NATIONAL ENERGY TECHNOLOGY LABORATORY

Assessing Future Supply Curves for Coal In Light Of Economic, Technological and Environmental Uncertainties

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Table of Contents

List of Figures

List of Tables

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Executive Summary

Current coal mining cost forecasting extrapolates historic mine cost statistical data. This practice would be fine if mining conditions remained the same in the future. However, these costs were incurred in shallower and thicker seams the resources that remain to be mined. To estimate future mine production and costs, a model was built in Analytica, and is representative of the breadth of geological conditions and equipment configurations found throughout U.S. coal mines. The model simulates a range of production and costs based on typical unit operations performance and capital, labor, and operators cost, taxes and fees found throughout the industry. The mine and its working areas are sized according to typical industry practice. Underground mining technologies represented in the model are longwall and continuous room-and-pillar mines, and surface shovel and truck operations comprise the model's surface mine simulation. The model estimates the average cost of operating the mine, over its lifetime. Starting costs are high, due to capital expenses and permitting costs, and end of lifetime costs are lower as these costs are paid off; it is assumed that the average lifetime cost is an appropriate indicator of mining costs.

To validate the model real mine production rates and cost were simulated. The validation was restricted to mines for which geologic and production data were available. Seventeen longwall mines, fourteen continuous mines and ten surfaces mines were reconstructed and simulated by the model. The model's estimated 5^{th} – 95^{th} percentile production and cost ranges are compared to the mine's historical production and price data. It is assumed that the coal market is close to equilibrium. Therefore, coal price is comparable to mining costs. The model's simulated production rate and costs capture most of the actual output and price. Model results are dependent on data uncertainty. The size of the estimated range reflects the availability and quality of data. The model estimated the tightest range of production rates for mine types that had discretely reported geological characteristics. Using the 50^{th} percentile estimate as a point of comparison, the model estimated the highest production rates for surface mines and longwall miens. The 50^{th} percentile production rates for surface, longwall, and continuous miens were 1.5 – 8.2 million tons, 3.6 – 16.1 million tons, and 1.2 – 1.9 million tons, respectively. The model estimated the highest costs for continuous mining, $33-46/ton. Longwall and surface mines estimated costs were $13 – 41/ton and $19 – 40/ton, respectively. The model's output estimated the validation sample mines' production rate and cost within 5 – 11 percent.

To determine the cost and availability of U.S. coal resources to meet future demand, the cost model was applied to U.S. coal regions defined by the USGS National Coal Resource Assessment (NCRA), which comprise the nation's demonstrated reserve base (DRB). It is not a complete resource assessment because geological data for all U.S. coal regions is not available. Furthermore, the data uncertainty is not clearly defined. However, as the NCRA is the most definitive coal seam geology dataset available, it is used. The full available coal dataset is analyzed. A subset of the data is also evaluated, to determine the resource cost change if the maximum seam depth decreases. This subset restricts the resource to shallow coal. The first resource scenario defines resources per the USGS definition of the DRB, which is accepted and followed by the NAS in their recent coal resource report. The full seam thickness is considered, but overburden depth is restricted to the DRB maximum allowable depth of 1,000 feet. The second resource scenario narrows the DRB to shallower depths, where the resource is more concentrated; mining to a maximum 1,000 feet might not be necessary if it yields only a marginal amount of additional coal. Full seam thickness is still considered in the second scenario. The first coal resource scenario results in a larger defined coal resource, while the latter is more restricted.

The coalbeds included, and assessed in this study, in the NCRA are the Colorado Plateau, Rocky Mountains and Great Plains, Northern and Central Appalachia, Illinois, and Gulf Coast coal basins.

Coal availability and cost are assessed to meet 100 years of business as usual demand based on historic EIA demand data. Based on the estimated mine production rate and costs per NCRA region, an average cost curve is determined. The average cost is the average cost over the lifetime of the mine and for a "typical" mine in the region. Ordinarily, it could be assumed that lowest cost resources in the region are mined first. However, because the resource characteristics are simplified, and the average depth and seam thickness are input to the model, average production costs and rates are output.

Restricting the coal resource affects the estimated mining cost and amount of coal available to meet demand. The average cost curves are generated by scheduling the coal regions by least cost. The lowest cost coal is in the Colorado Plateau and Rocky Mountains and Great Plains regions, followed by the Northern and Central Appalachian and Illinois Basins. When the full DRB is assessed, maximum costs could read \$200/ton but 50th percentile costs never exceed \$22/ton. For the limited DRB assessment scenario, it never exceeds \$33/ton.

The total projected coal demand over the 100 year time period is 157 billion short tons. This demand is met with 175 million short tons of the total DRB and 120 billion short tons of the examined DRB subset remaining. At the rate of coal demand, coal resource will be exhausted in less than 250 years. Cost to mine the remaining coal will be more expensive. To estimate the mining cost for the remaining resource, more geological data is needed. Furthermore, it is necessary to measure all coal resources in all regions, not just those that are currently mined, in order to gain insight into the reliability of coal or provide energy for the long term.

To add environmental impact metrics and costs to the assessment, the model's process simulation capabilities were used to estimate the size of mine working areas, volumes, and surface area and thus determine environmental impacts. Coal mining releases air and water pollutants, affects soil quality, ground stability and water availability. The environmental evaluation estimates total subsidence for underground mines, total pit area for surface mines, acid generation potential for all mines, water consumption rates, solid waste generated, and methane emissions. These impacts are valued according to their economic value as well as the cost to prevent damage. Land values are estimated according to USDA land values as well as ecosystem values in the general literature and methane according to the pollutant trading market. When valued on this basis, environmental impact is small relative to the total output of a given mine such that the environmental cost per ton of coal is very low. Land impacts on a per ton basis are generally less than \$1/ton, and methane emissions cost per ton is less than \$0.10/ton.

Technologies to prevent environmental impacts include backfill and groutfill for subsidence, sealants for acid mine drainage, revegetation for surface mined lands, robotic underground mining to avoid mountain top removal and valley fill, and coal seam methane development to avoid methane emissions.

These technologies, and results, are summarized as follows:

Mine sealant costs to avoid acid mine drainage are based on the cost to cover a surface mine pit with a landfill liner before filling it, or grouting or sealing a surface or underground mine. The costs ranged from $1-$49/ton for longwall mines, and were highest in the Rocky mountains and Great Plains, and $0 - $20/ton for continuous mines, also highest in the Rocky Mountains and Great Plains. The cost to line a surface mine with a landfill liner ranged from $2-$6000/ton, with the highest costs occurring in the Colorado Plateau. Sealant costs for surface mine pits was cheaper, ranging from $0-$365/ton.

Backfill was examined on a partial and complete fill basis for underground mines, assuming a 10:1 mixture of cement and fly ash or rockfill. Backfill with the cement mixture was more expensive than rockfill, and results in a lot of CO_2 emissions. Longwalls require more fill than continuous mines, and so have higher backfill costs. The cost to partially fill a mine was $1 - $108/ton for a longwall mine, and less than $1/ton for a continuous mine, if using the cement mixture. To completely fill a mine with the cement mixture, it would cost $94 - $1260/ton for longwall and $47-$65/ton for continuous mines. It is not significantly cheaper to fill by rockfill, but the avoided CO_2 emissions from cement production may make this option worthwhile.

Revegetation and reforestation costs were examined as a means to treat surface mine land degradation. Based on current OSM and PA DEP guidance, it was determined that the cost to revegetate and reforest affected mine land would cost less than $1/ton.

Robotic mining as a method to avoid surface mining, and to allow safe mining under uncertain conditions such as those found in Appalachia was examined. Based on current unmanned vehicle costs to the U.S. Army, and U.S. Army tank guiding system costs, the capital costs for longwall shearer and continuous miner units were adjusted to reflect the cost of autonomous mining vehicles. Adjusting production output to reflect the Australian autonomous miner experience of 30% productivity increase, the estimated mining costs decreased by about 10%.

Coalbed methane capture by four potential options, described by EPA, were examined. These options are use of gob wells during mining, premining vertical wells, a combination of premining vertical wells and gob wells, and using vertical wells, gob wells, and horizontal boreholes in combination. The result of this evaluation was that the first option would add an additional $10-$30/ton to underground mine costs, the second option would add $6-$24/ton to underground mine costs and $8 - $700/ton to surface mine costs depending on what region the mine is located. The third option results in additional costs of $19-47/ton for underground mines, and the fourth option costs an additional $20 - $50/ton for underground mines.

This study provides an analysis of the costs to continue mining in the U.S., as well as discussion of supply availability and environmental costs. To better understand these costs, it is imperative to better understand unit operations timing, productivity and costs, obtain more detailed and thorough measurements of coal reserves in all coal basins regardless of whether it is producing. It is also of importance to better develop a better understanding of coal mining's interaction with the surrounding ecosystem. Analyses of metal mining and heavy construction site impacts on the environment have been performed, and technologies developed to address them; to better understand and mitigate coal mining's environmental effects it is necessary to examine and develop these indicators for specific sites and regions.

Baseline Model and Validation

Baseline Model

Current coal mining cost forecasting extrapolates historic mine cost statistical data. This practice assumes that mining conditions will remain the same in the future. However, historical extraction costs are not indicative of future mining costs. Easily accessible deposits are extracted first, leaving less desirable coal seams for the future. If current and future mining cost estimates are based upon thick and shallow seams, rather than thin and deep seams, future coal supply costs will be underestimated. Work by the Energy Information Administration used CoalVal, a financial mine cost model developed by the United States Geological Survey, to examine the cost to mine resources [1]. The CoalVal [2] is a prescriptive model that does not consider the coal seam characteristics; the user must input specific equipment configurations to evaluate the cost of the mine system. A better way to estimate costs would account for geological characteristics of the resource, and operations chosen to extract it.

This chapter describes a probabilistic model of mining processes and costs for U.S. continuous, longwall, and surface mining operations. The purpose of this paper is to describe a process-based cost model as a suitable means to estimate production rates and costs for U.S. mines. The model described in this paper estimates a range of cost associated with the range of predefined possible operations based on user input coal seam information.

Method to Estimate Production and Cost Ranges

Mining conditions vary nationwide, due to specific geological conditions on site, and operational practices. Rather than assessing production and cost associated with a specific equipment configuration or practices adjusted for challenging conditions, the model predicts a range of estimates for a range of equipment configurations. The output accounts for the range of equipment, configurations, overburden composition, and seam thickness variation.

To create a model that represents the inherent uncertainty related to a wide array of mining practices, a model was built in Analytica – a stochastic modeling tool for estimating a range of potential outcomes. The components of the model, such as the timing and capacity of machinery, capital costs, and tax rates, are input as ranges to reflect mine operation and data uncertainty. The input range bounds are related to the output range bounds. The top end of the range represents the 95th percentile, or highest possible value. The bottom end of the range represents the 5th percentile, or lowest possible value. The model results are 5th – 95th percentile estimates range, which represents the widest range of possibilities. It shows the range in production and cost resulting from all possible equipment sizes, timing and configuration for a mine system.

U.S. Coal Characteristics

This model estimates production rates and costs for U.S. bituminous coal, which has accounted for over 50 percent of annual U.S. coal production since records have been kept [3]. Coal density is 1705 – 1846 tons/acre-ft [4]. Overburden contains sandstone, clay, gravel, shale, and various other materials. An overburden density range that accounts for all these possibilities is 1900 – 3190 tons/acre-ft with a swell factor of 1.25 – 1.6 [4].

Coal mining cost and production model overview

The model simulates mines, sizing them according to coal resource size. A schematic of the resource's simplified dimensions, as model input, is shown in Figure 1. Overburden depth, seam thickness, interburden depth, and resource width and length are inputs into the model. The model estimates production and costs in a single seam for underground mines, and up to ten seams for surface mines.

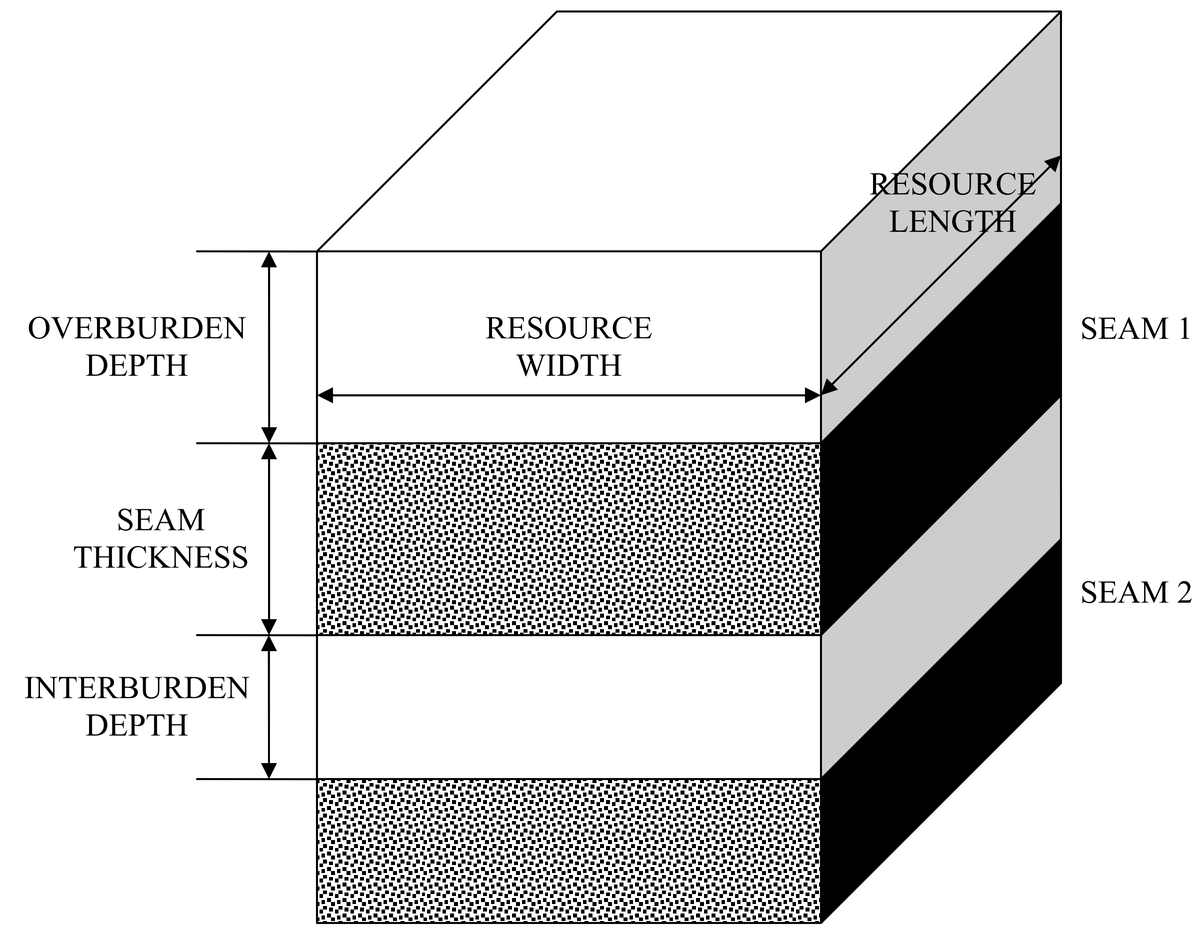

Figure 1. Simplified coal resource dimensions

The model schedules unit operations based on estimated sizes for surface mining pits, continuous mine rooms and pillars, and longwall panel and development sections. Equipment is sized according to the mine design literature [4-7]. Based on estimated production rates, it sizes a

Level III or IV preparation plant for the simulated mine [8]. It calculates U.S. federal taxes and regulatory fees; all equipment cost estimates are based on reported U.S. mine cost data [4, 9]. Furthermore, the model uses U.S. based equipment timing study data [4, 10-14] to configure unit operations and estimate production rate.

Mine system simulation

Surface mining system simulation

Surface mining is a series of processes of breaking and moving material. The model simulates a hydraulic shovel and truck operation. First, holes are drilled into the overburden, and explosives dropped into the holes to break up the overburden. The crumbled overburden is then excavated to expose the coal. Next, the coal is broken up by hydraulic excavators and removed by truck. Overburden from the pits is placed into surface storage or impoundments. The amount of material – overburden or coal – is dependent on pit size.

The model includes overburden removal steps in the surface mine simulation. After overburden is drilled, broken up with ANFO, and removed by shovel and truck, the coal is mined by shovel and truck. The model assumes 1 – 7 surface mining teams comprised of 1 – 2 excavating shovels or bulldozers, 2 – 5 trucks varying from 125 – 240 tons, a grader and drill. Drilling, blasting, shovel time, and road length algorithms are based on the industry standard and rules of thumb [4]. The model is capable of modeling up to 10 layers of coal and interburden. The text below describes operations within a single layer of coal, but if a mine is to access several layers this method is applied to each layer of coal in order to determine the total production rate.

Surface pit sizing, estimating coal and overburden volume

Surface mine pit sizing is based on the dimensions of the excavation equipment. In order to size a pit, the width and length must be ascertained. It is assumed that, at minimum, the pit must fit the base of a hydraulic excavator. The maximum pit width is assumed to be 150 ft [4]. A range of cutting radii, crawler widths, cleaning radii and excavator capacities were collected from manufacturer literature [15-19], and assumed to be 15.92 – 25.42 ft, 15.75 – 24.25 ft, 21.42 – 32.17 ft and 19 – 56 yd^3, respectively. The pit width range is assumed to be a uniform distribution between the minimum and maximum pit widths, and is determined according to equations (1 – 2).

$$PW_{\min} = \min(r_{cleaning}, r_{cutting}) + \frac{CW}{2} \quad (1)$$

$$PW = Uniform(PW_{\min}, 45.72) \quad (2)$$

where:
PW_{min} = minimum pit width
$r_{cleaning}$ = hydraulic excavator cleaning radius
$r_{cutting}$ = hydraulic excavator cutting radius
CW = crawler width

PW = pit width

The pit length is estimated in a similar fashion to pit length. It is assumed that the minimum pit length must accommodate the maximum size hydraulic excavator, and that the maximum pit length is equal to the length of the coal resource:

$$PL_{\min} = \max(r_{cleaning}, r_{cutting}) + \frac{CW}{2} \qquad (3)$$

$$PL = Uniform(PL_{\min}, L) \qquad (4)$$

where:
PL_{min} = minimum pit length
PL = pit length
L = length of coal resource

Pit area is estimated as the product of pit length and width. The volumes of overburden overlying the pit, and the coal contained in the pit are determined according to the user input overburden depth and seam thickness.

