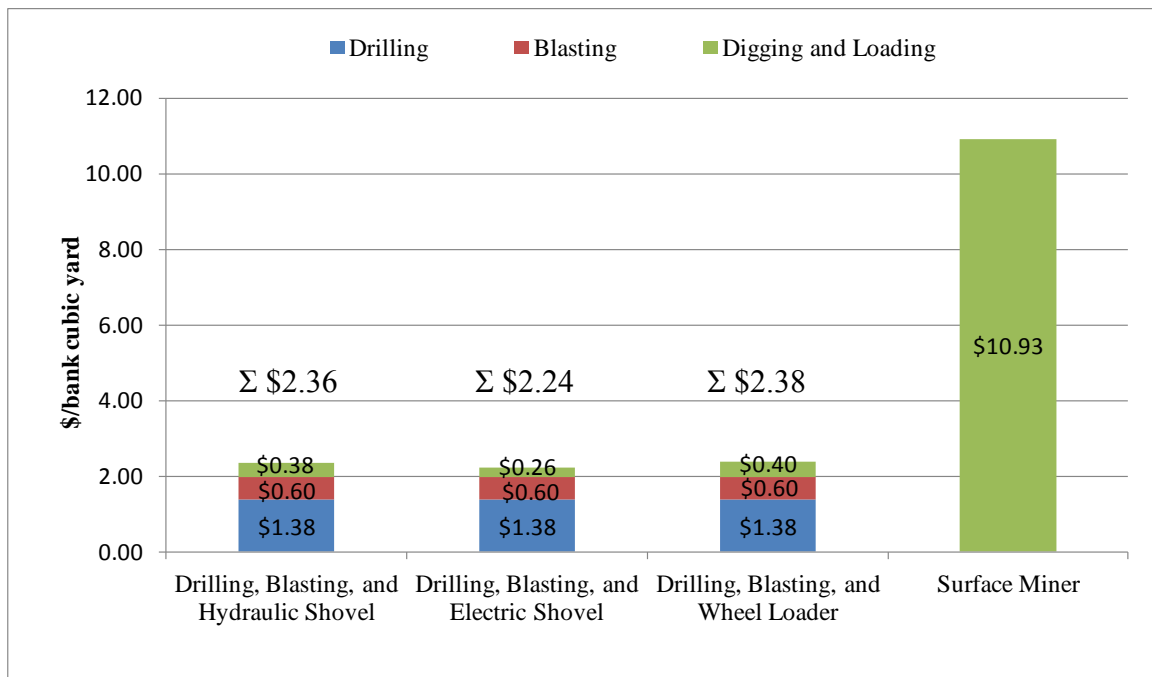


Figure 4-3 Conventional vs. SM Mining Method Comparison in Sandstone Overburden

Scenario #4, which uses only the SM, is much less economical than each of the other scenarios for the considered rock properties. The costs for a varying range of rock properties have been calculated for each of the applications of the SM. The variances in

costs based on the UCS and CAI of the material being extracted are displayed in Table 4.28. These values provide an idea for where the applicability of this machine could be. Table 4.28 shows that as the UCS and CAI decreases, the possible application of this machine increases. The material with higher UCS and CAI than what is shown results in costs that are obviously too high for any application and are disregarded.

Table 4-28 Total Cost of SM per Cubic Yard in Sandstone Overburden

OB	Sandstone (MPa)					
	60	70	80	90	100	110
CAI=.6	\$0.70	\$0.99	\$1.38	\$1.97	\$3.04	\$4.66
CAI=1	\$1.07	\$1.42	\$2.05	\$2.88	\$4.26	\$6.36
CAI=1.5	\$2.19	\$3.11	\$4.26	\$5.52	\$7.43	\$10.10
CAI=2	\$4.52	\$5.75	\$6.90	\$8.61	\$10.93	\$13.92
Total Cost (\$/yd ³)	<\$1.50		\$1.50-\$3.00	\$3.00-\$4.50	>\$4.50	

Figure 4.4 shows the cost of extraction of shale overburden (UCS: 75 MPa, CAI: 1.5) for four different scenarios: drilling, blasting, and hydraulic shovel (scenario #1); drilling, blasting, and electric shovel (scenario #2); drilling, blasting, and wheel loader (scenario #3); and SM (scenario #4). Once again, scenario #2 yields the lowest cost of \$1.41 per bank yd³, followed by, in increasing order, scenario #1 with the cost of \$1.51 per bank yd³, scenario #3 with the cost of \$1.53 per bank yd³, and scenario #4 with the cost of \$3.70 per bank yd³.