Coal is not completely extracted during surface coal mining. Excavator shovels are not fine tuned machines, and cannot precisely cut overburden and coal separately. A small amount coal is often cut with the last layer of overburden and lost in the waste pile. Additionally, a thin layer of coal is left in the pit before it is filled. It is too expensive to separate this thin layer of coal from the underlying material that would be extracted if the shovel were to dig it out, so it is left behind. To account for the lost coal, it is assumed that a total 2 – 10 inches of coal is lost in this manner, per pit. The amount of coal mined is equal to the original amount available in the pit, less this lost coal.

Estimating ANFO needs

The overburden is broken up by anhydrous fuel oil (ANFO). The spacing of drill holes, powder factor, and the amount of ANFO used is calculated by following the methods in the standard literature [4]. The amount of ANFO needed is based on the expected lifetime of the mine, and area to be cleared. 50^{th} percentile charge weight is 1053 lb according to methods in the literature [4], and assuming an industry standard drill length of 25 – 65 feet [9] and ANFO standard gravity of 0.75 – 0.95. The resulting powder factor estimate is 0.2 lb/yd^3 with 5^{th} and 95^{th} percentiles of 0.04 lb/yd^3 and 0.8 lb/yd^3, respectively. The estimated amount of ANFO to clear the mining area is calculated as per Equation (5):

$$ANFO = OB_V \times PF \qquad (5)$$

where:
ANFO = weight of ANFO required
OB_V = volume of overburden overlying coal resource to be mined
PF = powder factor

Overburden and coal cutting and loading time

The time needed to remove overburden and coal is the total drilling time, ANFO placement and explosion time, and overburden and coal excavating time. The time needed to haul the coal out of the pit is discussed below. The volumetric drill rate to insert ANFO into overburden is 750 – 3800 ft^3/minute [9], and borehole timing is 11 – 17 ms/ft [4]. The ANFO insertion and explosion time is calculated based on the number of boreholes previously calculated.

Using the previously mentioned overburden and coal swell factors, the volume of broken material is calculated. The rate to load the material into trucks to be removed from the pit is determined according to shovel rates and capacity. Shovel cycle time and capacity are estimated according to ranges provided in the general literature. The shovel cycle time is assumed to be 20 – 44 s, and is divided by a correction factor of 1 – 1.25 in the case that mining is undertaken in less than optimum conditions [4]. The excavator capacity is assumed to be 19 – 56 yd^3 [9], with an efficiency of 0.54 – 0.83 [4].

Surface mine road design and travel time estimation

Assuming a varying truck size of 125 – 240 tons, the number of truckloads needed to remove waste material and coal from the pit is determined. It is assumed that each truckload requires a single round trip to deliver the coal or waste material to an onsite collection area. Road distances in and out of pits are estimated so that hauling times can be calculated. It is necessary to know hauling time because the production rate is dependent upon the travel time for trucks in and out of the pit. In order to organize the pits for road designs, the model groups them into "pit regions" that are 1.5 mile by 3.75 mile, based on analysis of typical surface coal mine layout to be mined over a period of 20 years [11]. Although the model considers mine lifetimes that range between 10 and 30 years, assuming a typical surface coal mine layout designed for a 20 year lifetime is a best approximation at this time.

To estimate the road distance in and out of a pit, it is assumed that roads will be designed with an 8 percent grade, for greatest safety [4]. Using the pit width and length, the distance for a zig-zag or spiral road can be determined. The model chooses the shortest path. Assuming again, maximum safety, the truck traveling speed in and out of the pit is assumed to be 15 – 30 mph [4]. Truck dumping time is assumed to be 50 s [10]. It is assumed that travel time and dumping time is the same for waste materials, or overburden, and coal.

Estimating surface mine production rate

As described above, the model calculates the total production time needed to mine the pit by breaking up overburden with ANFO, and extracting the overburden and coal. Knowing the original amount of coal available in the resource, and the number of model defined pits that can be accommodated, the production rate (coal/year) is estimated by dividing it by the production time for the 1 – 7 surface mining teams used to extract coal.

Continuous mine system simulation

Continuous mining refers to a mine practice that uses several unit operations to cut, load, and remove coal from an underground mine. It is also called room and pillar mining because rooms of coal are extracted while pillars are left to support the overburden, or roof. It consists of cutting the coal with a continuous miner, backing the continuous miner out and bolting the roof with a roof bolter, then removing the roof bolter and bringing a shuttle car in to be filled with coal. This shuttle car then trams the coal to a central pick up point for transport to the surface. All the while, electricity, water, and ventilation systems must be steadily expanded and maintained in order to support the mine and miner's operations underground.

The model assumes that there is a uniform distribution of 2-4 continuous mining teams. Each team is comprised of a continuous miner, 3-6 shuttle cars and a roof bolter.

Room and pillar sizing

The model assumes that a continuous mine has at least three entries. The pillar width is determined as a function of overburden depth, such that the amount of coal contained in the pillars increases with depth. Equations (6 – 8) are developed from direct observations of underground mine pillar widths in West Virginia at 6 – 8 ft [20]:

$$W_{2.4} = 0.36 \times \left(OB_D\right)^{0.7} \tag{6}$$

$$W_{2.1} = 0.38 \times \left(OB_D\right)^{0.5} \tag{7}$$

$$W_{1.8} = 0.406 \times \left(OB_D\right)^{0.8} \tag{8}$$

where:
$W_{2.4}$ = pillar width for a seam with maximum thickness of 2.4 m
OB_D = overburden depth
$W_{2.1}$ = pillar width for a seam with maximum thickness of 2.1 m
$W_{1.8}$ = pillar width for a seam with maximum thickness of 1.8 m

It is assumed that these pillars are square, such that the length is equal to the width, and height equal to seam thickness. For continuous mines in a large coal resource, it is assumed that entry length is never more than 10,000 – 13,000 feet, which is the longest achievable length of a longwall panel [21]. It is assumed that continuous mine workings will not exceed this length because if it is not economical for longwall mining, a higher yield method, to sustain lengthier working areas then it certainly will not be affordable for a continuous mine. If the length of the coal resource is less than 10,000 feet, then the entry length of the mine is equal to the length of the resource. Based on these assumptions of mine length, pillar widths, and assuming entry width of 20 feet for minimum safety requirements, the number of rooms and pillars within the resource is estimated. The starting amount of coal for a continuous mine is estimated based on the maximum entry length, coal resource width, and seam thickness. The coal mined is estimated to be the original amount of coal in a continuous mine section less the amount of coal left in the pillars.

Continuous mine coal cutting, loading, and tramming time

After the amount of coal produced by the mine is estimated, the number of cuts and loads to extract the coal can be determined. It is assumed that the continuous miner has a cutting depth of 20 – 30 feet and cutting width of 20 feet based on published machine sizes [9]. The amount of coal that is broken per continuous miner cut is determined:

$$T_{CM} = CM_D \times Th \times CM_W \times \rho_B \qquad (9)$$

where:
T_{CM} = tons of coal cut by the continuous miner
CM_D = continuous miner cutting depth
Th = seam thickness
CM_W = continuous miner cutting width
ρ_B = bituminous coal density

Assuming a shuttle car hauling capacity that ranges from 8.5 – 17 yd^3, on average 11 shuttle car loads are needed to haul the cut coal. Those who are familiar with continuous mining may note that roof bolting has not been mentioned yet. The amount of roof bolting time needed is negligible [13], and the model's continuous mine system timing sequence accounts only for the continuous miner and shuttle cars.

Shuttle car timing is variable and is derived from published shuttle car length 30 feet [9], and timing studies data. The timing studies examined include methods to estimate total cut cycle time, coal hauling distance, which define tramming distance, based on recorded underground vehicle speed, loading rate, time to switch the continuous miner in and out of the mined room with the shuttle car, waiting delays, dump time, and in room cutting delays [14].

Estimating continuous mine production rate

Production rate is estimated by dividing the amount of coal mined by the total production time, for a total of 2 - 4 mining teams. As described above, the amount of coal produced is the starting amount of coal in the mine less the coal in the pillars. The total production time is the time needed to load, changeout the continuous miner and shuttle car, wait on a car if necessary, as well as delays for advance activities. Advance activities include installing ventilation, water and electrical systems to support miners and equipment.

Longwall mine system simulation

The model simulates a longwall mine with a minimum of one longwall panel and two continuous mining development sections and barrier pillars. It is assumed that 1 – 2 longwalls operate in a longwall mine. Altogether, the equipment configuration per longwall within the mine is assumed to be a longwall, 2 – 3 continuous mining teams as described above, a face conveyor and stage loader, longwall shields, a belt conveyor, and 4 – 8 shuttle cars (in addition to the shuttle cars devoted to the continuous mining teams in the development sections.)

The continuous mining teams mine the development sections. Two parallel development sections must be completed in order to support a longwall. It is assumed that when the longwall panel begins operation, additional development sections may begin in order to support future longwall panels. These development sections are mined in the same manner as a continuous mine, except that the pillar width and length are always 82' and 160', respectively, at any depth [20]. The coal extracted in the development sections is transported within the mine by shuttle cars, as it is in the previously described continuous mine system. Coal mined by the longwall shearer is collected and moved by the face conveyor and stage loader to a belt conveyor. It is assumed that the longwall cutting, loading, and transporting system is fully automated.

Longwall sizing

The average underground longwall panel dimensions are based on the current size reported by industry. The average face width is 939.2 feet [22] and entry width is 100 – 350 feet and barrier pillar width of 200 – 500 feet [4]. The maximum panel length is assumed to be that which is the maximum technically possible, 10,000 – 13,000 feet [21]. Development sections are assumed to have a maximum of 3 entries, with pillar widths determined in the same manner as for the simulated continuous mine system described above.

The number of panels that will fit within a coal resource are determined by the combined width of the development sections and panels. The width of the coal resource is divided by the estimated width of a panel with two development sections in order to ascertain how many panels can be mined within the resource. If the resource is not large enough to support a single panel with two development sections, then it is assumed that longwall mining cannot be pursued and will not be simulated.

Timing of longwall panels and development

Continuous mining is used in the development of the longwall. The model assumes the same operating conditions for continuous mining teams used in longwall development as in a standalone continuous mine. To simulate a longwall mine, the model coordinates the timing of longwall panel mining to start when the two necessary development sections are completed. After the number of panels and development sections is determined, the time it will take to mine the sections and panels is determined.

Longwall shearer cutting and conveyor loading

The model assumes that the longwall shearer makes each pass at the rate of 35 – 82 feet [4] with a cutting depth of 35.1 – 40.7 inches [22]. With each pass, the shearer zigzags through the coal. Each pass cuts the coal and it is loaded to the conveyor belt. The volume of coal cut per each shearer pass is determined, and the shearer advance rate is used to estimate the theoretical shearer production rate:

$$T_{LW} = LW_D \times Th \times LW_W \times \rho_B \tag{10}$$

$$LW_P = \frac{T_{LW} \times LW_{AR}}{LW_W}$$

where:
T_{LW} = tons of coal cut by longwall shearer
LW_D = longwall shearer cutting depth
LW_W = longwall face width
LW_P = longwall shearer production rate
LW_{AR} = longwall shearer advance rate

To determine the total amount of time it takes to mine a longwall panel, delays to straighten the longwall are added. It is assumed that the shearer takes 10 – 20 passes before it needs to be straightened, and that 30 – 90 minutes are needed to set it straight. Longwall move time between panels is assumed to take up to 4 weeks. Furthermore, data on coal conveyor losses is used; it is assumed that 8 – 12.9 tons/hour are spilled [12]. The production is adjusted to reflect these time delays and coal losses.

Estimating production rate for longwall mine

Total longwall production is comprised of the longwall panel and development section outputs, for the 1 – 2 longwalls assumed to be operating in the simulated longwall mine with associated continuous mining production. As mentioned above the development section production rate is determined in a similar fashion to the continuous mine simulation, accounting for possible delays in machine travel within narrower working areas. The estimated development section and longwall shearer production are added together to obtain the total production estimate for the longwall mine.

Coal Preparation Plant Simulation

Designing and simulating an onsite coal preparation plant was beyond the scope of this work. Instead, it is assumed that the majority of plants are Level IV plants. In 1996, a third of North American coal cleaning plants were Level IV [8] and it is assumed that this type of plant remains predominant today.

A Level IV plant has a 60 – 80% range of recovery, and consists of coarse and fine coal cleaning with froth flotation [8] from the run of mine production. The run of mine production rate is assumed to be coal plus partings. The amount of partings produced in addition to coal is estimated:

$$WR = \rho_B Area(M_{height} - Th) \qquad (11)$$

where:
WR = tonnes of waste rock mined over the entire mine lifetime
Area = area mined over mine lifetime

M_{height} = height of continuous miner or longwall shearer

It is assumed that partings within the coal seam itself are minimal. Based on this assumption, no waste rock is mixed with the run of mine output for a surface mine. For an underground mine, waste rock consists of the amount of overburden that the cutting machine – continuous miner or shearer – cuts from the roof in addition to cutting coal.

Project, or financial, life estimation

Based on the model simulation of production rate, the model assigns a financial lifetime to the mine project. The lifetime of the resource is estimated by dividing the total amount of coal in the resource by the production rate. A minimum financial lifetime of 10 years and a maximum of 30 years are assumed. If resource lifetime is less than 10 years, it is assumed that the financial lifetime of the project is 10 years. Similarly, if the resource lifetime is greater than 30 years, then 30 years of production and operation is assumed. For resource lifetimes between 10 and 30 years, the calculated lifetime is used.

Mine cost simulation

The model estimates costs corresponding to unit operations and steps in the production simulation for continuous, longwall, and surface mines. Costs are incurred before, during, and after mining. The four main process categories are premining, groundbreaking and preparation, operating and closure. Some costs are estimated following rules of thumb, such as pre-mine ground clearing. Other costs are estimated by interviewing industry experts, such as royalty and bonding costs. However, the majority of cost data used in the model is from the general literature [2, 9]. The engine sizing of the equipment is used to estimate the amount of fuel consumed to operate the equipment. Based on assumptions about the depreciation lifetime of equipment, it schedules equipment replacement. Costs for auxiliary operations, such as clearing surface land, digging shafts, installing and operating hoists and ventilation, are also estimated. Taxes on the sales of coal, purchase of capital, as well as those required by health, safety, and environmental regulations are estimated. These costs are all calculated according to the project lifetime that the model assigned to the mine. For all financial calculations, the model assumes an interest rate of 15 percent, and the financial lifetime estimated by the model as described above.

Equipment capital costs and depreciation

The capital costs of almost all mining equipment considered by the model were taken from the Western Mine Engineering Inc., Handbook. Table 1 shows the capital costs and equipment lifetime input into the model.

Table 1. Equipment Lifetime and Capital Cost[a]

Equipment Name	Life (Years)	Equipment Cost (Thousand 2005$)
Longwall shearer (46 – 177 inches)	5	1,700 – 2,500
Longwall shields	10	118 – 155
Face Conveyor and Stage Loader	5	1,709 – 3,197

Power Center and Hydraulic System	10	3,540
Continuous Miner	5	2,162 – 1,081
Shuttle Car	5	460 – 720
Roof Bolter	5	385 – 722
Rock Duster	7	25 – 30
Spare Shuttle Car	7	460 – 720
Conveyor Feeders/Breakers	5	275 – 315
Belt system (48 – 60 inches)	7	1,600 – 2,400
Power center (1500 kVa)	7	85.5
Power center (5000 kVa)	7	176
Shop/Warehouse facilities	30	243
Change facilities/mine offices	30	191
Access/Haulage road	30	280
Site/Surface building	7	93.9
Underground compressors and lines	30	130
Water/Sewage treatment facilities	30	67.1
Surface power substation and transmission lines	30	420
Mine dewatering system	30	101
Grader	7	2,060 – 2,420
240 ton truck	7	1,180 – 1,690
125 – 150 ton truck	7	8,810 – 2,700
Excavator shovel	5 – 7	3,613 – 8,810
Track dozer	5	50 – 400
Water truck	5	20 – 50
Rubber-tired dozer	5	18 – 30
Blasthole drill	5	633 – 777
Truck mounted coal drill	7	550 – 600
Fuel and lubricating oil truck	7	26 – 78
Longwall shield retriever	10	285 – 510
Personnel carrier	10	190
Self rescuer respirator	30	0.38
Shaft cutting machine	30	300 – 1,000

[a]Source: [2, 9].

Cost data for ventilation, hoists, and preparation plants were not readily available, because they are dependent upon mine size or production. The size and cost of these mine components were estimated by following general rules of thumb, found in the literature.

The model only considers ventilation systems and costs for underground mines. To estimate the cost to ventilate underground mines, the number of shafts and fans were determined. First, to estimate the number of shafts needed, it is assumed that the distance between shafts for an underground mine must be between 150 – 400 feet [23]. The number of shafts that can fit into the mine area are calculated, and assuming that the costs of inserting a shaft range from $82/ton - $1640/ton [24], the total cost of ventilation shaft sinking is determined. Second, the model sizes a ventilation system according to underground mine type. The method used by the model to size the ventilation system is adapted from those found in the literature, which bases the estimate on mine production rate [25]:

$$Q = 0.23(P_i)^{0.8} \tag{12}$$

$$f = \frac{ab \times OB_D}{1 + b \times OB_D} \tag{13}$$

$$Q_{adj} = fQ \tag{14}$$

where:
Q = air flow rate needed for mine, m^3/s
f = correction factor
a, b = correction factor coefficients
Q_{adj} = corrected air flow rate, m^3/s

The air flow rate (Eq. 12) is determined according to the production rate expected per mine type. However, mine production rate is not the only factor affecting ventilations requirements. Specific regional conditions also influence the amount of air needed in underground mining. Regional correction factors (Eq. 13) are used to determine a factor that can be used to estimate the actual air flow rate needed (Eq. 13). The model assumes regional correction factors for the Illinois No. 6 seam, such that a = 1.76 and b = 0.00075.

Having determined the necessary ventilation air flow rate, the model chooses fan sizes accordingly, and it is assumed that the fan will last the lifetime of the mine. Capital costs for fans and sizes are shown in Table 2.

Table 2. Underground ventilation fan and motor sizing and cost[a]

Air flow rate, m^3/s (tcf/min)	Fan Motor Size, W (hp)	Axial Fan Diameter, m (inches)	Fan Motor Capital Cost (1000 $)	Fan Capital Cost (1000 $)
≤ 47.2 (100)	40.6 – 223.1 (40 – 220)	1.54 (60)	20 – 70	81.6 – 101.6
≤ 94.4 (200)	243.3 – 567.8 (240 – 560)	2.13 (84)	40 – 116	40 – 180
≤ 141.6 (300)	365 – 851.6 (360 – 840)	2.43 (96)	53 – 134	134 – 164
≤ 188.8 (400)	486.7 – 1135.5 (480 – 1120)	2.54 – 2.94 (100 – 116)	70 – 182	195 – 225
≤ 236.0 (500)	608.3 – 1419.4 (600 – 1400)	3.05 (120)	78 – 220	195 – 225
≤ 283.2 (600)	730.0 – 1703.3 (720 – 1680)	3.05 (120)	90 – 250	200 – 246
≤ 19822 (700)	1135.5 – 1419.4 (1120 – 1400)	3 – 3.35 (120 – 132)	224 – 255	244.7 – 254.1
> 19822 (700)	1703.3 (1600)	3.66 (144)	224 – 255	244.7 – 265.1

[a]Source: [9].

It is assumed that 2 – 4 hoists are needed per mine [26, 27]. Individual hoist costs are dependent on the distance that they must move coal, supplies, and workers between the surface and mine workings. Hoist costs are evaluated for hoists of 1,000 – 3,000 feet. Capital and installation costs and the power rating of these hoists are shown in Table 3. The length of the hoist is determined according to the overburden depth overlying the seam.

Table 3. Hoist capital and installation costs, and motor size[a]

Depth, m (feet)	Cost (1000 $)	Engine power rating, W (hp)
305 (1,000)	800 – 3,800	253 – 3042 (250 – 3,000)
610 (2,000)	1,800 – 7,200	406 – 6083 (400 – 6,000)
1515 (3,000)	1,900 – 7,300	608 – 8111 (600 – 8,000)

[a]Source: [9].

As explained in a previous section, it is assumed that the on site preparation plant is a Level IV plant. The size and cost of this plant is, like the ventilation system, dependent on mine production rate. The capital cost of the plant was assumed according to the basic rule of thumb based on run of mine output [8]:

$$C = xROM \tag{15}$$

where:
C = prep plant capacity
x = cost multiplier
ROM = tonnes/s run of mine output

It is assumed that the cost multiplier is uniformly distributed between 3.8 and 15.2.

Having determined the capital cost of all equipment, the model assumes straight line depreciation to estimate depreciation costs over the mine's life. Throughout the mine's life, new capital expenses are incurred as equipment is replaced at the end of its life. The number of equipment per type of mine is shown in Table 4.

Table 4. Quantity of Equipment Assumed per Mine[a]

Equipment Name	Longwall Mine	Continuous Mine	Surface Mine
Longwall shearer (46 – 177 inches)	1 – 2	0	0
Longwall shields	5-10	0	0
Face Conveyor and Stage Loader	1 – 2	0	0
Power Center and Hydraulic System	1 – 2	0	0
Continuous Miner	4-12	4-16	0
Shuttle Car	6 – 30	9-20	0
Roof Bolter	4 – 20	2 – 6	0
Rock Duster	6 – 30	6 – 18	0
Spare Shuttle Car	3	3	0
Conveyor Feeders/Breakers	1	1	0
Belt system (48 – 60 inches)	8-22	4-20	0
Power center (1500 kVa)	1	1	0
Power center (5000 kVa)	1	1	0
Shop/Warehouse facilities	1	1	1
Change facilities/mine offices	1	1	1
Access/Haulage road	1	1	3 – 10
Site/Surface building	1	1	1
Underground compressors and lines	1	1	0
Water/Sewage treatment facilities	1	1	1
Surface power substation and	1	1	1

transmission lines			
Mine dewatering system	1	1	1
Grader	0	0	2
240 ton truck	0	0	2 – 14
125 – 150 ton truck	0	0	2 – 14
Excavator shovel	0	0	1 – 7
Track dozer	0	0	3 – 21
Water truck	0	0	3
Rubber-tired dozer	0	0	2
Blasthole drill	0	0	1 – 7
Truck mounted coal drill	0	0	1
Fuel and lubricating oil truck	0	0	2
Longwall shield retriever	1	0	0
Personnel carrier	5	5	0
Self rescuer respirator	10	10	0
Shaft cutting machine	1	1	0
Ventilation system	1	1	0
Preparation plant	1	1	1

[a]Sources: [26, 27].

Cost of consumables

The model estimates the amount of electricity, diesel and lubricating oil are needed to run the equipment. It also estimates the amount of ANFO needed to clear overburden from the coal resource for surface mining operations. Water, though used throughout the mining process, is not included in the model. The amount of fuel needed is estimated, based on the engine size of equipment. The model estimates these costs, instead of using the published data in the Western Mine Engineering Inc., Handbook, because it allows for greater flexibility in adjusting for real commodity costs. That is, users can change the electricity, diesel and lubricating oil costs in the model in order to estimate the cost to operate mining equipment.

To estimate energy needs, the model determines the amount of electricity, diesel, and lubricating fuel based on the equipment's operating time, an experience based factor per consumable category, and assumptions of consumable price. 2005 prices for electricity and diesel are assumed to be 0.056 – 0.064 \$/kWh, 2.52/gallon, respectively [28]. The current cost of lubricating oil could not be found, and it is assumed that a large operation like a mine would buy lubricating oil in bulk at a price that is prenegotiated with a seller. Therefore, the lubricating oil cost is estimated, based on a regression equation calculated from reported Western Mine Engineering Inc., Handbook lubricating cost data. This equation estimates lubricating oil costs as a function of engine size and capital cost:

$$L = 0.07613805 + 0.00022 \times PR + 5.602 \times 10^{-6} C_{cap} \qquad (16)$$

where:
L = lubricating oil price, \$/gallon
PR = equipment power rating
C_{cap} = equipment capital cost

Power ratings of equipment that requires lubricating oil are shown in Table 5. These power ratings are also to estimate the amount of electricity and diesel fuel consumed; the third and fourth columns indicate whether the equipment is electric or diesel powered.

Table 5. Power Rating of Mining Equipment[a]

Equipment Name	Power Rating, (hp)	Electric	Diesel
Longwall shearer (46 – 177 inches)	247 – 433	X	
Face Conveyor and Stage Loader	600 – 1800	X	
Continuous Miner	300 – 900	X	
Shuttle Car	40 – 80	X	
Roof Bolter	40 – 140	X	
Rock Duster	10	X	
Spare Shuttle Car	40 – 80	X	
Conveyor Feeders/Breakers	150 – 180	X	
Belt system (48 – 60 inches)	550 – 800	X	
Grader	140 – 500		X
240 ton truck	1790 – 2166		X
125 – 150 ton truck	1050 – 1200		X
Excavator shovel	3000 – 3350		X
Track dozer	70 – 120		X
Rubber-tired dozer	25 – 75		X
Blasthole drill	475 – 525		X
Truck mounted coal drill	525 – 700		X
Longwall shield retriever	100 – 150	X	
Personnel carrier	80	X	
Shaft cutting machine	100 – 400	X	
Ventilation	Varies, refer to Table 2	X	
Hoists	Varies, refer to Table 3	X	

[a]Source: [9]

Equipment operation hours are shown in Table 6. Continuous operation is assumed for power and safety equipment, such as the power centers, longwall shields, and ventilation. All other equipment is assumed to have 8 – 12 hours of down time during the day for maintenance. Equipment that is not continuously needed to extract coal, such as the grader, and blasthole drill, are operated as needed. Their operational hours are defined accordingly.

Table 6. Daily operating hours for mining equipment

Equipment Name	Operation (Hours/Day)
Longwall shearer (46 – 177 inches)	10 – 16
Longwall shields	24
Face Conveyor and Stage Loader	10 – 16
Power Center and Hydraulic System	24
Continuous Miner	10 – 16
Shuttle Car	10 – 16
Roof Bolter	10 – 16
Rock Duster	10 – 16
Spare Shuttle Car	10 – 16
Power center (1500 kVa)	24
Power center (5000 kVa)	24
Grader	2 – 4
240 ton truck	10 – 16

125 – 150 ton truck	10 – 16
Excavator shovel	10 – 16
Track dozer	2 – 20
Water truck	2 – 20
Rubber-tired dozer	2 – 20
Blasthole drill	1 – 5

As previously mentioned, ventilation, hoist, and preparation plant costs were not assembled from Western Mine Engineering Inc., Handbook information. Preparation plant operating costs are estimated by following rule of thumb, assuming that the operating cost per run-of-mine ton ranges from 0.50 – 4.00 \$/ton [8]. Ventilation and hoist operation costs are calculated separately.

The model calculates ANFO expense as the cost to supply necessary ANFO to clear overburden for surface mining. ANFO price is assumed to be 0.10 – 0.18 \$/lb [4].

Expected value of labor cost

It is assumed that the average mine will employ the proportion of employees per category as reported to the U.S. Bureau of Labor Statistics. As shown in Table 7, the range of occupations represented on a mine payroll range from office support, to mine management and machine operations, to construction and transportation support. The expected value of employee wages is calculated according to expected employment per type of mine. It is expected that all mines employ the same proportion of employees, except that surface mines will not employ underground mining specialists such as continuous miner operators, mine cutting and channeling machine operators, and roof bolters.

Table 7. Mine Occupation and Wages[a]

			Employment per Mine X = Yes 0 = No		
Occupation	**Percentage of Total Mine Workers**	**Annual Wages (Thousand \$)**	**Longwall**	**Continuous**	**Surface**
Management, business and financial	4.49	92.2	X	X	X
Professional and related	3.69	55.3	X	X	X
Service	0.47	26.0	X	X	X
Office and administrative support	3.4	31.9	X	X	X
Supervisors, construction and extraction	5.45	67.6	X	X	X
Construction trades and related workers	18.3	33.5	X	X	X
Other construction and related workers	18.3	33.5	X	X	X
Earth drillers, except oil and gas	0.32	38.1	X	X	X
Explosive drillers, ordinance handling experts, and blasters	0.77	42.0	0	0	X
Continuous mining machine operators	4.55	41.1	X	X	0

Mine cutting and channeling machine operators	1.82	40.3	X	0	0
Roof bolters, mining	5.9	42.3	X	X	0
Helpers – extraction workers	5.84	36.6	X	X	X
Extraction workers – all other	1.54	33.7	X	X	X
Installation, maintenance and repair occupations	13.2	43.8	X	X	X
Production support	13.2	43.8	X	X	X
Transportation and material moving	21.81	38.8	X	X	X

[a]Source: [29].

The data in Table 7 describes the types of workers employed by mines. The second and third columns list the percentage of mine workers and the total wages paid to those workers, per each category in the first column. The last three columns indicates the model's assumption about whether a given mine type will employ those workers. Using these data, the expected value of wages paid to mine employees:

$$W = \sum_{j} O_{i,j} \times S_j \times E_{i,j} \qquad (17)$$

where:
W = total annual wages to all mine employees
$O_{i,j}$ = percentage of employee of category j working in mine type i
S_j = mean annual reported salary for employee of category j
$E_{i,j}$ = 0 if category j employees are not employed at mine type i, 1 if category j employees are employed at mine type i

Expected value of the mine payroll is calculated, because it variation in the number and type of employees is not known. There are also non-miner employees that are employed, and it is not known how many of them are needed. Still, these positions – clerical, marketing, and other non-mining positions – are essential to mine operations and must be included in payroll estimation.

Land clearing costs

Before a resource can be mined, the land must be prepared for building construction, support roads, and mining activities. The model estimates clearing costs according to the estimation factors given by the literature [4]. It is assumed that the permitted surface area is being cleared. Permitted area is not necessarily the same as the mining area according to Equation (10), which is the area of the coal resource mined. The permitted area is all surface land that will be used for support facilities. For a surface mine, permitted area is assumed to be the same as the total mined area. However, for an underground mine, permitted area is assumed to be 25 – 50% of the total mined area. This fraction of surface land affected by underground mining is based on a 1997 ruling by Roderick Walston, which states that a maximum of 0.02 km^2 (5 acres) of support facilities are allowed for 0.08 km^2 (20 acres) of underground mining on federal lands. No data is available on the amount of surface land used for support facilities on private property, so it is assumed that the same practice holds true. The model determines clearing cost by the following:

$$CC_i = CCF_i \times Area_P^{0.9} \qquad (18)$$

Where:
CC_i = clearing cost for mine type i
CCF_i = clearing cost factor for mine type i
$Area_P$ = permitted area

The clearing cost factors for surface mining and underground mining are 75,000 – 500,000 \$/km^2 (300 – 2,000 \$/acre) and 640,000 \$/km^2 (1,600 \$/acre), respectively [4].

Taxes

Taxes estimated by the model over the mine lifetime are summarized in Table 8. Mine taxes are paid on items purchased, as well as coal produced and sold. In the United States, there are several environmental, health and safety regulations that levy taxes on mine operations. These taxes are predominantly paid as a function of the amount of coal that is mined; the proceeds are used to fund specific programs. Such taxes are the black lung tax and Surface Mining Control and Reclamation Act of 1977 tax. The Black Lung tax has an alternative rate, 4.40% of the price of coal if the price is less than $12/ton. However, the average U.S. price of coal is more than $12/ton, so the Black Lug tax rate based on production rate is assumed. Taxes on the sales of coal are federal income and state tax. State tax rate is assumed to be the Illinois state tax rate in this case. Taxes paid on the property and operational purchases such as fuel, electricity, and explosives, are also included.

Table 8. Mine Taxes

Tax	Rate, $/Ton	Rate, Percent	Description
Black lung	Surface, 3.00 Underground, 1.10		Paid on annual production
Capital		2	Paid on capital expenditures for equipment and surface support structures
Excise	Surface, 0.55 Underground, 1.10		Paid on annual production
Federal income		35	Paid on sales of coal, assuming 2005 U.S. price of $24.72/ton
Mineral valuation rate		1.7 – 30	Paid on the coal remaining in ground during mining operation period.
Real property tax rate		3.01	Paid on surface structure values. The model assumes that surface structure lifetime matches the maximum lifetime of the mine. The property value is adjusted by 30% for tax purposes.
Sales		6	Paid on consumables

			(fuel, lubricating oil, electricity, ANFO)
State income		1 – 10	Illinois state income tax rate paid on sales of coal, assuming 2005 U.S. price of $24.72/ton
Surface Mining Control and Reclamation Act of 1977	Surface, 0.35 Underground, 0.15		Paid on annual production

[a]Source: [1, 28, 30]

Royalties

It is assumed that royalties are paid on the mine production. Based on a conversation with a former mining consultant and Pennsylvania Department of Environmental Protection employee [31], it is assumed that royalties vary between 5 – 10% of sales on coal produced.

Permitting fees

The model estimates permitting fees, assuming the fees necessary to open a mine in Illinois. The permitting fee in Illinois is $125/acre for surface mines, and $5/acre for underground mines [32]. The area that the permitting fee applies to is the permitted area, or area used for surface support. Undermined lands due to underground mining are not included.

Bonding

The model assumes that the bond amount is based on the estimated reclamation cost. Typically, bond is posted by an insurer; leading insurers are Marsh USA, Etna Casualty Insurity, and St. Paul Fire & Marine Insurance Co. The cost to the mining company is an annual premium on the insurance policy until reclamation is completed. Alternatively, a letter of credit from a financial institution may be submitted, but the model does not evaluate the cost of this option.

Based on conversations with Marsh USA personnel [31], several assumptions about bonding fees are made by the model. Bonding fees are typically 4,000 – 15,000 $/acre for surface mined lands. Prime farm land is typically bonded at 10,000 – 12,000 $/acre. These costs include the cost of filling and regrading pits, soil replacement, and revegetation. For an underground mine, the bonding cost is approximately 3,000 $/acre. This cost covers removal of the surface structures, backfilling shafts, adding 4 feet of soil over any waste disposal areas. No bond is required on undermined lands, which are referred to as “shadow area.” Surface support areas include shafts, waste disposal, change rooms, conveyors. The Bureau of Land Management assumes that the bond premium is 5% of the total bond [33], but Marsh USA personnel state that reclamation bond rates are 100 – 150 basis points; in real terms, this is $10 - $15 per $1000 paid on an annual basis. The latter definition of the bond premium is assumed to be the current industry standard.

It is assumed that the mining operation must pay premiums on the bond from the time that mining starts through the time that the mine is reclaimed. In the absence of data on the amount of time that it takes to reclaim the mine, it is assumed that bond life after mining activities ends is 5 – 50 years. The bottom end of this assumption of reclamation time is based on the observation that a minimum of 10 years is required in areas of less than 26 inches, and a minimum of 5 years in areas of more than 2 feet of rainfall [34]. The top end of this range is defined at 50 years because there is little information about the total amount of time that reclamation bonds may be held as outstanding, and 50 years may be enough time to resolve reclamation requirements.

Discussion

Process based modeling is a tool to estimate mine production and cost, based on technology choices, unit operations and costs. The stochastic model described in this paper can account for uncertainty. It considers a range of possible equipment configurations within a range of geological conditions for a given mine, and outputs a range of likely costs and production rates. The stochastic results represent the fullest range of possibilities. This model considers geological conditions only, and is independent of delays that may be inherent due to operator preferences and site-specific problems.

The approach undertaken in this paper has several applications. It can estimate coal surface and underground mining costs in a new resource; the least cost means can then be chosen. This model is based on a simulated system of unit operations to extract coal. Unit operation improvements may be incorporated, to determine changes to production and cost. The benefit of a process-based model is two-fold; optimize resource development for lowest cost and greatest production, and evaluate new technologies if performance and cost data are known.

Data Validation

The model is used to simulate real U.S. coal mines, for which production and price data are available as well as seam thickness and depth. The simulation results are compared to the mines' coal prices and production rates. The validation dataset is a sample of U.S. coal mines. Seventeen longwall mines, ten surface mines, and fourteen continuous mines are simulated. These mines were selected because their seam thickness, overburden depth, and production rate data are publicly available. The seam characteristics are input into the model in order to simulate mining under those conditions. The coal resource area is unknown. The model's estimated 5th – 95th percentile ranges of production rate and cost are compared to the mine's historical production and price data. It is assumed that there the coal market is close to equilibrium. Therefore, coal price should fall within the range of projected mining costs.