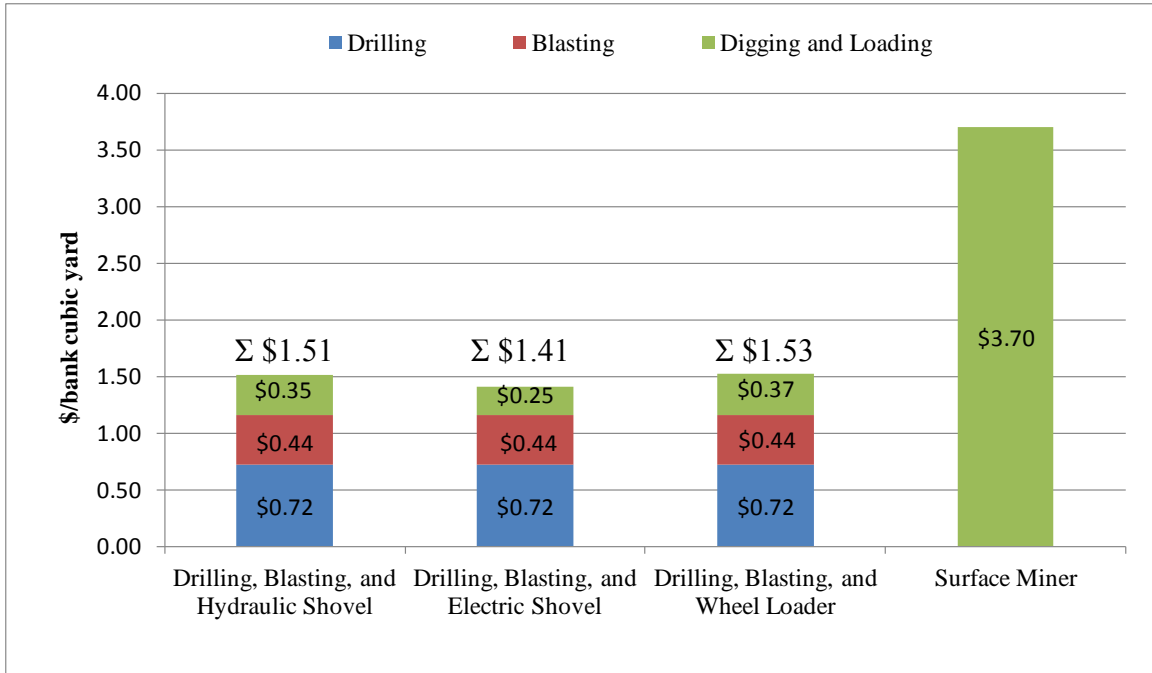


Figure 4-4 Conventional vs. SM Mining Method Comparison in Shale Overburden

Scenario #4 is much higher in cost when compared to the remaining three scenarios. This is the situation created when calculating the costs based on the average property values as stated in module one. Table 4.29 shows the SM costs for various values of UCS and CAI in shale overburden. The same trend as seen in sandstone overburden exists in shale, the lower the UCS and CAI, the lower the costs will be.

Table 4-29 Total Cost of SM per Cubic Yard in Shale Overburden

OB	Shale (MPa)						
	50	60	70	80	90	100	
CAI=.6	\$0.55	\$0.70	\$0.99	\$1.38	\$1.97	\$3.04	
CAI=1	\$0.76	\$1.07	\$1.42	\$2.05	\$2.88	\$4.26	
CAI=1.5	\$1.57	\$2.19	\$3.11	\$4.26	\$5.52	\$7.43	
CAI=2	\$3.30	\$4.52	\$5.75	\$6.90	\$8.61	\$10.93	
Total Cost (\$/yd ³)	<\$1.00		\$1.00-\$2.00		\$2.00-\$3.00		>\$3.00

The overburden calculations provide an estimated comparison that could be expected if the given conditions were encountered in the field. While the SM is more expensive in each type of overburden, it is much more expensive in the stronger and more abrasive sandstone than in shale. If the UCS and CAI are found to be lower, the likelihood of using the SM in shale will be higher than in sandstone.

Figure 4.5 shows the cost of extraction of sandstone interburden for two different scenarios: ripping and pushing by bulldozer and loading by backhoe shovel (scenario #1); and SM (scenario #2). Scenario #1 yields the cost of \$1.90 per bank yd³, while the cost for scenario #2 is \$12.19 per bank yd³.

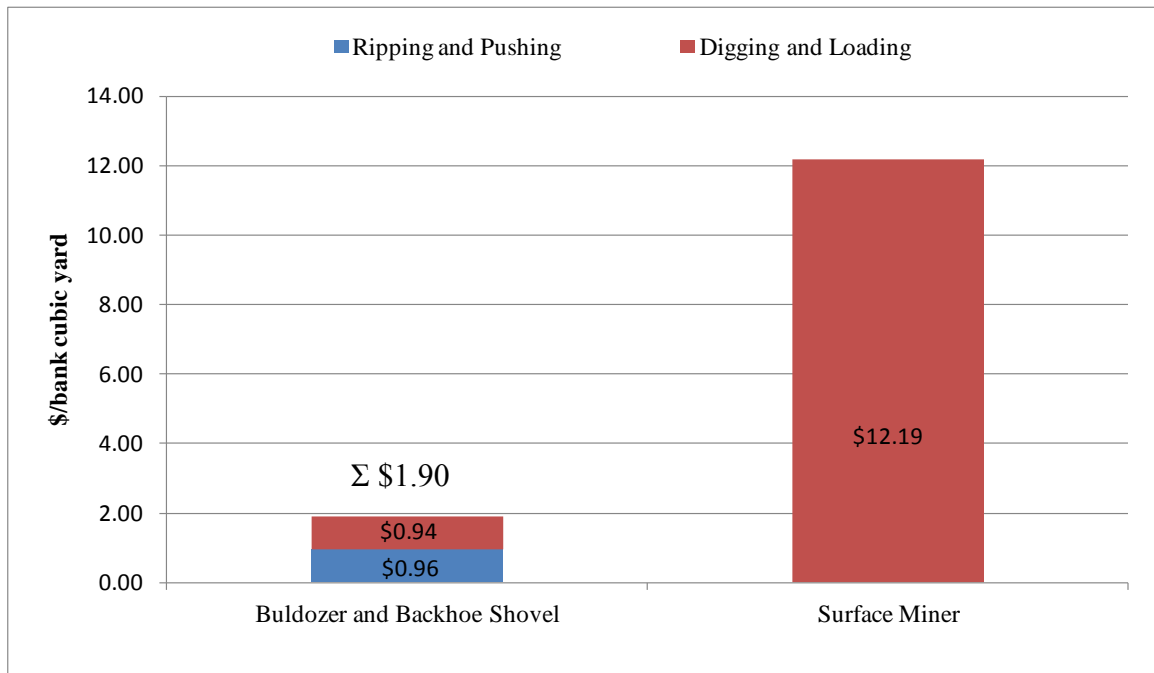


Figure 4-5 Conventional vs. SM Mining Method Comparison in Sandstone Interburden

Table 4.30 shows the SM costs for various values of UCS and CAI in sandstone overburden. The same trend as seen in sandstone overburden can be observed for sandstone interburden.