Mine sample description and data sources

A comprehensive production and geological dataset for all U.S. coal mines is not available. The dataset described here is the most complete compilation of operating conditions and production rates from public data. The mine and coal seam data used in validation are compiled from the Energy Information Administration (EIA) Annual Coal Report, Illinois Department of Mines and Minerals annual statistical report, *Coal Age* magazine, and the Society of Mining Engineers Mining Engineering Handbook. The most complete reports are the Illinois Department of Mines and Minerals annual statistical reports and the *Coal Age* longwall census. The first is specific to

Illinois, but provides detailed configuration and production information about all Illinois mines; the second provides complete description of all U.S. longwall mines' configurations but no production data. The Illinois Department of Mines and Minerals annual statistical reports summarize Illinois coal mines' production rate, seam characteristics, and number of continuous mining units. The mines described in these reports are the lowest producers in the dataset. Coal resource and production data for mines outside Illinois were combined from several sources. Production data for the fifty top producing U.S. mines is available from the EIA Annual Coal Report; geological data for longwall mines and some of the surface mines on the list were available from the *Coal Age* longwall census and Society of Mining Engineers' 2nd edition Mining Engineering Handbook, respectively. The *Coal Age* longwall census also describes seam depth and thickness, as well as the number of panels and their dimensions. The uncertainty inherent in values reported varies by source. The Illinois Department of Mines and Minerals and EIA report discrete values, whereas the longwall census and SME report discrete values and ranges. The reporting style likely reflects the amount of information available from the operator.

Surface, continuous and longwall mines are all simulated according to the geological data collected. The seam depth and thickness data are input into the model in order to simulate the sample mines. The model is run for a range of coal resource areas between 494 – 2,300 acres. Some of the mine seam thicknesses, overburden and interburden depths are reported in the literature as ranges. In the case that a value range was available, it was input into the model as a uniform distribution of minimum to maximum value. The geological data for the sample mines are summarized in Table 9 - Table 11 while the ownership information and production data are presented in Table 12 - Table 14 for the surface, continuous and longwall mines, respectively.

The sample represents a breadth of production ranges and operations in varying geological conditions. Because more data was available throughout the U.S. for surface and longwall mines, these sample mines operated in the widest range of conditions. Continuous mines operated in the narrowest range of conditions because all sample data is from a few seams in Illinois. Surface mine seam thickness ranged from 0 – 55 ft, with up to ten seams extracted by a single operation. Interburden and overburden depths for the seam mined by the sample mines ranged from 10 – 200. Longwall mines included in the sample operated in seams almost as thick, 5 – 23 ft, and at much deeper depths, 300 – 9301 ft. In some cases, more than one longwall was operating at the mine site; in this case, if the seam thickness and overburden depths were not the same for both longwall units, the widest value range for seam thickness and overburden depth was used. Continuous mines operated in small seams, with thickness ranging from 5 – 8 ft and seam depths of 110 – 900 ft.

Table 9. Geologic characteristics for selected U.S. surface mines[a]

State	Company Name	Seam Name(s)	Seam Mininum Thickness, ft	Seam Maximum Thickness, ft	Minimum Seam Depth, ft	Maximum Seam Depth, ft
IL	Wildcat Hills	No. 6	4.5	NA	50	NA
		No. 7	2	NA	100	NA
IL	Eagle Valley	No. 6	4	NA	65	NA
IL	Creek Paum	M-Boro	4	NA	70	NA
		No. 5	4	NA	100	NA
		No. 6	6	NA	100	NA
IL	Elkville	No. 6	6	NA	100	NA
		No. 7	8	NA	90	NA
IL	Prairie Eagle	No. 7	2	NA	28	NA
IL	Red Hawk	No. 5	2	NA	110	NA
		No. 6	6	NA	80	NA
IL	Friendsville	Friendsville	5	NA	60	NA
CO	Colowyo Mine	Y3	5	NA	33	NA
		Y2	3	NA	36	NA
		X	13	NA	82	NA
		A2	4	NA	41	NA
		A3	2	NA	10	NA
		B	6	NA	54	NA
		C	6	NA	35	NA
		D	10	NA	29	NA
		E	7	NA	29	NA
		F	5	NA	21	NA
WY	Jacobs Ranch Mine	Upper Wyodak	0	8	150	200
		Middle Wyodak	40	55	0	38
		Lower Wyodak	0	9	0	73
TX	Big Brown Strip	NA	5	8	40	155
		NA	6	10	28	45

[a]Sources: [4, 35].

Table 10. Geologic characteristics for selected U.S. continuous mines[a]

State	Company Name	Mine Name	Seam	Seam Thickness ft	Seam Depth ft
IL	ICG Illinois	Viper	IL #5	6	280
IL	Freeman United Coal Mng.	Crown 2	IL #6	8	320
IL	Freeman United Coal Mng.	Crown 3	IL #6	8	365
IL	Knight Hawk Coal, LLC	Prairie Eagle U/G	IL #6	6	120
IL	Coulterville Coal Co	Gateway	IL #6	5	200
IL	Arclar Company	Willow Lake	IL #5	5	270
IL	Black Beauty Coal Co.	Wildcat Hills	IL #6	5	390
IL	Nubay Mining	Liberty Mine	IL #5	6	257
IL	Black Beauty Coal Co.	Riola	IL #6	6	250
IL	Black Beauty Coal Co.	Vermillion Grove	IL #6	6	250

IL	Wabash Mine Holding Co.	Wabash	IL #5	7	850
IL	White County Coal Corp.	Pattiki	IL #6	8	900
IL	Mach Mining LLC	Pond Creek	IL #6	7	460

[a]Sources: [35].

Table 11. Geologic characteristics for selected U.S. longwall mines[a]

State	Company Name	Seam Name	Seam Min Thickness, ft	Seam Max Thickness, ft	Min Seam Depth, ft	Max Seam Depth, ft
CO	Elk Creek	D	9	15	300	1600
CO	West Elk	B	23	NA	600	1400
CO	Foidel Creek Mine	Wadge	8	10	600	1400
IL	Galatia	Harrisburg (No. 5)	5	5	500	800
IL	Galatia	Harrisburg (No. 5)	5	5	450	550
NM	San Juan	Fruitland No. 8	10	15	450	1200
OH	Century Mine	Pittsburgh (No. 8)	5	NA	400	600
OH	Powhatan No. 6	Pittsburgh (No. 8)	5	NA	400	600
PA	Bailey	Pittsburgh	5	6	600	1000
PA	Enlow Fork	Pittsburgh (No. 8)	5	6	600	1000
PA	Enlow Fork	Pittsburgh	5	6	600	1000
PA	Cumberland	Pittsburgh (No. 8)	7	8	750	1050
PA	Emerald	Pittsburgh (No. 8)	6	7	380	950
UT	Sufco	Upper Hiawatha	7	17	800	1100
UT	Dugout Canyon	Rock Canyon	6	8	1000	1600
VA	Buchanan	Pocohontas No. 3	5	6	1400	2000
WV	McElroy	Pittsburgh	5	5	500	1000
WV	Loveridge	Pittsburgh	8	NA	1000	9300
WV	Robinson Run	Pittsburgh	8	NA	500	900
WV	Federal No. 2	Pittsburgh	8	NA	750	1400

[a]Source: [22, 36].

Simulation comparison data

Sample mine production, and state and national coal prices were used to evaluate the model's simulation output. These data for the three mine types, along with location and owner, are shown in Table 12 - Table 14. Average 2006 surface mine production is 5.0 million tons/year (Table 12), average continuous mine production is 1.2 million tons/year (Table 13), and average longwall mine production was 5.6 million tons/year (Table 14). The 2006 average national prices of surface and underground mined coal were $22/ton and $48/ton [36], respectively.

The surface mine data set includes small mines in Illinois and larger mines in Colorado and the Powder River Basin. The average production rate among large surface mines is 18 million tons per year [37]. At 40 million tons per year output, Jacobs Ranch mine produced more than twice the average top producing mine. Colowyo and Big Brown Strip are also among the top producing U.S. surface mines; they produced 6.2 million and 4.5 million tons in 2006, respectively. They produced a third or less of the average output for a top producing surface mine.

Table 12. Production and owner information per surface mine used in validation[a]

State	Company Name	2006 Production, Million Tons	Owner	State Coal Price, $/Ton
IL	Wildcat Hills	2.6	Black Beauty Coal Co	31.17
IL	Eagle Valley	0.2	Black Beauty Coal Co	31.17
IL	Creek Paum	1.4	Knight Hawk Coal, LLC	31.17
IL	Elkville	0.4	S Coal Co	31.17
IL	Prairie Eagle	0.8	Knight Hawk Coal, LLC	31.17
IL	Red Hawk	0.7	Knight Hawk Coal, LLC	31.17
IL	Friendsville	0.3	Vigo Coal Co	31.17
CO	Colowyo Mine	6.2	Colowyo Coal Company LP	24.27
WY	Jacobs Ranch Mine	40.0	Jacobs Ranch Coal Company	9.03
TX	Big Brown Strip	4.5	TXU Mining Company LP	18.61

[a]Sources: [35, 36].

Continuous mine production data used in this validation were reported in the Illinois Department of Mines and Minerals annual statistical reports [35]. Coal price data per state and the national average is also available [36]. None of the continuous mine owners are publicly traded companies. The owner per each mine, their 2006 production rate, and number of continuous mining machines are shown in Table 13. The least producing continuous mine is the Prairie Eagle mine. It produces an order of magnitude less than the next lowest producing mine. The continuous mine production is part of an underground and surface mining activity; the underground portion is not the primary focus of the mine, instead it provides some additional production.

Table 13. Production and owner information per continuous mine used in validation[a]

Owner	Mine Name	Number of Continuous mining Units	2006 Production, Million Tons	State Coal Price, $/Ton
ICG Illinois	Viper	6	3.9	31.17
Freeman United Coal Mng.	Crown 2	4	1.3	31.17
Freeman United Coal Mng.	Crown 3	5	1.6	31.17
Knight Hawk Coal, LLC	Prairie Eagle U/G	1	0.1	31.17
Coulterville Coal Co	Gateway	4	2.4	31.17
Arclar Company	Willow Lake	10	3.6	31.17
Black Beauty Coal Co.	Wildcat Hills	2	0.5	31.17
Nubay Mining	Liberty Mine	NA[b]	0.3	31.17
Black Beauty Coal Co.	Riola	2	0.3	31.17
Black Beauty Coal Co.	Vermillion Grove	4	1.4	31.17
Wabash Mine Holding Co.	Wabash	6	1.2	31.17
White County Coal Corp.	Pattiki	8	2.5	31.17
Mach Mining LLC	Pond Creek	2	0.1	31.17

[a]Source: [35, 36].
[b]NA = Not Available

Longwall description and ownership are summarized in Table 14. The range of production among the sample mines is 4.4 – 9.6 million tons. The average production rate of large longwall mines is 6.5 million tons; 8 of the sample mines exceed this production level and 14 are below it. All mines have one operating longwall except Galatia, Bailey, Enlow Fork, and McElroy. These two panel mines are located in 5 feet thick seams, but owe their high output to having more than one panel.

Table 14. Production and owner information per longwall mine used in validation

State	Mine Name	2006 Production, Million Tons	Owner	State Coal Price, $/Ton
CO	Elk Creek	5.1	Oxbow Mining	24.10
CO	West Elk	6.0	Arch Coal Incorporated	24.10
CO	Foidel Creek Mine	8.6	Peabody	24.10
IL	Galatia	7.2	Foundation	31.17
NM	San Juan	7.0	BHP Billiton	29.15
OH	Century Mine	6.5	American Energy Corporation	27.40
OH	Powhatan No. 6	4.4	Ohio Valley Coal	27.40
PA	Bailey	10.1	Consol Energy	37.40
PA	Enlow Fork	10.7	Consol Energy	37.40
PA	Cumberland	7.5	Foundation Coal	37.40
PA	Emerald	5.9	Foundation Coal	37.40
UT	Sufco	7.9	Arch Coal Incorporated	24.98
UT	Dugout Canyon	4.4	Arch Coal Incorporated	24.98
VA	Buchanan	5.0	Consol Energy	52.99
WV	McElroy	10.5	Consol Energy	45.94
WV	Loveridge	6.4	Consol Energy	45.94
WV	Robinson Run	5.7	Consol Energy	45.94
WV	Federal No. 2	4.6	Peabody	45.94

[a]Source: [36].

Production and Price Data Are Complicated

There is not a singular geographical, geological, or operational factor that predicts the production rate of any of the sample mines. There are operating conditions that are site specific that the model can not account for, which may run the gamut of innovative technology, more pieces of equipment, more efficient management, miner training and skills, which lend themselves to a high production rate. The number and type of equipment is likely the greatest factor in determining production rate differences among mines located in similar geological conditions.

The data shows that geological characteristics and production rates vary within the same seam. It is not possible to truly correlate productivity according to geography, seam thickness, seam, or company:

1. Production may vary within a state. For example, Illinois surface mine production rates range from 0.1 – 2.6 million tons per year. Illinois continuous mine production rates vary between 0.1 – 3.9 million tons per year. The longwall mines, Century and

Powhatan, are in the same seam in Ohio; however their production rates are 6.5 million tons and 4.4 million tons per year.

2. Production may vary within a seam. The sample set includes two surface mines that are both mining in Illinois No. 6 and No.7; these mines, Wildcat Hills and Elkville, produce 2.6 million tons and 0.4 million tons, respectively. No comment can be made about possible reasons for the discrepancy because the exact equipment configuration is not known. The continuous mines, Willow Lake and Liberty, are both located in the No. 5 seam, at the same reported thickness. The Liberty Mine mines are located under larger overburden depth than the Willow Lake mine, so produces less than Willow Lake. The Wabash mine is also in the No. 5 seam, in a thicker and deeper portion of the seam, and also produces less than Willow Lake because it has fewer continuous miner units and operates at greater depth. Continuous mining recovery rates decrease as overburden depth increases because more coal must be left in the pillars for roof stability. However, this mine produces almost 4 times the amount of coal that Liberty Mine produces. The Century, Powhatan No. 6, Bailey and Enlow Fork mines are all located in the Pittsburgh seam, at the same reported thickness. The Bailey and Enlow Fork mines are located under more overburden depth than the Century and Powhatan mines, but they are more productive because they have two longwall panels. Because they have two panels, they are more productive than the Cumberland and Emerald mines, which are also in the Pittsburgh seam, even though the latter mines are in a thicker portion of the seam.

3. Production may vary within a company. The Black Beauty Coal company owns two surface mines in Illinois that produce 0.17 million tons and 2.6 million tons; Knight Hawk coal owns three surface mines whose production range from 0.7 – 1.4 million tons per year. Nothing is known about the mine's equipment configuration, and reasons for the production difference. Black Beauty Coal owns two continuous mining operations in Illinois that are included in this sample. These mines are the Riola and Vermillion Grove mines. They are both located in the No. 6 seam, of almost the same thickness and depth. However, the Vermillion Grove mine produces about 4 times the amount of coal the Riola mine. Vermillion Grove has four continuous miner units, while Riola has two. In addition to being less equipped than Vermillion Grove, Riola has roof control problems [38]. Of the seventeen longwall mines examined for the data sample, six are owned by Consol Energy. However, the production rates for these mines vary from 5.7 million tons of coal per year for the Robinson Run mine in West Virginia to 10.7 million tons of coal per year for the Enlow Fork mine in Pennsylvania. The Robinson Run mine is located under less overburden depth than the Enlow Fork mine, and is located in a thicker portion of the Pittsburgh seam. The reason for this discrepancy is that there are two longwall panels operating at the Enlow Fork mine. There are also two panels operating at the Bailey and McElroy mines.

Factors Affecting Mining Costs That Can't Be Modeled

Although price is not the same as cost, it is the only publicly available financial data related to mining. The cost calculated by the model is not fully representative of the price charged by a company. Energy and sulfur content dictate the coal's quality and demand for it. Furthermore, there are operating costs beyond the minesite that are included in the price of coal, and sometimes transportation costs are added; these additional costs account for part of the difference between cost and price. In order to best estimate the difference between cost and price, the owner's annual revenue and profit were examined. Publicly held companies report their revenue and profit to the Securities Exchange Commission. Several of the mines are owned by large publicly held companies, and their overall revenue and profit are published in their annual 10-K report. The owner of each mine, their 2006 production rate, the 2006 price of coal in that state, and availability of publicly reported revenue and profit are shown in Table 4-6. None of the continuous and surface mine owners are publicly traded. Some mining companies in the sample are small, local companies that are not subsidiaries of a larger company; no 10-K report could be found. The rest of this discussion focuses on longwall mining, which can provide an example of factors affecting cost. The 2006 national price of coal, which is also used in order to validate the model's output, was $38.28 per ton. The national price is used because the coal price varies per region based on a variety of coal quality and extraction factors previously discussed. A national basis for comparison is useful to see how the model is applicable to a national average.

Table 15 summarizes annual revenue and net income reported by publicly held companies that own mines included in the data sample. All of these companies, except for BHP Billiton, specialize in coal mining. The larger revenues and net incomes reported by BHP Billiton in their 2007 annual report are likely due to their sales in other minerals. These data are used to estimate the price of coal to be charged, based on the estimated mining costs output by the model.

Table 15. Revenue and Net Income Reported by Public Companies (Billion$)

	Consol Energy[1]		**Arch Coal Incorporated**[2]		**Peabody**[3]		**Foundation Coal**[4]		**BHP Billiton**[5]	
	Revenue	Net Income	Revenue	Net Income	Revenue	Net Income	Revenue	Net Income	Revenue	Net Income
2007	3.72	0.27	2.41	0.17	4.57	0.26	1.49	0.03	41.27	13.50
2006	3.72	0.41	2.50	0.26	5.14	0.60	1.47	0.03	34.14	10.53
2005	3.81	0.58	2.51	0.04	4.55	0.42	1.32	0.09	24.76	6.63
2004	2.78	0.20	1.91	0.11	3.55	0.18	0.10	-0.05	NA	NA
2003	2.22	-0.01	NA	NA	2.73	0.03	0.10	0.03	NA	NA
2002	2.18	0.01	NA	NA	2.72	0.11	0.90	0.03	NA	NA

[1] [39]
[2] [40]
[3] [41]
[4] [42]
[5] [43]

The ratio between revenue and net income illustrates the percentage of revenue that may be attributed to profit or cost. The revenue and income for each company is shown in Table 3. From this, the percent of revenue that is cost is determined as

$$c_i = \frac{(R_i - I_i)}{R_i} \times 100 \qquad (19)$$

where: c_i = ratio of cost to revenue for company i

The model's cost ratio compared to historic price is determined as

$$C_{i,M} = \frac{P_{i,M} - C_{i,M}}{P_{i,M}} \times 100 \qquad (20)$$

where: $c_{i,M}$ = ratio of cost to price for company i, mine M

$P_{i,M}$ = price for company i, mine M

Equation 3 is computed using state and national price for coal.