Table 4-30 Total Cost of SM per Cubic Yard in Sandstone Interburden

IB	Sandstone (MPa)					
	60	70	80	90	100	110
CAI=.6	\$1.12	\$1.35	\$1.68	\$2.81	\$3.70	\$5.74
CAI=1	\$1.43	\$1.78	\$2.89	\$3.66	\$4.86	\$7.31
CAI=1.5	\$2.49	\$3.89	\$4.92	\$6.66	\$8.39	\$11.36
CAI=2	\$5.24	\$6.35	\$7.98	\$9.51	\$12.19	\$15.48
Total Cost (\$/yd ³)			<\$2.00	\$2.00-\$3.00	\$3.00-\$4.00	>\$4.00

Figure 4.6 shows the cost of extraction of shale interburden material for two different scenarios: ripping and pushing by bulldozer and loading by backhoe shovel (scenario #1); and SM (scenario #2). Here, scenario #1 yields the cost of \$1.77 per bank yd³, while the cost for scenario #2 is \$4.42 per bank yd³.

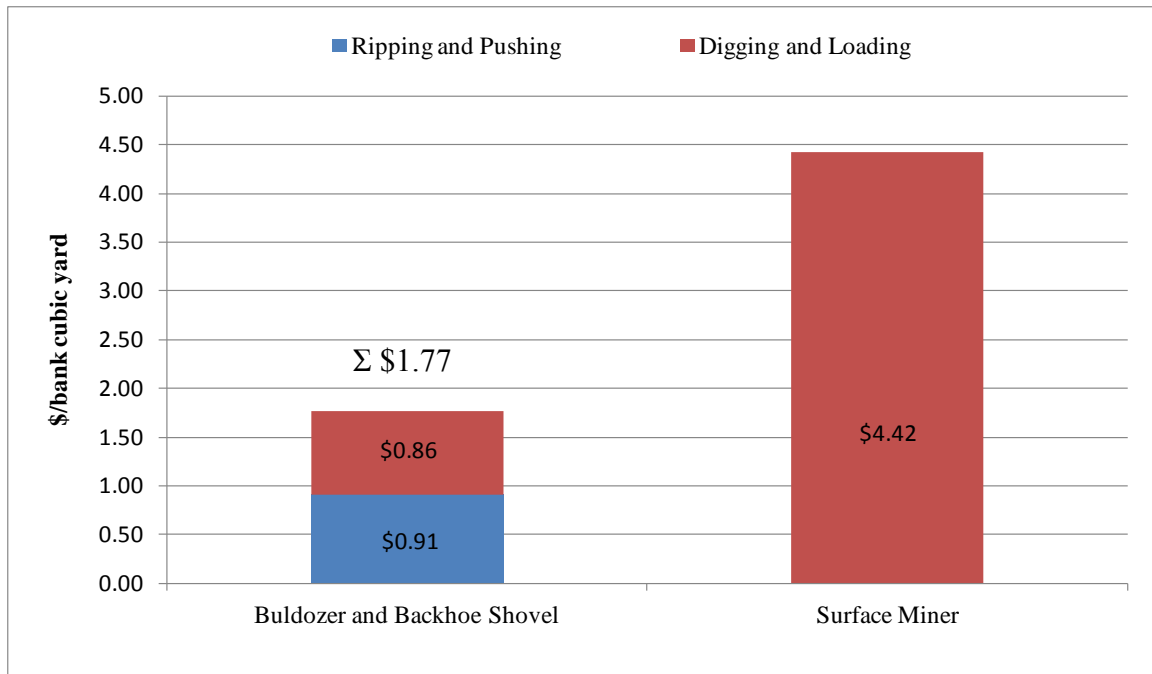


Figure 4-6 Conventional vs. SM Mining Method Comparison in Shale Interburden

Again, using the SM in a shale type interburden would be less economical than the conventional mining methods. The costs of the SM operating in shale interburden for various values of UCS and CAI are displayed in Table 4.31.

Table 4-31 Total Cost of SM per Cubic Yard in Shale Interburden

IB	Shale (MPa)					
	50	60	70	80	90	100
CAI=.6	\$0.97	\$1.12	\$1.35	\$1.68	\$2.81	\$3.70
CAI=1	\$1.18	\$1.43	\$1.78	\$2.29	\$3.06	\$4.26
CAI=1.5	\$1.93	\$2.49	\$3.89	\$4.92	\$6.66	\$8.39
CAI=2	\$4.14	\$5.24	\$6.35	\$7.98	\$9.51	\$12.19
Total Cost (\$/yd ³)			<\$1.00	\$1.00-\$2.00	\$2.00-\$3.00	>\$3.00

The interburden calculations provide an estimated comparison that could be expected if the given condition were encountered. While the SM is more expensive in each type of interburden, it is much more expensive in the stronger and more abrasive sandstone than in shale. If the UCS and CAI are found to be lower, the likelihood of using the SM in shale would be higher than in sandstone.

Figure 4.7 shows the cost of extraction of bituminous coal for two different scenarios: digging and loading with backhoe shovel and crushing (scenario #1); and SM (scenario #2). Here, scenario #2 yields the lowest cost of \$0.64 per ton, while the cost for scenario #1 is \$0.77 per ton of coal.