The results of equations 19 and 20 per each mine is shown in Table 16.

Table 16. Percentage of Revenue Attributed to Cost, based on Company 10-K reports and Model Estimates

Mine Name	Ratio of Cost to Revenue	Owner
Elk Creek	NA	Oxbow Mining
West Elk	94	Arch Coal Incorporated
Foidel Creek	93	Peabody
Galatia	98	Foundation
San Juan	70	BHP Billiton
Century and Powhatan	NA	American Energy Corporation
Bailey and Enlow Fork	93	Ohio Valley Coal
Cumberland	98	Consol Energy
Emerald	98	Consol Energy
Sufco	94	Foundation Coal
Dugout Canyon	94	Foundation Coal
Buchanan	93	Arch Coal Incorporated
McElroy	93	Arch Coal Incorporated
Loveridge	93	Consol Energy
Robinson Run	93	Consol Energy
Federal No. 2	94	Consol Energy

The Bailey and Enlow Fork mines are paired in Table 16 because they operate under the same geologic conditions; the same is true for the Century and Powhatan mines. The Century and Powhatan mines are each owned by non-publicly traded companies, so that revenue and income data for those companies is not available. In general, companies operated on a slim profit margin. On average, 3 – 7% of their income was pure profit. The exception is the San Juan mine, owned by the large international company, BHP Billiton. The additional charges can include transportation, or items tabulated in the company's annual report.

Looking at company 10-K reports, additional costs related to mining as reported by companies owning the sample mines are summarized in Table 17. These items are described as affecting the reported cost and revenue reported in their 10-K reports. Not all companies provided this information. The costs in Table 17, are the additional costs that comprise price, which cover fire costs, accidents, property acquisitions and sales, are costs that reflect operation of a company beyond a single mine operation. The model does not reflect these costs, only the costs of a greenfield mine to extract coal under set geological conditions.

Table 17. Items that Affect Reported Costs and Profit

		Cost (-) or Profit (+), million $		
Company	**Item**	**2006**	**2005**	**2004**
Consol Energy	Buchanan Mine Fire	0	-34	NA
Consol Energy	Buchanan Mine skip hoist accident	0	-3	NA
Consol Energy	Sales contract buy outs	0	-13	NA
Consol Energy	Litigation settlements and contingencies	-1	-10	NA
Consol Energy	Incentive compensation	-24	-35	NA
Consol Energy	Bank fees	-9	-12	NA
Consol Energy	Accounts receivable securitization fees	0	-2	NA
Consol Energy	Terminal/River operations	-51	-24	NA
Consol Energy	Stock-based compensation expense	-23	-4	NA
Consol Energy	Miscellaneous transactions	-12	-19	NA
Arch Coal	Sale of select Central Appalachia operations	NA	75	0
Arch Coal	Peabody reserve swap and asset sale	NA	46.5	0
Arch Coal	West Elk combustion event			
	Idling	-30		0
	Insurance recovery	42	33	
Arch Coal	Accounting for pit inventory	-41	0	0
Arch Coal	Sales of interest in Natural Resource Partners LP	0	0	91
Arch Coal	Acquisition of Triton Coal Company, LLC	0	0	-382
Arch Coal	Acquisition of remaining interests of Canyon Fuel	0	0	NA

Results

Although mine performance varies throughout the country, the model is blind to geographic location. Results are presented and discussed by mine type, and are explained according to geological conditions input into the model.

The model's simulated production rate, and costs capture most of the actual output and price. Model results are dependent on data uncertainty. The size of range reflects the availability of data, and whether the data were input to the model as discrete values or ranges. Production rate is directly related to seam thickness in the model. Thicker seams have higher production rates than thinner ones. As expected, when more mining equipment units are included in the mine simulation, the estimated production rate increased. The model estimated the tightest range of production rates for mine types that had discretely reported geological characteristics. Therefore, it estimated the tightest ranges for continuous mining, followed by surface mining. The ranges of longwall estimated production rates and costs are greatest because longwall geological data was typically reported as data ranges. The continuous mine geological data was reported as discrete data points.

The 50th percentile estimate is mentioned here as a means to compare the output of simulating all three mine types, although the complete range of estimates should be considered when evaluating the model output. Considering the 50th percentile estimate, the model estimated the highest production rates for surface mines and longwall mines. The 50th percentile production rates for surface mines, longwall, and continuous mines were 1.5 – 8.2 million tons, 3.6 – 16.1 million tons, and 1.2 – 1.9 million tons, respectively. The model estimated highest 50th percentile mining costs for continuous mining, $33 – 46/ton. Longwall and surface mines simulated 50th percentile cost estimates range from $13 – 41/ton and $19 – 40/ton, respectively.

Comparison of surface mine simulation results to real mine data

The estimated ranges of production costs and rates are compared to actual price and production. With the exception of the top-producing surface mine in the sample set, real production was within 5 percent of the 5th or 95th percentile if it did not fall within the estimated range. Table 18 shows the 5th to 95th percentile range of surface mining cost estimates, with the 50th percentile estimates delineated within the range. The 2006 state coal price (**Table 14**) is compared to the model's estimated production cost range in Table 19; it can be seen that the historical price data fall within the cost estimate range. The range of estimated mining cost decreases as more seams are mined, or more uncertainty in the input data, e.g. seam thickness, is assumed.

As shown in Table 18, the model slightly overestimates production rate for the low producing mines, and underestimates the highest producer in the sample. For mines producing between 1 – 10 million tons per year, the actual production rate falls within the model's 5th – 95th percentile estimate range.

Table 18. Relationship between actual surface mine production rates and predicted production rates for baseline model assumption of 1 – 7 truck and shovel teams. X indicates where actual production falls within range.

Mine	Predicted Production, million short tons							Actual Production
		5th		50th		95th		
Creek Paum		0.6	x	1.7		3.9		1.4
Wildcat Hills		0.6		1.5	x	3.5		2.6
Eagle Valley	x	0.8		2.1		4.9		0.2
Elkville	x	1.1		3		8		0.4
Prairie Eagle	x	1.2		2.9		7.5		0.8
Red Hawk		0.5	x	1.3		3.1		0.7
Friendsville	x	1		2.8		7.6		0.3
Colowyo		2.6	x	8.2		22.1		6.2
Jacob's Ranch		2.1		6.3		15.5	x	40
Big Brown		1.3		3.1	x	11.8		8.6

Table 19 shows that except for the Jacob's Ranch mine, the average state price fell within the predicted range. In the case of the Jacob's Ranch mine, the 5th percentile cost estimate was within 11 percent of the actual price. The cost to price ratio was high, such that cost exceeded price in five cases, and greater than the 93 – 97% typical cost to price ratio shown in Table 16.

Table 19. Actual surface mined coal price and predicted mining cost for baseline model assumption of 1 – 7 truck and shovel teams. X indicates where actual price falls within predicted range.

Mine	Predicted Cost ($/Ton)						Actual State Price ($/Ton)	Cost-Price Ratio
		5th		50th		95th		
Creek Paum		19	x	33		114	31.17	1.06
Wildcat Hills		20	x	37		119	31.17	1.19
Eagle Valley		17		31	x	110	31.17	0.99
Elkville		14		24	x	104	31.17	0.77
Prairie Eagle		14		25	x	102	31.17	0.8
Red Hawk		22	x	40		122	31.17	1.28
Friendsville		14		24	x	106	31.17	0.77
Colowyo		10		19	x	98	24.1	0.79
Jacob's Ranch	x	10		21		102	9.03	2.33
Big Brown		11	x	23		100	18.61	1.24

Comparison of continuous mine simulation results to real mine data

A comparison of real mine production data for the continuous mines in the sample, and the model's estimated production ranges for these mines, is shown in Table 20. For mines producing between 1 and 2 million tons of coal per year, the model predicted a range of production rates that was inclusive of historical production for that mine. The model overestimated small producers and underestimated large producers. For mines that produced less than 1 million tons, the model overestimated production; likewise, for mines that produces more than 2 million tons, the model underestimated production.

In Table 20, we can see that the range of estimated continuous mine production is generally between 0.8 million and 2.5 million tons per year, with the 50th percentile production being about 1.5 million tons per year. The consistency in this estimate is due to the fact that all the sample mines lie within the same seams.

Table 20. Relationship of actual production rates to predicted rates for continuous mines assuming baseline model assumption of 3 – 4 operating continuous miner units. X indicates where actual production falls within the predicted range.

Mine	Production, million short tons							Actual Production
		25th		50th		95th		
Pond Creek	x	1.1		1.6		2.1		0.1
Crown 2		1.3	x	1.9		2.5		1.3
Crown 3		1.2	x	1.8		2.4		1.6
Prairie Eagle	x	1		1.4		1.9		0.1
Gateway		1.1		1.6		2.1	x	2.5
Wildcat Hills	x	0.8		1.2		1.7		0.5
Riola	x	0.9		1.4		1.9		0.3
Vermilion		1		1.4	x	1.9		1.4

Grove							
Pattiki		1.2		1.8	2.4	x	2.5
Viper		0.9		1.4	1.9	x	3.9
Willow Lake		0.8		1.2	1.7	x	3.6
Liberty Mine	x	0.8		1.2	1.7		0.3
Wabash		1.1	x	1.7	2.3		1.2

The 2006 Illinois underground coal price was within the 5th and 95th percentile cost estimated by the model for mining operations under the same geological characteristics as the sample mine, as shown in Table 21. Table 21 also shows that the 2006 national underground coal price was within the estimated cost range. For all the mines except for Wildcat Hills, Willow Lake and the Liberty Mine. For these three mines, the price was within 6 percent of the actual price. In all cases, the model's 50th percentile cost estimate overestimated price.

Table 21. Relationship of actual state prices to predicted costs for continuous mines assuming baseline model assumption of 3 – 4 operating continuous miner units. X indicates where actual price falls within the predicted range.

Mine	Predicted Cost ($/Ton					Actual State Price ($/Ton)	Cost-Price Ratio
		25th		50th	95th		
Pond Creek		27	x	37	52	31.17	1.19
Crown 2		24	x	33	44	31.17	1.06
Crown 3		25	x	34	46	31.17	1.09
Prairie Eagle		29	x	41	58	31.17	1.32
Gateway		27	x	37	51	31.17	1.19
Wildcat Hills	x	33		45	65	31.17	1.44
Riola		31	x	42	60	31.17	1.35
Vermilion Grove		30	x	41	58	31.17	1.32
Pattiki		25	x	34	48	31.17	1.09
Viper		30	x	42	59	31.17	1.35
Willow Lake	x	34		46	62	31.17	1.48
Liberty Mine	x	34		46	63	31.17	1.48
Wabash		28	x	36	50	31.17	1.15

Comparison of longwall mine simulation results to real mine data

For twelve of the seventeen mines examined in the sample set, the model predicted a range of production rates that was inclusive of historical production for that mine, as shown in Table 14. As previously stated, it is assumed that there are 1 – 2 shearers operating and 2 – 6 continuous mining teams in the mine. Seam thickness is the dominant factor in the model's determination of longwall production rate.

For all sample mines, the 2006 state and national coal price were within the 5th and 95th percentile cost estimated by the model (Table 22). The model generally estimated a 50th percentile cost estimate less than the national 2006 coal price and respective state coal price for each simulated mine.

The diversity in longwall production in Table 22 is owed to the variety of seams represented. The range of predicted production is 2.2 million to 26 million tons per year. Mine production is dependent on seam thickness, and the number of known operating longwall faces. In Table 22, we see that the 50th percentile estimate appears to be close to the actual rate in mines that produce 6 million tons or less.

Table 22. Relationship between actual and predicted longwall production rates assuming baseline model assumption of 1 – 2 longwall panels. "X" indicates where actual production falls within prediction range.

Mine	Predicted Production, million short tons							Actual Production
		25th		50th		95th		
Elk Creek		4.5	x	8.7		14.3		5.1
West Elk	x	10.1		16.1		26.3		6
Foidel Creek		3.8		6	x	10.1		8.6
Galatia		2.2		3.6		5.7	x	7.2
San Juan		4.9	x	8.2		15		7
Century		2.2		3.6		5.7	x	6.5
Powhatan		2.2		3.6	x	5.7		4.4
Bailey		2.5		3.8		6.3	x	10.2
Enlow		2.4		3.9		6.4	x	10.7
Cumberland		3.4		5.4	x	8.5		7.5
Emerald		2.9		4.6	x	7.5		5.9
Sufco		3.3	x	8.3		17.5		7.9
Dugout Canyon		2.9	x	4.7		8.1		4.4
Buchanan		2.3		3.7	x	6.2		5
McElroy		2.3		3.6		6.1	x	10.5
Loveridge		3.5		5.7	x	9.2		6.4
Robinson Run		3.4		5.5	x	8.9		5.7
Federal No 2		3.5	x	5.7		9.2		4.6

A comparison of price to predicted longwall mine costs is shown in Table 23. The actual price always fell within the predicted range. The cost to price ratio, calculated by comparing the 50th percentile to the price shows that in most cases the estimated cost was less than the price, but in five cases, it was greater than the price.

Table 23. Actual state prices and predicted costs for longwall mines assuming baseline assumption of 1 – 2 longwall panels. X indicates where actual price falls within prediction range.

Mine	Predicted Cost, $/Ton					Actual State Price ($/Ton)	Cost-to-Price Ratio
	5th		50th		95th		
Elk Creek	14		22	x	45	38.28	0.57
West Elk	13		23	x	166	24.1	0.95
Foidel Creek	16	x	26		64	24.1	1.08
Galatia	22	x	41		109	31.17	1.32
San Juan	13		21	x	47	29.15	0.72
Century	22	x	41		108	27.5	1.49
Powhatan	22	x	41		108	27.5	1.49
Bailey	21		37	x	100	37.4	0.99
Enlow	20	x	38		96	37.4	1.02

Cumberland	17		29	x	73	37.4	0.78
Emerald	19		32	x	76	37.4	0.86
Sufco	14		13	x	50	24.98	0.52
Dugout Canyon	20	x	32		70	24.98	1.28
Buchanan	22		39	x	84	52.99	0.74
McElroy	22		38	x	92	45.94	0.83
Loveridge	17		27	x	62	45.94	0.59
Robinson Run	17		28	x	63	45.94	0.61
Federal No 2	17		27	x	62	45.94	0.59

Production Rate Sensitivity Analysis

As discussed, the model estimates the "average" mine production range. The 5th – 95th percentile range is the extent of production expected for a given set of geological characteristics. For simplicity, only the 50th percentile estimate is compared to reported production rates in this sensitivity analysis. As shown in Figure 2, the 50th percentile model results underestimate production rate for mines producing more than 5 million tonnes per year and overestimates that for mines producing less than 1 million tonnes per year.

The model results reflect the typical, or average, mine. This may be expected, because the model assumes "typical" equipment configurations per mine type based on industry literature. For example, U.S. longwalls have 1 or 2 longwall faces [22]. Therefore the model assumes a uniform distribution of 1 or 2 faces operating in the mine, such that the mean is 1.5 faces. Similarly the model assumes a uniform distribution of 1 to 7 shovel and truck teams. In order to determine whether the model is adequate to simulate large and small producers, the model was with discrete quantities of shovel and truck teams, continuous miner units, and longwall faces.

Because the exact number of truck and shovel teams was not known for the sample surface mines, quantities through a maximum of 24 teams were modeled. The numbers of continuous miner units and longwall faces were available from the Illinois Office of Mines and Minerals annual statistical report and *Coal Age* longwall census, respectively.

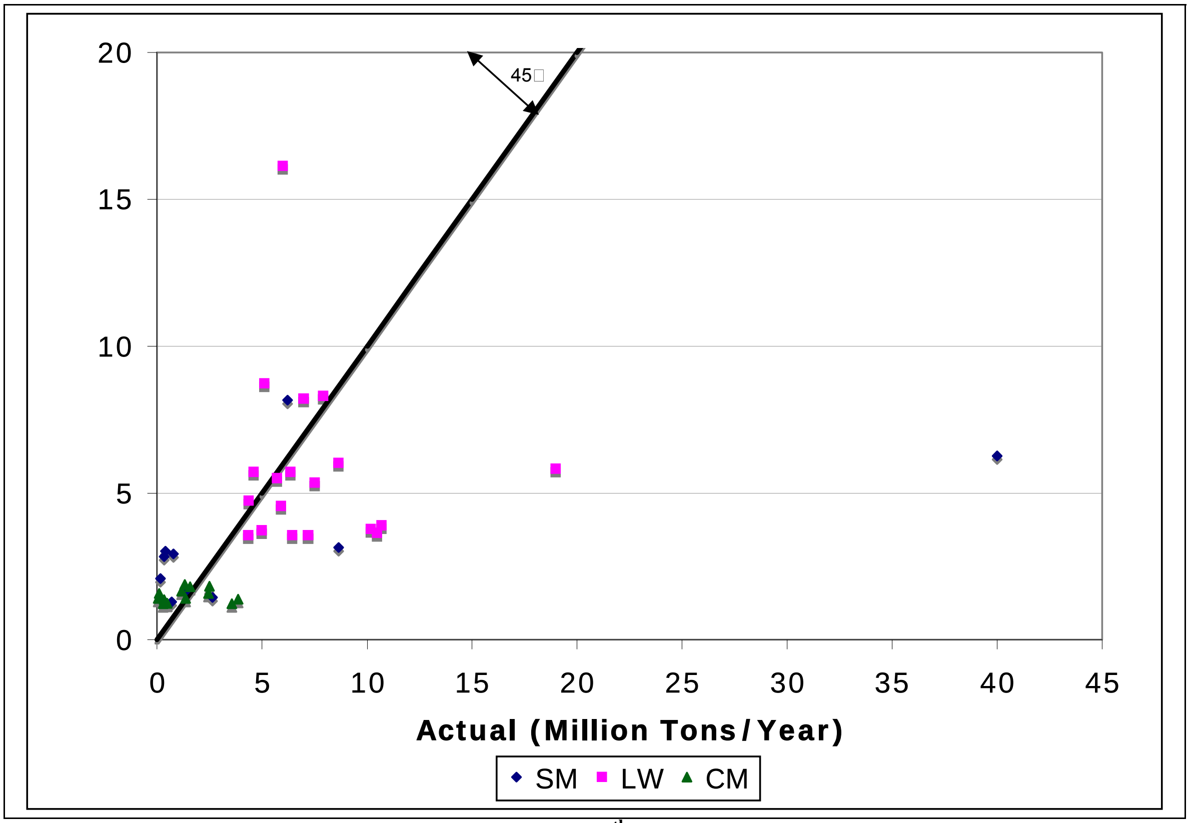

Figure 2. Relationship of actual production and 50th percentile predicted production for all mines.