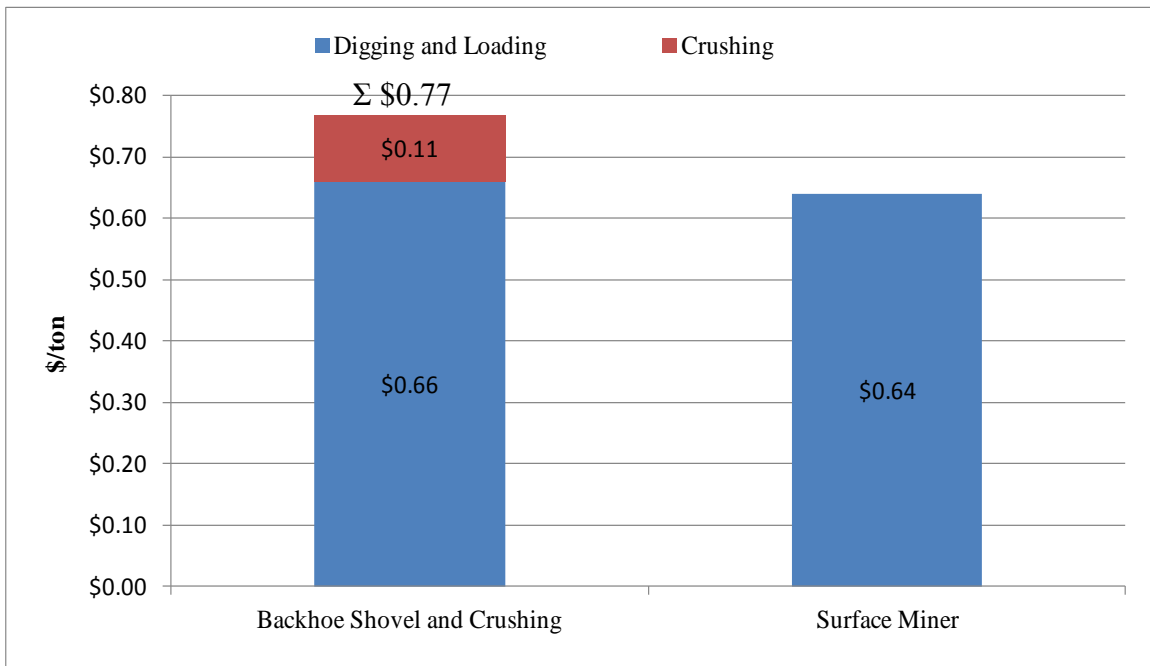


Figure 4-7 Conventional vs. SM Mining Method Comparison in Coal

Unlike the previous comparison, the SM was calculated to be the most economical option for the extraction of bituminous coal. As the SM is less expensive than the conventional extraction method, a crushing unit could be added to the SM mining method if necessary and still be the more economical mining option. As with the previous comparison the values for costs of the SM in varying coal properties have been calculated and displayed in Table 4.32. This table shows much more favorable results than the previous ones. If the coal properties vary in either direction, the cost of the SM will likely still be lower than the conventional mining methods.

Table 4-32 Total Cost of SM per Ton in Bituminous Coal

	Coal (MPa)				
	10	20	30	40	50
CAI=.6	\$0.55	\$0.61	\$0.67	\$0.74	\$1.23
CAI=1	\$0.57	\$0.64	\$0.72	\$0.85	\$1.33
CAI=1.5	\$0.59	\$0.70	\$0.84	\$1.45	\$1.68
CAI=2	\$0.62	\$0.82	\$1.11	\$1.88	\$2.44
CAI=3	\$0.73	\$1.15	\$2.13	\$2.85	\$4.32
Total Cost (\$/ton)	<\$1.00	\$1.00-\$2.00	\$2.00-\$3.00	>\$3.00	

The comparison of the conventional mining method and the SM mining method have shown that for the values tested in this research, the SM will only be more economical in the coal mining process. However, for overburden and interburden materials, the conventional mining methods are the more economical option.

4.2.5.1 Surface Miner Benefits

A benefit of introducing SMs to surface coal mining in WV would be an environmental one. If the conditions are favorable and the SM can be applied, most of the blasting processes that concern communities can potentially be eliminated. Specifically, SM usage can eliminate the generation of flyrock, toxic fumes, ground vibration, and surface vibration created by air-blast. While these issues are not frequent when blasting, the possibility still exists and this is a tremendous opportunity to address these environmental issues. In addition, the application of SMs enables clean cut edges and benches and very stable benched highwalls as opposed to blasted edges. The crushing process provided directly by the machine generates an even grade (96% < 6") of small particle sizes that can either be sold as secondary product (such as road base) or the material could also be used without further treatment for the reclamation process (Wirtgen, 2010).

The elevated grade control eliminates the need of at least the secondary crusher and possibly even (depending on the material) the need for a primary crusher. The SM creates a complete even and smooth surface, which leads to reduced damage to vehicles and tires. The mine operator has continuous control over drainage because the SM is able to cut a slope.

4.2.6 AHP Results and Analysis

The results of the cost and production analysis indicate that the SM would be applicable for the coal extraction phase. The cost comparison does not account for the negative or positive environmental factors created by either mining method. For this reason the use

of the AHP for optimal mining method selection adds a vital piece of information to the overall judgment by considering these additional factors. The selection of the optimal mining method was developed by the author of this thesis. Since the AHP is based on the user-defined input values, the results will be subjective to the user's preferences. To combat the subjective nature of the AHP, a sensitivity analysis of the results was included.

The values in the comparison matrices were determined based on the user's knowledge and point of view in this case the user is the author of this thesis. The level of importance of each issue/criteria can be decided on when all aspects of surface coal mining in WV have been considered. When comparing production to cost it was decided that production is 1.2 times more important than cost. When comparing cost to environmental issues it was decided that environmental issues were 1.4 times more important than cost. When comparing production to environmental issues it was decided that environmental issues were 1.5 times more important than production. Finally, when comparing ownership cost to operating cost it was decided that operating cost is 1.1 times more important than ownership cost, due to the variations in operating cost over the life of the operation. The scale used for the comparison matrices ranges from 1 to 9, as seen in Figure 3.4, with 9 representing an extremely important issue/criterion. The values decided on for these matrices represent small variations in the level of importance and still affect the outcome. These values are entirely biased to the user and will differ from user to user. To provide the most neutral outcome it is important to have multiple users

input values. The pairwise comparison matrices for the mining method selection can be seen in Table 4.33.

Table 4-33 Pairwise Comparison Judgment Matrices for the Selection of the Optimal Mining Method

Goal	Costs	Production	Environmental	Priority
Costs	1	0.8	0.7	0.275
Production	1.2	1	0.6	0.304
Environmental	1.4	1.5	1	0.420
				CR = 0.01
Cost	Ownership Cost	Operating Cost		Priority
Ownership Cost	1	0.9		0.476
Operating Cost	1.1	1		0.524
				CR = 0.0

The values given to each comparison can be seen in Table 4.33. The eigenvalues, or priority matrix, of the comparison matrices yields the priority weights for costs, production, environmental, ownership cost, and operating cost. These values are 0.275, 0.304, 0.420, 0.476, and 0.524, respectively. Finally, a Consistency Ratio (CR) is determined and for each comparison matrix. The CR is calculated to measure how consistent the judgments of the user have been when compared to large samples of random judgments (Coyle, 2004). They are considered acceptable because the CR is less than 0.10.

The next step in the AHP was to generate the global weights for each criterion involved in the AHP model. The global weight combines all local weights from each level to achieve an overall weight. These global weights can be seen in Table 4.34.

Table 4-34 Composite Priority Weights for Critical Factors

Strategic Issues	Local Weights	Criteria	Local Weights	Global Weights
Costs	0.275	Ownership Cost	0.476	0.131
		Operating Cost	0.524	0.144
Production	0.304	Production Rate	1.000	0.304
Environmental	0.420	Blasting Hazards	1.000	0.420

The local weights for each strategic issue and criteria are displayed in Table 4.34. Global weights account for all of the local weights and the capital cost, operating cost, production rate, and blasting hazards are 0.131, 0.144, 0.304, and 0.420, respectively. Blasting hazards has the highest weight, as seen in Table 4.34, due to the level of importance placed on the environmental issues in the comparison matrices. A ranking of the highest to lowest global weights is shown in Table 4.35.

Table 4-35 Ranking of Critical Factors

Rank	Criteria	Global Weights
1	Blasting Hazards	0.420
2	Production Rate	0.304
3	Operating Cost	0.144
4	Ownership Cost	0.131

The Global Weights (GW) show what criteria will have the most effect on the decision making process. Table 4.36 shows the ratings and scores applied to each of the criteria and the resulting adjusted global weight. The rating scores are as follows:

- Outstanding (O) – 0.513
- Good (G) – 0.261
- Average (A) – 0.129
- Fair (F) – 0.063
- Poor (P) – 0.034

Table 4-36 Application of the AHP Model to Mining Method Selection

Strategic Issues Criteria	Global Weights	Conventional Mining Method			Surface Miner Method		
		Rating	Score	xGW	Rating	Score	xGW
Cost							
Ownership Cost	0.131	A	0.129	0.0169	A	0.129	0.0169
Operating Cost	0.144	G	0.261	0.0377	P	0.034	0.0049
Production							
Production Rate	0.304	O	0.513	0.1561	A	0.129	0.0393
Environmental							
Blasting Hazards	0.420	P	0.034	0.0143	O	0.513	0.2156
Total Scores				0.2250			0.2767
Renormalized Scores				0.4485			0.5515

The results shown in Table 4.36 indicate that the SM mining method is the better option due to the high weight associated with the environmental strategic issue. Figure 4.8 shows the results of the AHP with the global weights and scores in Table 4.36 as percentages. In Figure 4.8 each mining method has a designated color, red for the SM mining method and blue for the conventional mining method. As each line crosses a strategic issue (cost, production, or environment) it represents the level of importance for that issues. These are simply used to visually assist in understanding the strengths and

weaknesses for each mining method. For example, the conventional mining method is more important in terms of cost and production as seen by the blue line being higher than the red in those two categories. The results of performing the AHP provide the global weights for each criterion and the final weight or priority of the alternatives.

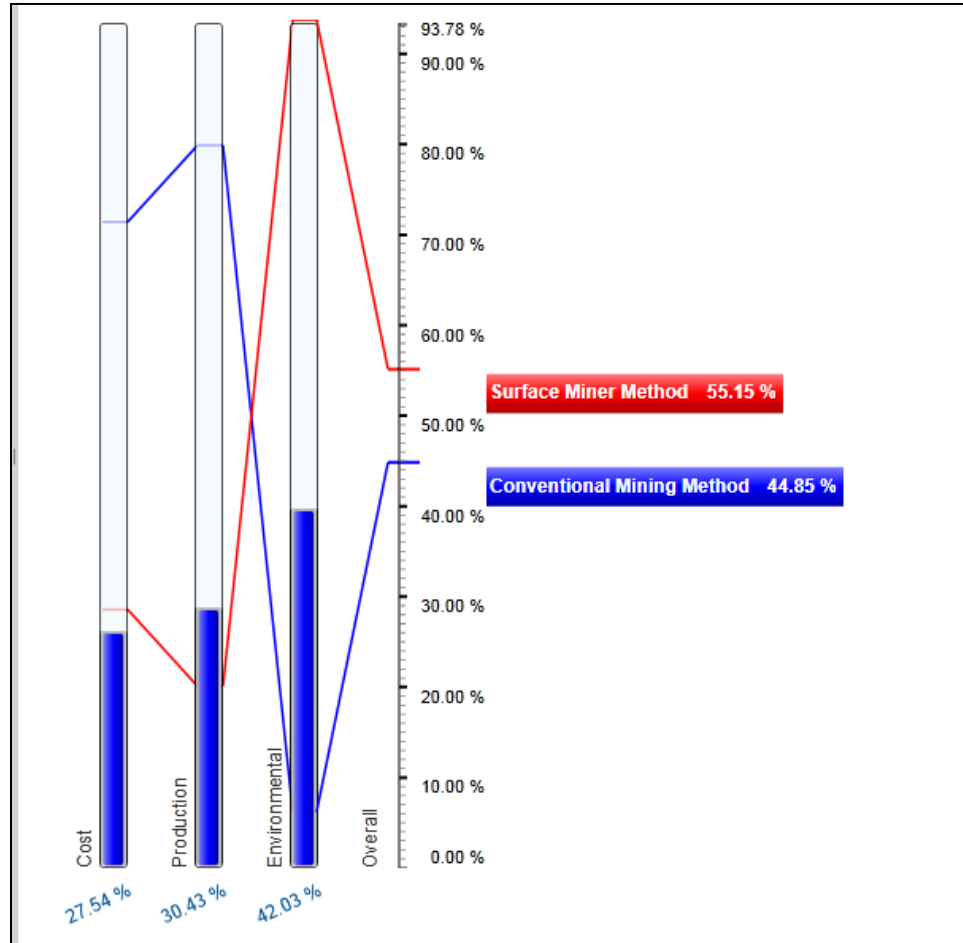


Figure 4-8 Performance Sensitivity Analysis

Figure 4.8 displays the global weights of each criterion for cost, production, and environmental as 27.54%, 30.43%, and 42.03%, respectively. After applying a rating score to each of the criteria the optimal mining method is determined by the highest

priority. In this case the SM mining method with 55.15% is the optimal method compared to 44.85% for the conventional mining method.