Production rates were estimated for surface mines with 4, 8, 16, and 24 trucks, continuous mines with 1, 2, 3, 4, 5 or 6 continuous miners, and longwall mines with 1, 2, or 3 longwall faces. These equipment levels were chosen based on known equipment configurations. The Jacob's Ranch mine, the largest producing surface mine in the sample, uses a dragline, 46 trucks, 8 shovels in its mining operation [44]. In the model, this truck configuration is equivalent to 24 truck teams, the additional shovel output calculated is assumed to be equivalent to the mine's dragline output. Therefore, increments between 4 and 24 were run through the model in order to examine how production is affected. The sample continuous mines have 1 – 10 continuous miners [35]. The Galatia, Enlow Fork, Bailey and McElroy mines, all large longwall mines that produce 10 million tons of coal or more, have 2 longwall faces. All other longwall mines in the sample have 1 longwall face.

Comparisons of the estimated 50th percentile production rate to the reported rate are shown in Table 24 - Table 29. Table 24 displays the varying production rates of increasing surface mining equipment. As expected, production increases as more equipment is used. The 50th percentile estimates increase with greater steps in sample mines where there is greater uncertainty in thickness and depth data. The majority of the low producing mines have discrete thickness and depth data, so production estimates as additional equipment teams are added to the simulation model do not result in as large an increase for the large producing mines for which the depth and thickness are reported as ranges. Table 17 shows estimated production increases as continuous miners are added to the model mine, and is closer to the actual production rate for the low producers than the production estimate resulting from the baseline assumption of 1 – 7 continuous miner units. Table 18 shows the results of estimating longwall production rate by simulating the true number of longwall panels per mine. As expected, when simulating the real number of longwall panels per mine, the estimated production rate fell for one panel mines and rose for two panel mines. All of the two panel mines are located in seams that are approximately five feet thick. The model predicts the same mining rate for these mines, despite their location at different depths. The construction of a longwall mine at any depths is the same. Gateway pillars in the development section are the same size regardless of depth, and panels are always of the same dimensions. Knowing the number of panels tightens the predicted range of production rates.

Table 24. Relationship Between Actual Surface Mined Production Rates and 50th Percentile Prediction Rates for 4, 8, 16, or 24 Truck and Shovel Teams. X indicates actual production rate within estimate range.

Mine	50th Percentile Predicted Production, million short tons				Actual Production
	4 teams	8 teams	16 teams	24 teams	
Creek Paum	x 2.4	4.7	9.5	14.2	1.4
Wildcat Hills	2.1	x 4.1	8.2	12.3	2.6
Eagle Valley	x 2.9	5.8	11.6	17.4	0.2
Elkville	x 4.2	8.3	16.7	25	0.4
Prairie Eagle	x 4.3	8.7	17.3	26	0.8
Red Hawk	x 1.9	3.7	7.4	11.1	0.7
Friendsville	x 4	8.1	16.2	24.3	0.3
Colowyo	x 8.8	17.7	35.3	53	6.2
Jacob's Ranch	6.2	12.3	24.6	37	x 40
Big Brown	4.7	x 9.3	18.7	28	8.6

Table 25. Relationship of actual continuous mine production rates predicted rates for known number of operating continuous miner units. X indicates actual production rate within range.

Mine	Predicted Production, million short tons			Continuous Miner Units	Actual Production
	5th	50th	95th		
Pond Creek	x 0.6	0.9	1.3	2	0.1
Crown 2	x 1.5	2.1	3	4	1.3
Crown 3	x 1.8	2.6	3.6	5	1.6
Prairie Eagle	x 0.3	0.4	0.6	1	0.1
Gateway	1.3	1.8	x 2.5	4	2.5
Wildcat Hills	0.5	x 0.7	1	2	0.5
Riola	x 0.5	0.8	1.1	2	0.3
Vermilion Grove	1.1	x 1.6	2.2	4	1.4
Pattiki	x 2.8	4.1	5.8	8	2.5
Viper	1.6	2.4	x 3.3	x 6	3.9
Willow Lake	2.5	3.5	x 5.1	10	3.6
Liberty Mine	0	0	0	x NA	0.3
Wabash	x 2	2.7	3.9	6	1.2

NA = not available

Table 26. Relationship of Actual Longwall Production to Predicted Production Range for Known Number of Operating Panels. X indicates actual production within range.

Mine	Predicted Production, million short tons						Number of Longwall Panels	Actual Production
	5th		50th		95th			
Elk Creek	4.1	x	6.1		8.1		1	5.1
West Elk	5.9	x	7.6		9.1		1	6
Foidel Creek	3.4		4.5		5.5	x	1	8.6
Galatia	4.6		6.4	x	8.1		2	7.2
San Juan	4.7		6.3	x	8.5		1	7
Century	2		2.6		3.1	x	1	6.5
Powhatan	2		2.6		3.1	x	1	4.4
Bailey	5.2		7.1		9.2	x	2	10.2
Enlow	4.9		7.3		9	x	2	10.7
Cumberland	3		3.9		4.8	x	1	7.5
Emerald	2.5		3.4		4.1	x	1	5.9
Sufco	3.3		6.1	x	8.3		1	7.9
Dugout Canyon	2.8		3.5		4.3	x	1	4.4
Buchanan	2.2		2.9		3.4	x	1	5
McElroy	3.5		6.3	x	10.9		2	10.5
Loveridge	3.3		4.2		5	x	1	6.4
Robinson Run	3.1		4		4.8	x	1	5.7
Federal No 2	3.3		4.2	x	5		1	4.6

As shown in Table 12, there are several surface mines that produced a small amount of coal – these mines are Eagle Valley, Friendsville, and Elkville. The model overestimated production for mines that produced less than 3 million tons of coal in 2006, and the Colowyo mine. When the model assumed 8 teams mining, it then overestimated Big Brown and Wildcat Hills production by 8 and 56 percent. The Jacob's Ranch mine was still underestimated by 8 percent when 24 teams were assumed. Based on these results, it can be said that for mines producing less than 1 million tons per year, it might be suitable to simulate less than 4 teams in the model, for mines producing less than 10 million tons per year, between 4 and 8 teams, and for mines producing 40 million tons, a 24 team assumption results in a fairly close estimate to the reported production rate.

In Table 27, the 50th percentile surface mining cost estimates for 16 and 24 team simulations overestimate costs. For mines producing less than 0.4 million tons per year, the model's four and eight team simulations underestimate cost. For most of these mines, the actual price falls within the simulated cost of sixteen or fewer teams. The exception is the largest producer, which is also known to have 24 teams, but the simulated cost assuming this equipment quantity is three times the actual price.

Table 27. Relationship of Actual Surface Mine Coal Price and 50th Percentile Predicted Mining Cost for 4, 8, 16, or 24 Truck and Shovel Teams. X indicates actual coal price within range.

Mine	Actual Price ($/Ton)	50th Percentile Cost Estimate ($/Ton)							Cost-Price Ratio			
			4 teams		8 teams		16 teams	24 teams	4 teams	8 teams	16 teams	24 teams
Creek Paum	31.17	x	35		40		53	64	112	128	170	205
Wildcat Hills	31.17	x	39		46		60	73	125	148	192	234
Eagle Valley	31.17		31	x	35		44	58	99	112	141	186
Elkville	31.17		23		26	x	33	41	74	83	106	132
Prairie Eagle	31.17		24		27	x	32	41	77	87	103	132
Red Hawk	31.17	x	43		50		67	85	138	160	215	273
Friendsville	31.17		24		27	x	34	43	77	87	109	138
Colowyo	24.1		18		21	x	25	28	75	87	104	116
Jacob's Ranch	9.03	x	18		20		25	29	199	221	277	321
Big Brown	18.61	x	22		25		32	37	118	134	172	199

Table 28 shows that the model always overestimates price. However, actual price fell within the predicted range in all cases except for Wildcat Hills, Willow Lake and Liberty Mine. In the case of these three mines, the actual price was within 6% of the predicted 5th percentile cost.

Table 28. Relationship between actual and predicted continuous mining cost for known number of continuous miner units. X indicates actual cost within range.

Mine	Actual Cost ($/Ton)	Predicted Cost ($/Ton)				Cost-Price Ratio
			5th		50th	95th	
Pond Creek	31.17		27	x	37	52	1.19
Crown 2	31.17		24	x	33	44	1.05
Crown 3	31.17		25	x	34	46	1.08
Prairie Eagle	31.17		29	x	41	58	1.30
Gateway	31.17		27	x	37	51	1.18
Wildcat Hills	31.17	x	33		45	65	1.45
Riola	31.17		31	x	42	60	1.34
Vermilion Grove	31.17		30	x	41	58	1.31
Pattiki	31.17		25	x	34	48	1.09
Viper	31.17		30	x	42	59	1.33
Willow Lake	31.17	x	34		46	62	1.47
Liberty Mine	31.17	x	34		46	63	1.48
Wabash	31.17		28	x	36	50	1.17

Longwall costs are represented accurately when the true number of longwall panels per mine are simulated. As shown in Table 29, the real price falls within the estimated cost range, close to the 50th percentile predicted cost. When looking at cost estimate, the difference in seam depth is apparent. The deeper the mine for the same thickness seam, more money is spent, presumably on accessing the seam from the surface. Again, knowing the number of operating panels decreases the estimation uncertainty and range. The predicted range still captures the actual price.

Table 29. Relationship of Actual Longwall Coal Price and Predicted Longwall Cost. X indicates actual cost within predicted range.

		Predicted Cost ($/Ton)					
Mine	**Actual Cost**	25^{th}		50^{th}		95^{th}	**Cost-Price Ratio**
Elk Creek	38.28	14		22	x	45	58
West Elk	24.1	13		23	x	166	96
Foidel Creek	24.1	16	x	26		64	108
Galatia	31.17	22	x	41		109	132
San Juan	29.15	13		21	x	47	72
Century	27.5	22	x	41		108	146
Powhatan	27.5	22	x	41		108	146
Bailey	37.4	21		37	x	100	100
Enlow	37.4	20	x	38		96	103
Cumberland	37.4	17		29	x	73	78
Emerald	37.4	19		32	x	76	86
Sufco	24.98	14		13	x	50	52
Dugout Canyon	24.98	20	x	32		70	128
Buchanan	52.99	22		39	x	84	74
McElroy	45.94	22		38	x	92	83
Loveridge	45.94	17		27	x	62	59
Robinson Run	45.94	17		28	x	63	61
Federal No 2	45.94	17		27	x	62	59

Discussion

The model was able to estimate a range of production costs and rates within 5 – 11 percent of historic prices and production rates. In many cases, the historic mine performance data did not fall within the 5^{th} and 95^{th} percentile estimates. The model, however, is suitable to simulate mine production and costs.

The model is sensitive to the input data. If the coal seam data is reported as a range, the uncertainty inherent in this information leads to tighter estimated cost ranges, but greater uncertainty in production rate estimates. In the case of surface mines, additional trucks and shovels are more costly in mines that have discrete definitions of thickness and depth. More accurate production estimates were achieved when known quantities of continuous miner units and longwall panels were simulated. However, specific configurations of surface mines were not available to complete a more detailed simulation.

National Average Cost Curve

Introduction

To determine whether there is substantial coal to meet demand, the baseline cost model (Chapter 1) is applied to U.S. coal regions. It is not a complete resource assessment because geological data for all U.S. coal regions is not available. Furthermore, the data uncertainty is not clearly defined. The estimated costs are dependent on the quality of the data input to the model. The result of the analysis described in this chapter is a sketch of average mining cost to meet projected U.S. coal demand.

The full available coal dataset is analyzed. A subset of the data is also evaluated, to determine the resource cost change if maximum mining overburden decreases. The first resource scenario defines resources per the USGS definition of "demonstrated reserve base" (DRB) [45], which is accepted and followed by the NAS in their recent coal resource report [46]. The full seam thickness is considered, but overburden depth is restricted to the DRB maximum allowable depth of 1,000 feet. The second resource scenario narrows the DRB to shallower overburden depths, where the resource is more concentrated. The depth at which the most coal is available is referred to as the "modal overburden" or "overburden mode." It is assumed that modal overburden depth is the maximum depth to which mining would be pursued in the coalbed. Full seam thickness is still considered. The first coal resource scenario results in a larger defined coal resource, while the latter is more restricted but results in lower estimated mining costs.

Data Description and Manipulation

Coal resource and historical demand data are used to establish the amount of coal available and the annual quantity needed. Although the demand data are fairly straight forward – annual coal demand was recorded on an annual basis since 1947 – national coal data is inconsistently reported. If the coalbed was presumed to be inexpensive to mine, then it is richly described. For example, overburden and thickness estimates for coalbeds in the Powder River Basin are reported for depths up to 11,000 feet. However, less attractive seams, such as the Kittaning coal seam in northern Appalachia, are simply described as "deeper than 700 feet." To represent the available coal data a consistent basis, the USGS established maximum "demonstrated reserve base" overburden depth is used in order to estimate the true resource extraction cost. To evaluate the cost to extract the majority of a given seam, the modal overburden depth is assumed to be the maximum overburden depth. Grouping and compilation of resource quantities and characteristics are described in the following sections.

Coalbed data was collected from the USGS National Coal Resource Assessment (NCRA), the most complete U.S. coal geological dataset. The result of this assessment is a set of reports that summarize location, overburden depth, seam thickness, and coal quality of coal coalbeds that have current and past mining activity in the Colorado

Plateau, Rocky Mountains and Great Plains, Northern and Central Appalachia, Illinois, and Gulf Cost coal basins.

Overview

In 1999, the USGS began the NCRA, which was borne out of the need to understand how much coal was available in the U.S. Five regions were examined: Appalachia, Gulf Coast, Illinois, Northern Rocky Mountains and Great Plains and the Colorado Plateau. Of the data examined thus far, the best dataset appears to be the Northern Rocky Mountains, due to the interest in developing the Powder River Basin coalfield. It excludes coalfields where there is no mining; these include the Interior basin, Alaska coal, southern Appalachia, and part of the Gulf Coast region. It is believed that the five regions assessed will continue to serve as the main coal source in the U.S. Table 30 summarizes the dates in which the NCRA reports were published.

Table 30. NCRA reports in order of publication on the USGS website, total estimated coal

Region	Year
Northern Rocky Mountains and Great Plains	1999
Colorado Plateau: Arizona, New Mexico, Utah	2000
Northern and Central Appalachian Basin	2000
Illinois Basin	2002
Gulf Coast: Central Texas and Northwestern Louisiana	2002, 2005
Powder River Basin: Sheridan-Birney coalfields, Birney-Custer-Recluse coalfields [47]	In Progress, report given in 2006

The NCRA also includes an assessment of the economical viability of mines for some of the reported coal resources. This subevaluation of the NCRA, is the Recoverable Coal Resource Assessment (RCRA). It used the CoalVal model to estimate the regional cost of coal extraction on a regional basis for the Pittsburgh seam, Herrin, Danville, Springfield and Colchester seams in the Illinois Basin, and the Wyodak-Anderson seam in the Gillette coalfield. As described in the USGS contracted peer review [48], CoalVal assumed a ±25% uncertainty. Mines simulated by CoalVal had to achieve 60% NPV within 10 years in order to be deemed a "Logical Production Unit" for that region, and thereby deem the coalbed to be cost effective to mine. Preparation plant capital and operating costs were not included for Eastern coal, and it is not mentioned whether these costs were included for the Illinois and Gillette basin coals.

Data uncertainty

USGS describes the extent to which the coal resource is geologically measured and analyzed according to distance from the sample site. It also provides ranges for the reporting of seam thickness and overburden depth. USGS defined levels of measurement and projection describe the total available coal quantity, as well as provide rules of thumb by which to judge whether the coal is better mined by surface or underground methods. The Energy Information Administration (EIA) and the National Academy of Science (NAS) in their reports about coal resources also follow these standards.

Coal available per coalbed is reported according to thickness of overburden and seam thickness categories as defined in USGS Circular 891 [45]. On the basis of these defined categories, surface mining is not an option for mines more than 500 feet deep; underground mining can be pursued at all depths (Table 31). Although the USGS defined mandatory overburden depth reporting categories, the categorical ranges vary throughout the NCRA reports.

Table 31. Mandatory and optional overburden and seam thickness categories defined by the USGS Circular 891	
Overburden depth	
Mandatory underground mining categories	**Mandatory and optional surface mining categories**
0-500 feet (0-150 m)	0-500 feet (0-100 m) mandatory use
500-1000 feet (150-300 m)	0-100 feet (0-30 m) optional use
1000-2000 feet (300-600 m)	100-200 feet (30-60 m) optional use
2000-3000 feet (600-900 m)	0-200 feet (0-60 m) optional use
3000-6000 feet (900-1800 m)	200-500 feet (60-150 m) optional use
Optional other occurrence category: >6000 feet (>1800 m)	
Thickness	
Anthracite and bituminous coal	**Subbituminous coal and lignite**
14-28 inches (35-70 cm)	2.5-5 feet (75-150 cm)
28-42 inches (70-105 cm)	5-10 feet (150-300 cm)
42-84 inches (105-210 cm)	10-20 feet (300-600 cm)
84-168 inches (210-420 cm)	20-40 feet (600-1200 cm)
168 inches or thicker (420 cm+)	40 feet or thicker (1200 cm+)

The four "reliability" levels of resource reporting, based on level of conjecture and measurement are "measured," "identified," "inferred," and "hypothetical." The most detailed level of resource data are "measured" and deemed most "reliable", which means that the depth, thickness, and coal quality measurements are directly measured with a high degree of geological assurance with sampling points less than 0.5 miles apart. The amount of "measured" coal available is known to be within 0.25 miles from the measurement site. On the opposite end of the spectrum, "hypothetical" coal resource is completely projected. This coal lies more than 3 miles from a sampling point, and has not officially been discovered. Further exploration would establish that it truly exists. In between these two extremes are "indicated" and "inferred" resources. "Indicated" resource estimates are based partly on measurements and partly on projection. This type of resource is projected to lay 0.25 – 0.75 miles from sampling points. "Inferred" resource estimates are mostly projected data based on assumptions about the coal bed's geology, and projected to be within 0.75 – 3.0 miles from the sampling points. Due to the accuracy of reported data, "measured" and "indicated" coal in seams more than 28 inches thick at depths up to 1,000 feet comprise the Demonstrated Reserve Base (DRB) [49].