4.2.6.1 AHP Sensitivity Analysis

As the presented AHP results are obtained based on user-defined input values; this approach requires sensitivity analyses to be performed. For this purpose, the Expert Choice (2012) is used to adjust the criteria's priority, while simultaneously adjusting the remaining criteria relative to one other. Three separate sensitivity analyses were conducted; the first on the level of importance assigned to a criteria in the comparison matrix, the second on the final priority percentages of the criteria, and thirdly on the rating given to blasting hazards.

The first sensitivity analyses consider the level of importance given to a criterion when compared to another in a comparison matrix. An example of decreasing the level of importance by 10 percent is shown below.

- Production is 1.2 time more important than cost is decreased to 1.1 times more important than cost.

This is an analysis of the level of importance only and does not consider the ratings and scores. Tables 4.37 – 4.39 display the results of these analyses. The check mark displays the optimal mining method for each scenario. The optimal mining method remains the same in each case regardless of the importance level varying by 20 percent in both the

positive and negative direction. This analysis shows that variations of the criteria's importance do not have the largest effect on the outcome of the AHP.

Table 4-37 Production and Cost Sensitivity Analysis

Production and Cost Comparison Matrix					
Sensitivity Analysis	Production Importance	Conventional Mining Method		SM Mining Method	
-20%	1.0	<input type="checkbox"/>	44.29%	<input checked="" type="checkbox"/>	55.71%
-10%	1.1	<input type="checkbox"/>	44.58%	<input checked="" type="checkbox"/>	55.42%
0%	1.2	<input type="checkbox"/>	44.85%	<input checked="" type="checkbox"/>	55.15%
10%	1.3	<input type="checkbox"/>	45.11%	<input checked="" type="checkbox"/>	54.89%
20%	1.4	<input type="checkbox"/>	45.36%	<input checked="" type="checkbox"/>	54.64%

Table 4-38 Environmental and Cost Sensitivity Analysis

Environmental and Cost Comparison Matrix					
Sensitivity Analysis	Environmental Importance	Conventional Mining Method		SM Mining Method	
-20%	1.0	<input type="checkbox"/>	45.96%	<input checked="" type="checkbox"/>	54.04%
-10%	1.1	<input type="checkbox"/>	45.38%	<input checked="" type="checkbox"/>	54.62%
0%	1.2	<input type="checkbox"/>	44.85%	<input checked="" type="checkbox"/>	55.15%
10%	1.3	<input type="checkbox"/>	44.35%	<input checked="" type="checkbox"/>	55.65%
20%	1.4	<input type="checkbox"/>	43.88%	<input checked="" type="checkbox"/>	56.12%

Table 4-39 Environmental and Production Sensitivity Analysis

Environmental and Production Comparison Matrix					
Sensitivity Analysis	Environmental Importance	Conventional Mining Method		SM Mining Method	
-20%	1.0	<input type="checkbox"/>	46.34%	<input checked="" type="checkbox"/>	53.66%
-10%	1.1	<input type="checkbox"/>	45.57%	<input checked="" type="checkbox"/>	54.43%
0%	1.2	<input type="checkbox"/>	44.85%	<input checked="" type="checkbox"/>	55.15%
10%	1.3	<input type="checkbox"/>	44.17%	<input checked="" type="checkbox"/>	55.83%
20%	1.4	<input type="checkbox"/>	43.54%	<input checked="" type="checkbox"/>	56.46%

The second sensitivity analysis adjusts the criteria (cost, production, and environmental) in increments of approximately 10 percent in both the positive and negative directions. This is a sensitivity analysis of the criteria only and does not include variations in rating scores. The optimal mining method has the highest priority percentage and is displayed in red font. The results of these adjustments for the cost criteria can be seen in Table 4.40.

Table 4-40 Sensitivity Analysis on Final Criteria

Sensitivity Analysis	<u>Cost</u>				<u>Production</u>			<u>Environment</u>		
	<u>-10%</u>	<u>0</u>	<u>+10%</u>	<u>+20%</u>	<u>-10%</u>	<u>0</u>	<u>+10%</u>	<u>-10%</u>	<u>0</u>	<u>+10%</u>
Cost %	17.44	27.54	37.52	47.50	31.53	27.54	23.63	32.24	27.54	22.76
Production %	34.67	30.43	26.24	22.05	20.36	30.43	40.32	35.62	30.43	25.15
Environment %	47.88	42.03	36.24	30.45	48.11	42.03	36.05	32.14	42.03	52.10
Surface Miner Method %	56.79	54.67	52.30	49.60	61.33	54.67	48.63	47.29	54.67	61.95
Conventional Mining Method %	43.21	45.33	47.70	50.40	38.67	45.33	51.37	52.71	45.33	38.05

Table 4.40 displays the global weights of each criterion for cost, production, and environmental as well as the optimal mining method (displayed in red font). The optimal mining method selection continues to be the SM mining method until the of cost priority

percentage is increased by 20%. This result was expected as the SM has a lower score for costs. A similar trend exists when adjusting the production priority percentage. The optimal mining method remains the SM mining method until the production priority is increased by 10%. This was expected as the SM has a lower score for production and will affect the optimal method at some point. Conversely, as the environmental priority percentage decreases (-10%) the optimal mining method becomes the conventional mining method. This was expected as the SM has a higher score for environmental issues and at a point in lowering its priority the optimal mining method will change.

As mentioned previously this sensitivity analysis does not account for variations in rating score input values. Since blasting hazards is the highest weighted criteria, a sensitivity analysis of the rating score was performed on it. In this sensitivity analysis the global weights of the three strategic issues will remain unchanged, while the influence of the rating scores will vary and impact the decision accordingly. Figure 4.9 shows that the conventional mining method score for blasting hazards rise to “fair” with all remaining scores unchanged.

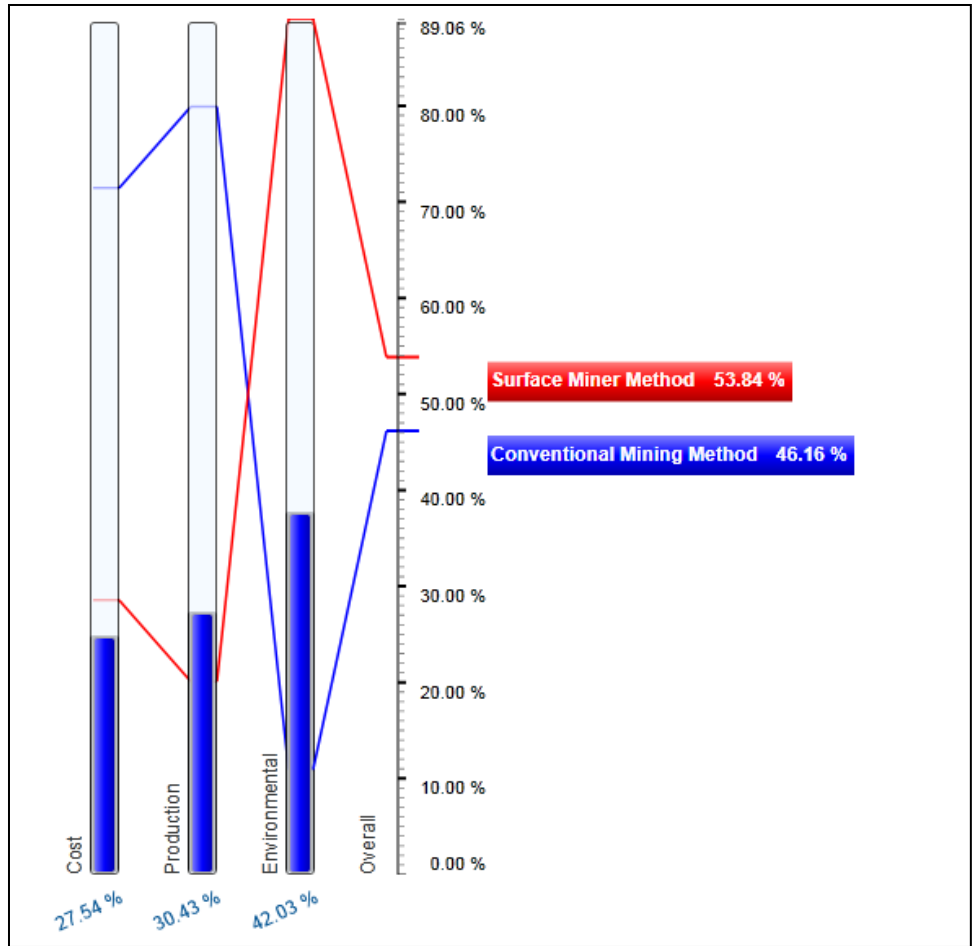


Figure 4-9 Sensitivity Analysis (Conventional Blasting Rating: Fair)

Raising the blasting hazards score for the conventional mining method to “fair” will not affect the selection of the optimal mining method as the priority of the SM mining method is 53.84% compared to 46.16% for the conventional mining method.

Figure 4.10 shows the SM mining method score for blasting hazards dropping to “good” with all remaining scores unchanged.

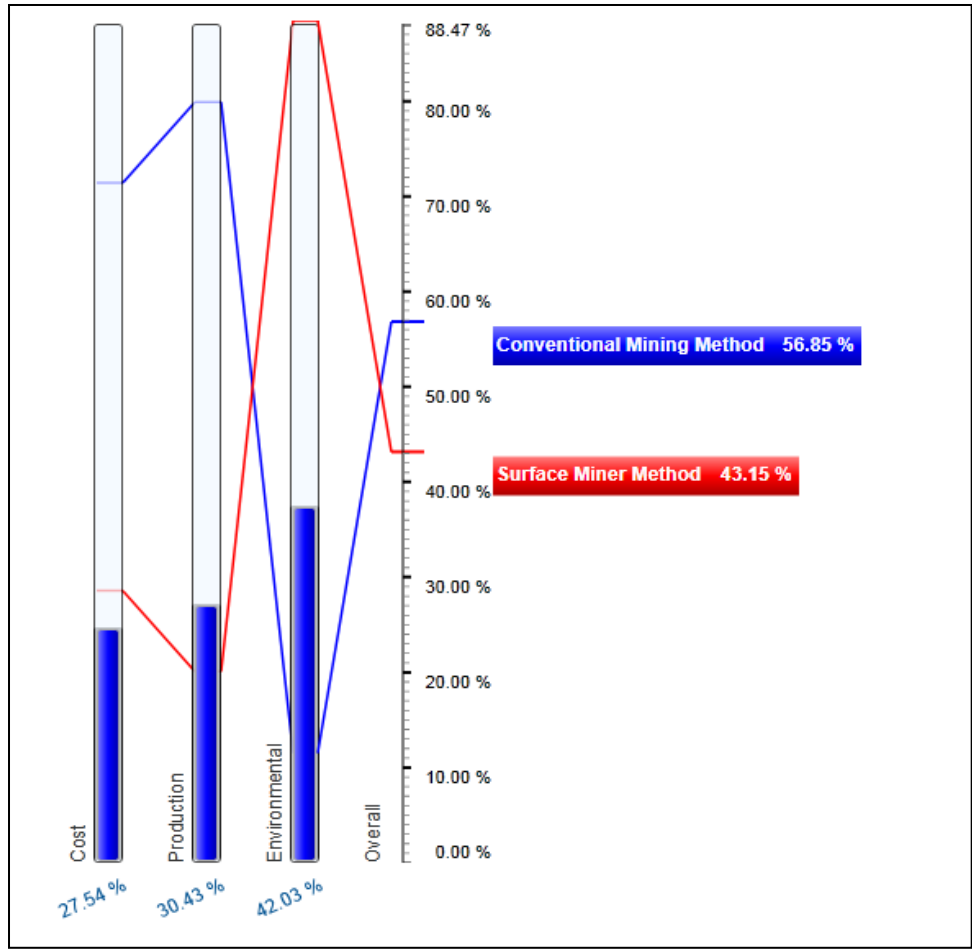
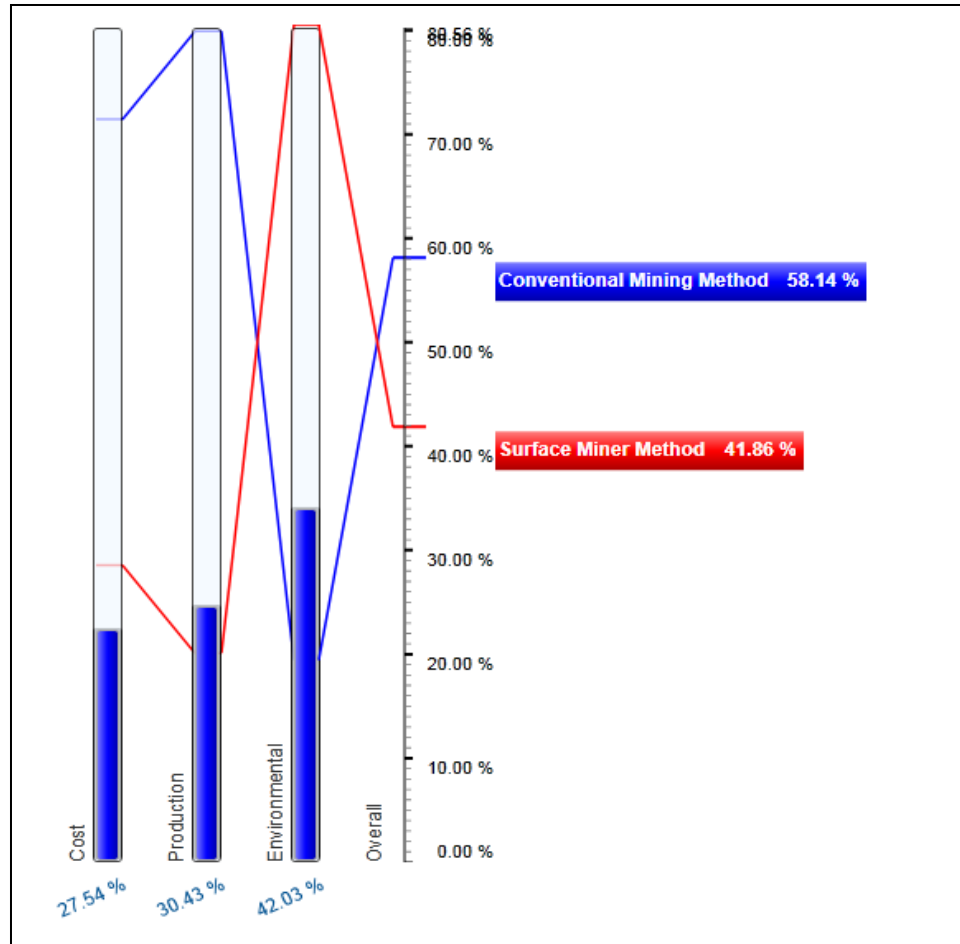