Data critique and NAS questions

The raw NCRA data shows the wide range in how coal characteristics are reported per region, and per bed or seam within that region. Some of the data is more than 50 years old. Appendix 1 tabulates the thickness and overburden depth ranges per coalbed, and amount of coal reported per measurement category. There are a variety of reported ranges and degrees of measurement. The total raw data totals 976 billion short tons of coal. The data also shows that a third of the reported resource is "inferred" and "hypothetical"; there are 457 billion short tons of "measured" coal, 157 billion short tons of "indicated" coal, 153 billion short tons of "inferred" coal and 165 billion short tons of "hypothetical" coal. The official USGS review of the nation's coal resources conclude that 2.24 trillion short tons of the 3.68 trillion ton coal resource inventory are classified as "undiscovered" or "hypothetical" [45].

Depth and overburden reporting varies by coalbed, despite the USGS "reliability, coal seam thickness and overburden depth reporting guidelines. Not all coalbed analysis follows the USGS categorization method. Defined ranges are not consistent among the coalbeds. Some reports publish coal quantities per the USGS seam thickness, overburden, and reliability categories. However, other reports define different ranges and reliability for reporting, choose to omit reliability reporting, or ignore the range guidelines altogether. Table 32 summarizes the level of data reporting per seam.

Table 32. Compliance with USGS coal resource reporting criteria

Coal seam name	Overburden depth	Thickness	Reliability categories
Colorado Plateau			
Danforth Hills	λ	ω	λ
Deserado	λ	ω	μ
South Piceance	λ	ω	μ
South Wasatch	λ	ω	μ
Yampa	λ	λ	λ
Henry Mountains	ω	ω	ω
San Juan	λ	λ	λ
Rocky Mountains and Great Plains			
Ashland	λ	λ	λ
Colstrip	λ	λ	λ
Decker	ω	λ	λ
Gillette	λ	λ	λ
Sheridan	λ	λ	λ
Williston-Beulah Zap	λ	λ	λ
Williston-Hagel	λ	λ	λ
Williston-Hansen	λ	λ	λ
Williston-Harmon	λ	λ	λ
Hanna-Ferris 23,25,31,50,65	λ	λ	λ
Hanna-Hanna 7, 78, 79, 81	λ	λ	λ
Carbon-Johnson	λ	λ	λ
Green River-Deadman	λ	λ	λ
Gulf Coast			
Wilcox	λ	λ	λ
Upper Wilcox	λ	λ	λ
Northern and Central Appalachia			
Pittsburgh	λ	λ	μ

Upper Freeport	λ	ω	μ
Lower Kittanning	λ	ω	μ
Pond Creek	λ	ω	μ
Fire Clay	λ	ω	μ
Pocohontas	λ	ω	μ
Illinois Basin			
Springfield	ω	λ	ω
Herrin	ω	λ	ω
Danville	ω	λ	ω

λ = USGS defined categories
ω = Self defined categories
μ = No categories

As shown in Table 32, western coal data adheres to the USGS guidelines, while other resources often include self defined categories. Resources reported in the Rocky Mountains and Great Plains and Colorado Plateau reports follow the USGS categories for categorizing coal depth and thickness. Data categorization is not as consistent in the Illinois and Northern and Central Appalachia reports. The coal in the Colorado Plateau South Piceance seam reported quantities of coal per USGS defined reliability category, but did not further categorize this coal by depth and thickness. To ascertain the amount of coal per reliability category, it was assumed that the ratio of identified to hypothetical resource was constant throughout the coal zone. Coal reliability categories were ignored in the Northern and Central Appalachia resource report. This report also did not tabulate the coal resource per coal thickness and depth; the data was estimated from plots of estimated coal. The Illinois report created their own categories – I-A, I-B and II-C – which are assumed to be the equivalent of “measured”, “indicated” and “hypothetical, although no explicit definition with respect to estimation distance form the borehole is provided [50]. The overburden depth data was not as detailed in the Illinois and Appalachia reports. A maximum measured depth of 1,500 feet was reported for Illinois seams [51]. However, the maximum overburden category provided was 150+ feet [50]. The Kittanning seam in Northern Appalachia reported all of its coal to lay at 700+ feet depths, while the Pocohontas seam reported a total range of overburden depth without categorizing the resource by depth. Depths through 10,000 feet were reported for western seams. The lack of further definition in Illinois and Appalachian resources adds to the uncertainty in its geological profile. While many reports complied with the USG guidelines to describe the coal resource assessed, the discontinuity in reporting categories appears to be arbitrary, with maximum overburden and coal thickness definitions varying throughout. The lack of consistency makes them difficult to compare, and does not lend itself to accurate portrayal of the distribution of coal thickness and depths. Knowing that coal is more than 150 or 700 feet underground does not aid in extraction planning, when it is necessary to consider the true depth of the coal before investing in its development.

Given the discrepancies between all the U.S. coal region reports, it is not surprising that a 2007 National Academies of Sciences (NAS) report on U.S. coal resources [46] stated a range of total coal resources – 268 billion short tons of coal available in the DRB and 493 billion short tons of coal available in the EIA estimated recoverable reserves (ERR).

These estimates are based on the NAS committee analysis of the NCRA and EIA Annual Energy Outlook and Annual Energy Review coal resource and demand analysis. The EIA estimated its ERR estimate on coal unavailable due to surface obstructions and economical extraction [52]. The NAS report criticizes the selective NCRA coverage, and the lack of uncertainty in reported estimates, ultimately questioning the reliability of the data to support a "coherent national energy policy," and whether the reserves are certain enough to provide 1.7 billion tons of coal to meet projected 2030 demand [46]. The lack of uncertainty in data reporting leads to doubts in the applicability of the data for energy planning and resource allocation. The NAS report recognizes that there is a need in more uncertainty to be reported with the coal data in order to develop a better estimate of how long our resources will last. It asserts that it is not possible to confirm that coal resources will last 250 years as conventionally believed, citing a 2006 USGS presentation that claimed less than 50% of identified coal reserves are available to be mined.

Better resolution of coal resources could be obtained if non-producing coal regions were added to the NCRA, and coal producer's resource surveys were accessed. If the latter were publicly available, it would bolster data quality and quantity. Producer surveys are more detailed than USGS and state geological survey analyses [46].

Compilation of distributions

Given the variety of overburden depths and coalbed thickness throughout the country, it is expected that mining cost will vary accordingly. However, cost may be misrepresented if the maximum overburden depth is not accurately interpreted. Assuming a maximum overburden depth of 1,000 feet eliminates some of the uncertainty in estimating the true cost of mining. Because there is no definition of how coal is distributed throughout the given overburden or thickness range, it is not possible to directly estimate mine costs. It is not possible to directly estimate mine costs because specific depth and thickness values are unknown. The maximum depth is not known, and it appears in many cases that it is greatly underreported in order to demonstrate that it is simply a "deep" resource. However, without a known maximum resource depth, it is likely that costs are underestimated due to shallow overburden assumptions or overestimating available resource. By assigning a maximum depth of 1,000 feet in the cases where the maximum is not reported, the total available coal will be overestimated. When the maximum overburden is not reported, it can't be known whether the coal lies at 200 feet or 10,000 feet; it is simply at a depth greater than the maximum value given. By assuming all the reported coal is located at depths less than 1,000 feet, the supply is overestimated. A maximum 1,000 feet depth for Illinois coal is suitable because Illinois coal has been measured at depths up to 1,500 feet [51]. The shortcoming of this approach is the remaining lack of uncertainty in the amount of coal lying between 1,000 and 1,500 feet.

A summary of coal quantities that match DRB requirements and a reduced sample of the DRB are shown in Table 33. The DRB column reports the amount of "measured" and "indicated" coal per coal seam that meets the DRB thickness and depth criteria. In cases where no maximum overburden depth is defined, it is assumed that maximum depth is 1,000 feet. The reduced DRB column displays the quantity of coal per seam above its modal depth. The depth mode is the mean of the overburden category that contains the most coal and is shown in the last column.

A challenge to revising the coal estimate to fit the DRB definition was that the resource is reported in categories that vary by coalbed. For example, the entry level thickness range provided for Louisiana and Texas coal was 1.2 – 2.5 feet. The de minimus thickness falls within this range. Because the data was not further refined, the full amount of coal reported within the range was used. Due to the inability to further reduce the data, the total coal resource estimate assumed here is greater than the NAS estimate. 22,692 million short tons are in known resources 1.2 – 2.3 feet thick. If there were a method to retrieve them, that would be a lot of coal.

Table 33. Demonstrated reserve base coal per coal seam, and subset of this coal per modal overburden

Coalbed	DRB	DRB subset	Overburden mode (feet)
Colorado Plateau Region			
San Juan	24700	24700	1000
Henry Mountains	1100	1100	550
Yampa	1500	1500	1000
South Piceance	7000	90	800
Deserado	280	280	250
Danforth Hills	12100	12100	250-1000
South Wasatch	1200	1200	1000
Rocky Mountains and Great Plains Region			
Carbon-Johnson	840	120	50
Hanna-Ferris 23, 25,31,50,65	320	350	350-1000
Hanna-Hanna 77,78,79,81	1300	1300	1000
Ashland	3700	3700	3700
Colstrip	4800	4800	375
Decker	17400	4100	0
Gilette	59900	59900	750
Sheridan	6100	6100	750
Williston-Beulah-Zap	2700	2700	350
Williston-Hagel	3300	1600	50
Williston-Hansen	2000	5000	350
Williston-Harmon	5400	5000	350
Green River	410	340	350
Gulf Coast Region			
Lower Wilcox	640	320	150
Wilcox	3500	1600	50
Illinois Basin Region			
Danville	13300	10000	325
Herrin	54500	47900	325
Springfield	28300	24500	325
Northern and Central Appalachia Region			
Pittsburgh	11600	2200	100
Upper Freeport	24600	1580	250

Lower Kittaning	26600	26600	1000
Pond Creek	8200	8200	750
Fire Clay	5100	3900	350
Total	332,000	277,000	

Method

The cost model described in Chapter 1 was used to simulate mining in each coalbed. These costs are then scheduled to meet estimated future demand, which is projected according to historical demand. The overburden depth and seam thickness data were input as stochastic distributions to capture the full range of depth and thickness per coalbed. **Table 33** shows the thickness and overburden assumptions for analysis of the full DRB and the DRB subset.

Inputs to Model

As previously mentioned, the NCRA coal resource data was edited and compiled into two groupings for analysis – one that best represented the complete DRB dataset, and a DRB subset that takes advantage of available detailed overburden reporting in order to redefine the maximum overburden to correspond to the depth at which the most coal is located. In order to evaluate the cost to mine coal, the seam characteristics are input to the model. The thickness and overburden per the DRB and DRB subset analysis are shown in Table 34. Thickness remains the same for the DRB and DRB subset; seam thickness does not change with depth in the resources examined. The Danforth Hills, Deserado, Hanna and Ferris coalfields have multiple overlying seams. Individual seam thickness and overburden are shown. Although overburden depths are truncated at 1,000 feet for the total DRB analysis and at the modal overburden for the DRB subset, coal resources thinner than the DRB criteria were not omitted. Mining can end at a set depth, but thin seam thickness data was retained because the variance throughout the seam is not known from looking at the data alone and performing a complete geological survey of the resources is not part of this work. It is assumed that thin seam portions will be mined regardless of their overburden depth, and the additional waste material will be separated in the preparation plant.

Table 34. Seam characteristics for DRB and DRB subset analysis

Coalseam	Thickness, feet (min, mode, max)			DRB overburden, feet (min, mode, max)			DRB Subset overburden, feet (min, max)	
Colorado Plateau Region								
Danforth Hills	2.5	160	410	0	250	1000	0	250
	3.7	210	310	0	250	1000	0	250
	7.5	280	500	0	250	1000	0	250
	3.5	120	250	0	250	1000	0	250
	12	115	195	0	250	1000	0	250
	6	110	230	0	1000	1000	0	1000
	8	130	280	0	1000	1000	0	1000
Deserado	1.2	10.5	14	0	250	1000	0	250
	1.2	10.5	14	0	250	1000	0	250
South Piceance	1	10.5	14	0	800	1000	0	800
South Wasatch	7	14	14	0	1000	1000	0	1000
Yampa	1.2	10.5	14	0	1000	1000	0	1000
Henry Mountains	2	10	10	0	550	1000	0	550
San Juan	1.2	14	14	0	1000	1000	0	1000
Rocky Mountains and Great Plains Region								
Ashland	2.5	25	100	84	1000	1000	84	1000
Colstrip	2.5	15	40	0	375	1000	0	375
Decker	2.5	75	150	0	0	1000	0	0
Gilette	2.5	75	200	0	750	1000	0	750
Sheridan	2.5	75	150	0	750	1000	0	750
Williston-Beulah-Zap	2.5	15	40	0	350	500	0	350
Williston-Hagel	2.5	15	40	0	50	500	0	50
Williston-Hansen	2.5	7.5	40	0	350	500	0	350
Williston-Harmon	2.5	15	40	0	350	500	0	350
Hanna-Ferris 23, 25,31,50,65	2.5	7.5	20	0	1000	1000	0	1000
	2.5	7.5	30	0	350	1000	0	350
	2.5	7.5	30	0	750	1000	0	750
	2.5	15	30	0	1000	1000	0	1000
	2.5	7.5	30	0	750	1000	0	750
Hanna-Hanna 77,78,79,81	5	45	100	0	1000	1000	0	1000
	2.5	35	50	0	1000	1000	0	1000
	2.5	35	40	0	1000	1000	0	1000
	2.5	35	40	0	1000	1000	0	1000
Carbon-Johnson	2.5	40	40	0	50	500	0	50
Green River-Dead Man	2.5	25	40	0	350	1000	0	350
Gulf Coast Region								
Wilcox	1.5	3.75	40	0	50	500	0	50
Lower Wilcox	1.5	3.75	40	0	150	500	0	150
Northern and Central Appalachia Region								
Pittsburgh	1.17	5.25	14	0	100	1000	0	100
Upper Freeport	3.5	7	14	0	250	1000	0	250
Lower Kittaning	1.17	2.89	3.5	700	1000	1000	700	1000
Pond Creek	1.17	4.41	14	0	750	1000	0	750
Fire Clay	1.171	5.25	14	0	350	1000	0	350
Pocohontas	1.17	5.25	14	0	1000	1000	0	1000
Illinois Basin Region								

Springfield	1.2	3	4	0	325	1000	0	325
Herrin	0	3.5	10	0	325	1000	0	325
Danville	1.2	2.8	4	0	325	1000	0	325

Demand Assumptions Based on Historical Demand

In order to construct the supply curves, some notion of demand must be understood. Demand in this case entails the amount of coal required each year. Because an amount of coal that must be supplied is known, coalmines may be scheduled to meet this demand. In this way, we will know how long it will take to deplete a resource, based on necessary demand. Historical demand from 1970 – 2006 is used as the basis to project the future trend of coal use.

The future business as usual demand of coal [53] can be determined as per Figure 3:

$$D_T = -0.147x^2 + 603.03x - 616752 \qquad (21)$$

where D_T = Demand at time T, million tons/year;

x = year

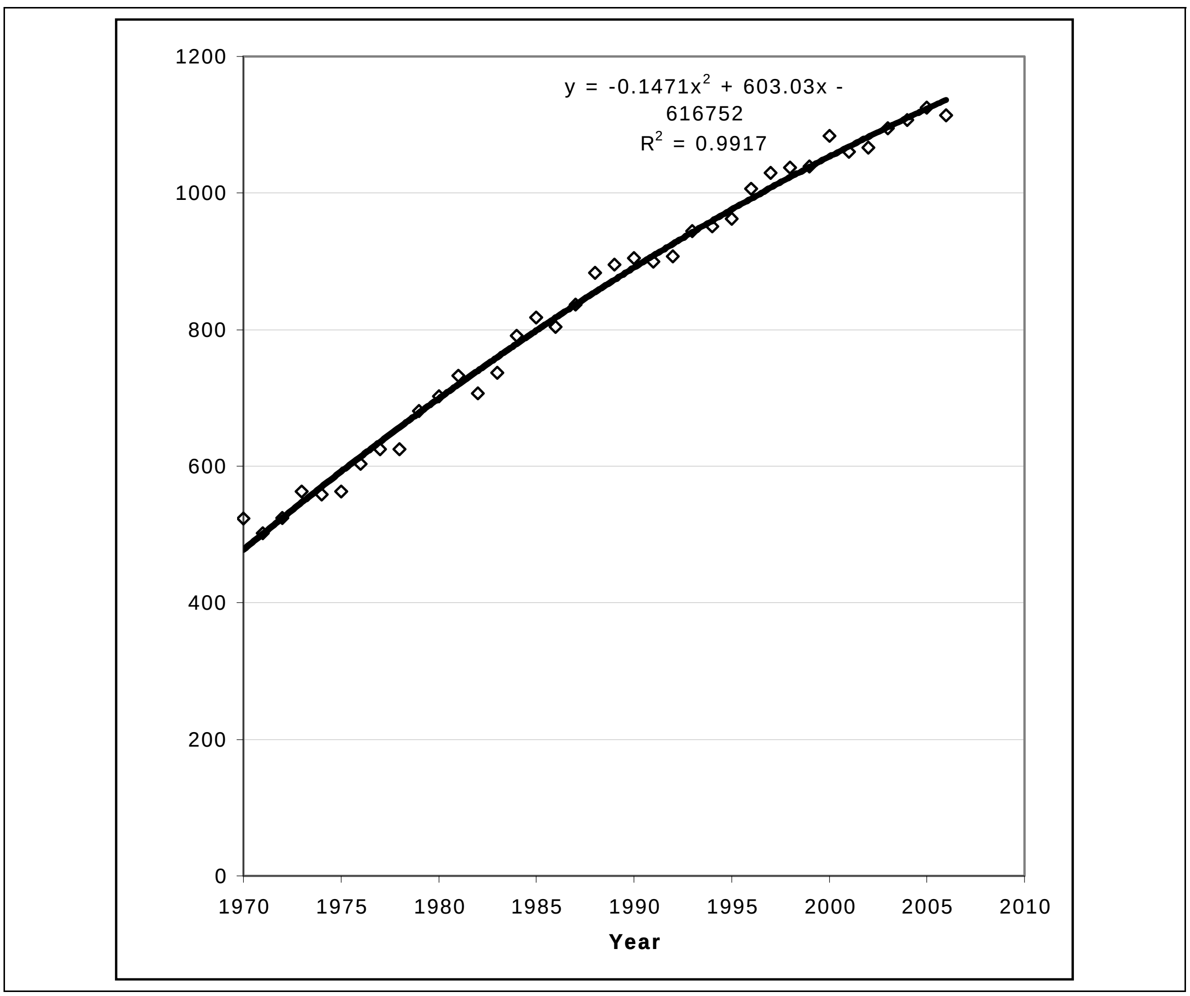

Figure 3. Projected Business as Usual Coal Demand. Source: [53]

Scheduling lowest cost extraction to meet demand

Resource per coal seam is adjusted to reflect the amount of coal that can be recovered by the least cost method:

$$AdjCR_i = r_{i,j} \times CR_i \qquad (22)$$

where $AdjCR_i$ = adjusted resource for coalbed i (million short tons),
$r_{i,j}$ = recovery rate of mine type j in coalbed i (percent),
CR_i = coal resource reported by the USGS NCRA (million short tons)

The coal resource, CR_i, is the amount of coal compiled while defining the total DRB and the DRB subset. The coal seam is then scheduled to meet demand until it is depleted.