Figure 4-10 Sensitivity Analysis (SM Blasting Rating: Good)

Lowering the blasting hazards score for the SM mining method to “good” will affect the selection enough to change the optimal mining method to the conventional mining method with a priority of 56.85% compared to 43.15% for the SM mining method.

Finally, Figure 4.11 shows the conventional mining method rising to “fair” and the SM mining method dropping to “good” with all remaining scores unchanged.



**Figure 4-11 Sensitivity Analysis
(Conventional Blasting Rating: Fair, SM Blasting Rating: Good)**

Lowering the blasting hazards score for the SM mining method to “good” and raising the conventional to “fair” will affect the selection to the conventional mining method with a priority of 58.14% compared to 41.86% for the SM mining method.

The results of the sensitivity analysis for the blasting hazards score are to be expected as it is the only rating where the SM mining method is superior. By performing these sensitivity analyses, it can be seen how dependent the selection of the optimal mining

method is on environmental factors. The combination of the results produced by the cost and production analysis and the application of the AHP, allows for an informed decision on the optimal mining method to be made.

Chapter 5

Conclusions and Recommendations for Future Research

5.1 Conclusions

Based on the results of the study it can be concluded that the application of the SM would benefit surface coal mining operations in WV. These benefits include: lower costs of extraction of bituminous coal when compared with conventional mining unit operations; improved selective mining where thin coal seams, overburden, and interburden can be mined separately, thus preventing dilution of coal; generates an even grade of coal which reduces costs of further processing; and a more environmentally friendly mining method.

The downside of using the SM in surface mining operations begins with the limited quantity of the overburden and interburden material being extracted. An application problem arises with higher values of unconfined compressive strength and abrasivity of overburden and interburden material. Picks on the cutting drum will wear out faster and require more changes. At a certain point this method becomes unfeasible due to the slower advance rate and cost involved in a high amount of pick changes. New pick changing tools have accelerated this process, but it still remains one of the main factors of the operating cost. The negative aspects of using the SM are: a lower production rate when compared with conventional mining units; a high operating cost or inability to work in rocks with a very high value of unconfined compressive strength of overburden/interburden material.

As governmental regulations become more stringent and public pressure increases, future coal mining operations will need to achieve “low impact” mining. Incorporating new design features and practices that can substantially reduce negative impacts is one of many steps that will need to be taken. New mining technologies and systems can also minimize environmental disturbances while improving coal recovery. The SM is one such technology that can eliminate blasting, which is one of the major public concerns. These types of changes are necessary to ensure that the coal industry can continue to operate safely and economically in the changing atmosphere.

5.3 Recommendations for Future Research

Some of the limitations in this research include the lack of data from any specific mine in WV; absence of the haulage considerations (specifically haul trucks) in the model; and the absence of dragline consideration as a digging and disposal mining unit. These three limitations should be included and addressed in future research.

To confirm the results of this research, it is necessary to test a SM in one of the existing surface coal mines in WV. Data collected from the testing should be focused on the engine load factor, the amount of pick wear, cutting depth, and cutting velocity in sandstone and shale overburden and interburden. This would greatly help determine the fuel consumption and maintenance costs involved with the SM. The formulas used for calculation of pick wear would be confirmed or determined to need revision given the data generated from an experimental trial in a material with the known properties.

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