Results

Results of Mine Cost Estimation

Model estimated average cost, production rate, recovery rate, and adjusted resource for each of the coal basins are shown in Tables 6 and 7, for total DRB and the DRB subset respectively. Except for the Danforth Hills, Deserado, Hanna, and Ferris coalbeds, which had multiple interlaying seams, it was assumed that all mining methods could be used to mine a given seam. For the four aforementioned coalbeds, it was assumed that surface mining would be used because they contain multiple concave seams that would result in a lot of waste rock if mined by underground methods.

Results according to the grouping of coal resources are shown and compared. Results are shown in order of least cost options. The first column is the name of the coalbed, and the second column is the least cost mining method. Costs ($/Ton), production rates (million short tons/year), and recovery rates are all estimated by the model. Coal resource (million short tons) is the total amount of coal available per coalbed as reported by the USGS NCRA.

Longwall and surface mines have the highest estimated extraction rates, and lowest average mining costs. In the full DRB analysis (Table 6), the seam thickness is the driving factor in determining cost because maximum overburden depth is a constant 1,000 feet for all coal seams. The lowest cost seam is Danforth Hills, which has coal thickness ranging from 2.5 – 410 feet, followed by Gillette, with a coal thickness range of 2.5 – 200 feet. Seams follow in order of descending thickness through Deserado. This coalfield does not have the thinnest seams. However, it has two relatively thin seams with deep interburden (1000 feet). In the DRB subset analysis (Table 34), four of the five lowest cost seams have maximum 50 feet overburden. The exception is the Danforth Hills coalfield, which has multiple thick seams, with 250 feet overburden. Costs are dictated by seam thickness for seams with the majority of their coal under more than 50 feet of overburden. Except for the Decker seam, three of these four least cost mines in this scenario – Carbon-Johnson, Hagel, and Wilcox – are predicted to provide low cost surface mined coal. Except for the Decker seam, when evaluated under the assumption that coal would be mined through their maximum overburden depth, they were predicted to be longwall mined.

When the full DRB is assessed, the shallowest thickest seams are mined first; these seams are predominantly in the Rock Mountains and Great Plains, which houses the Powder River Basin. The Colorado Plateau Coal is the next least costly basin to mine, followed by the Gulf Coast and Northern and Central Appalachia. The Illinois Basin seams are mined last. When the DRB is restricted to the defined subset, mining costs decreased, and some seams that were previously assigned to longwall methods are now most effectively surface mined. These seams are Decker, Carbon-Johnson, Hagel, Wilcox, Pittsburgh, Deserado. When the maximum overburden decreases, the estimated cost decreases. Order of extraction also changed slightly; all basins except Illinois rank among the low cost coal resources. The Illinois Basin is extracted last.

Table 35. Results of Average Cost Estimation per U.S. Coalbed Ranked Least to Highest for DRB Coal

		Cost ($/Ton)			Production (MST/Year)			Recovery Rate				Adjusted Resource (MST)		
Basin	Least Cost Method[a]	0.05	0.5	0.95	0.05	0.5	0.95	0.05	0.5	0.95	Coal Resource (MST)	0.05	0.5	0.95
Danforth Hills	SM	8	13	23	1.9	7.4	20.4	0.99	1.00	1.00	12078	11984	12047	12067
Gilette	LW	10	15	23	13.5	18.6	24.1	0.96	0.95	0.95	59928	57495	56932	56932
Sheridan	LW	10	15	23	13.5	18.6	24.1	0.96	0.95	0.95	6144	5882	5837	5837
Ashland	LW	10	15	26	10.2	18.6	24.1	0.91	0.95	0.95	3723	3381	3537	3537
Carbon-Johnson	LW	11	15	26	9.0	18.5	26.0	0.91	0.95	0.95	835	760	793	793
Williston-Harmon	LW	11	15	26	7.0	17.6	23.4	0.90	0.97	0.95	5362	4851	5209	5094
Williston-Hagel	LW	11	15	30	5.9	16.7	25.4	0.90	0.97	0.95	3314	2979	3226	3148
Williston-Beulah-Zap	LW	11	15	34	6.6	17.0	23.9	0.90	0.96	0.95	2728	2446	2622	2592
Green River-Dead Man	LW	11	15	27	8.5	18.0	26.0	0.91	0.95	0.95	411	373	390	390
Colstrip	LW	11	16	27	6.7	17.6	23.6	0.90	0.97	0.95	4845	4369	4676	4603
Williston-Hansen	LW	11	17	29	6.1	15.8	23.1	0.90	0.93	0.95	2039	1842	1895	1937
Wilcox	LW	11	17	42	3.9	15.2	21.4	0.85	0.92	0.95	3508	2975	3240	3333
South Wasatch	LW	12	17	30	8.7	15.0	20.8	0.91	0.92	0.93	1180	1072	1084	1095
Lower Wilcox	LW	11	19	58	2.7	15.5	23.0	0.85	0.92	0.95	637	541	588	605
Upper Freeport	LW	14	19	46	5.6	10.1	15.6	0.89	0.91	0.92	24560	21909	22393	22682
Yampa	LW	13	20	57	3.0	11.2	18.4	0.85	0.91	0.93	1544.88	1317	1411	1431

Table 35 continued.

		Cost ($/Ton)			Production (MST/Year)			Recovery Rate			Coal Resource (MST)	Adjusted Resource (MST)		
Coalbed	Least Cost Method[a]	0.05	0.5	0.95	0.05	0.5	0.95	0.05	0.5	0.95		0.05	0.5	0.95
San Juan	LW	12	20	53	3.0	12.6	20.3	0.86	0.92	0.93	24673.3	21175	22578	22880
South Piceance	LW	13	20	60	2.5	11.3	17.0	0.82	0.91	0.93	7044	5751	6438	6516
Henry Mountains	LW	14	22	69	3.5	9.2	14.1	0.88	0.91	0.92	1062.9	935	967	982
Decker	SM	8	22	201	0.1	2.5	45.9	0.98	0.99	1.00	17370	17039	17260	17330
Pocohontas	LW	14	24	62	2.0	8.2	16.3	0.81	0.91	0.92	0	0	0	0
Fire Clay	LW	14	24	60	2.4	8.0	15.0	0.82	0.91	0.92	5146	4206	4672	4758
Pittsburgh	LW	14	24	55	3.0	8.6	14.5	0.84	0.91	0.92	11600	9747	10539	10694
Pond Creek	LW	15	24	71	2.2	7.8	17.1	0.82	0.91	0.92	8155	6668	7392	7532
Herrin	LW	16	31	136	1.1	5.7	11.1	0.70	0.90	0.92	54499	37973	48939	50061
Springfield	LW	27	44	106	1.8	3.5	5.4	0.74	0.86	0.91	28343	20952	24395	25848
Danville	LW	27	48	92	1.8	3.3	5.2	0.72	0.86	0.91	13281	9596	11416	12022
Lower Kittaning	LW	28	49	132	1.6	3.3	5.0	0.73	0.85	0.90	26600	19312	22641	23942
Hanna-Ferris 23, 25,31,50,65	SM	15	63	274	0.2	0.8	4.8	0.89	0.95	0.98	317	281	301	311
Hanna-Hanna 77,78,79,81	SM	15	63	197	0.2	0.6	12.1	0.97	0.99	1.00	1294	1253	1281	1289
Deserado	SM	15	108	863	0.0	0.3	3.5	0.84	0.94	0.98	284.48	238	267	278

[a]CM = continuous mining, LW= longwall mining, SM = surface mining

Table 36. Results of Average Cost Estimation per U.S. Coalbed Ranked Least to Highest for DRB Subset

		Cost ($/Ton)			Production (MST/Year)			Recovery Rate			Coal Resource (MST)	Adjusted Resource (MST)		
Basin	Least Cost Method[a]	0.05	0.5	0.95	0.05	0.5	0.95	0.05	0.5	0.95		0.05	0.5	0.95
Decker	SM	5	8	12	7.9	26.6	98.4	0.98	0.99	1.00	4055	3957	4027	4046
Carbon-Johnson	SM	5	9	13	5.5	18.6	66.7	0.95	0.98	0.99	119	113	117	118
Williston-Hagel	SM	6	9	13	5.3	14.4	52.2	0.92	0.97	0.99	1587	1466	1545	1573
Danforth Hills	SM	5	11	22	3.2	11.8	45.1	0.99	1.00	1.00	12078	11971	12046	12064
Wilcox	SM	6	11	17	4.3	13.7	37.7	0.85	0.97	0.99	1593	1358	1540	1578
Pittsburgh	SM	8	14	27	1.3	4.6	20.9	0.78	0.93	0.97	2200	1716	2036	2145
Lower Wilcox	SM	7	14	45	0.7	5.1	41.1	0.84	0.96	0.99	318	269	306	315
Gilette	LW	11	15	24	12.5	18.6	24.4	0.92	0.95	0.95	59928	54930	56932	56932
Sheridan	LW	11	15	24	12.2	18.6	24.4	0.91	0.95	0.95	6144	5618	5837	5837
Ashland	LW	11	15	28	6.2	18.6	24.4	0.90	0.95	0.95	3723	3333	3537	3537
Green River-Dead Man	LW	11	15	34	7.0	18.1	24.4	0.91	0.95	0.95	340	308	323	323
Williston-Beulah-Zap	LW	11	15	43	4.7	17.7	24.4	0.90	0.96	0.95	2728	2446	2622	2592
Colstrip	LW	11	15	40	7.2	17.6	24.3	0.90	0.97	0.95	4845	4368	4679	4603
Williston-Harmon	LW	11	16	47	3.8	17.5	24.4	0.90	0.97	0.95	4975	4501	4833	4726
Williston-Hansen	LW	12	16	50	3.5	16.1	24.1	0.90	0.93	0.95	4975	4493	4623	4726
Deserado	SM	7	16	44	0.6	3.0	24.7	0.86	0.94	0.98	284.48	245	268	277

Table 36, Continued

Coalbed	Least Cost Method[a]	Cost ($/Ton) 0.05	Cost ($/Ton) 0.5	Cost ($/Ton) 0.95	Production (MST/Year) 0.05	Production (MST/Year) 0.5	Production (MST/Year) 0.95	Recovery Rate 0.05	Recovery Rate 0.5	Recovery Rate 0.95	Coal Resource (MST)	Adjusted Resource (MST) 0.05	Adjusted Resource (MST) 0.5	Adjusted Resource (MST) 0.95
South Wasatch	LW	12	17	30	8.7	15.1	19.8	0.90	0.92	0.93	1180	1068	1085	1096
San Juan	LW	13	19	50	2.7	12.8	19.5	0.81	0.92	0.93	24673.3	19882	22579	22890
South Piceance	LW	13	20	85	2.7	11.5	16.9	0.80	0.91	0.93	92	74	84	85
Yampa	LW	13	20	55	2.6	11.1	17.4	0.80	0.91	0.93	1544.88	1235	1411	1429
Upper Freeport	LW	14	21	58	4.2	10.4	16.0	0.89	0.91	0.92	15840	14131	14443	14629
Henry Mountains	LW	15	21	73	3.0	9.6	14.1	0.84	0.91	0.92	1062.9	895	967	982
Fire Clay	LW	14	23	81	2.0	8.3	16.1	0.82	0.91	0.92	3925	3208	3563	3629
Hanna-Ferris 23, 25,31,50,65	SM	10	23	56	0.7	2.4	13.4	0.89	0.95	0.98	347	308	330	340
Pocohontas	LW	15	25	71	1.8	8.8	15.0	0.81	0.91	0.92	0	0	0	0
Pond Creek	LW	14	25	85	2.1	8.1	15.3	0.82	0.91	0.92	8155	6668	7392	7532
Herrin	LW	16	33	192	0.6	5.4	11.0	0.70	0.90	0.92	47851	33340	42969	43954
Hanna-Hanna 77,78,79,81	SM	7	36	169	0.2	1.7	27.2	0.97	0.99	1.00	1294	1255	1281	1289
Springfield	LW	27	46	143	1.7	3.5	5.3	0.74	0.86	0.91	24534	18137	21117	22374
Danville	LW	26	48	140	1.6	3.5	5.1	0.72	0.86	0.91	10042	7255	8632	9090
Lower Kittaning	LW	29	50	149	1.3	3.2	4.6	0.73	0.85	0.90	26600	19312	22641	23942

The costs, starting adjusted coal resource, and production rates reported in Table 35 and Table 36 are used to determine the cost to meet annual projected demand from 2010 – 2110, as shown in Figure 4 and Figure 5. In the first scenario, Rocky Mountain and Great Plains coal will be mined first. In the DRB subset scenario, coal is mined from all regions except the Illinois Basin. In Figure 1, the DRB range of costs for the first 42 years is shown to be 10-23 $/ton with a 50th percentile estimate of $15/ton. All this coal comes from the Colorado Plateau Danforth Hills, Rocky Mountains and Great Plains coal region's Gillette, Sheridan and Ashland coal seams, and Carbon, Williston and Green River coal zones. Cost increases, but the 50th percentile estimate does not reach $20/ton for another 20 years. The 95th percentile estimate increases faster than the 5th or 95th percentile as an artifact of the model, which predicts high 95th percentile costs for longwall mines as seam thickness decreases. However, the results show that over the next 100 years, when the entire DRB is considered, maximum costs could reach $200/ton, but 50th percentile costs are never more than $22/ton. In the final years of the analysis, coal is mined from the Decker seam in the Rocky Mountains and Great Plains region.

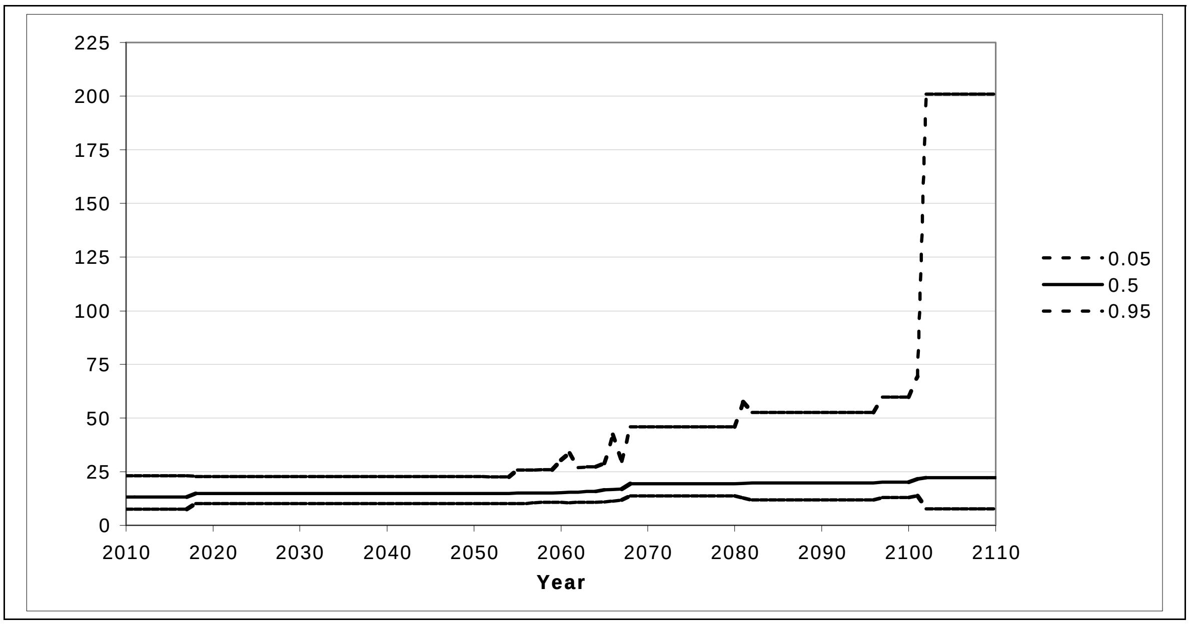

Figure 4. DRB Average Cost Curve Based on Demand Projected from EIA Data.

Figure 5 shows the average cost to mine coal for the DRB subset analyzed. In this scenario, because less coal resource is considered, low cost coal is consumed before the end of the 100 year period. In the final 100 years of the analysis, coal is mined from the Illinois Herrin seam with a cost ranging from $16 - $192/ton, and 50th percentile cost of $33/ton.

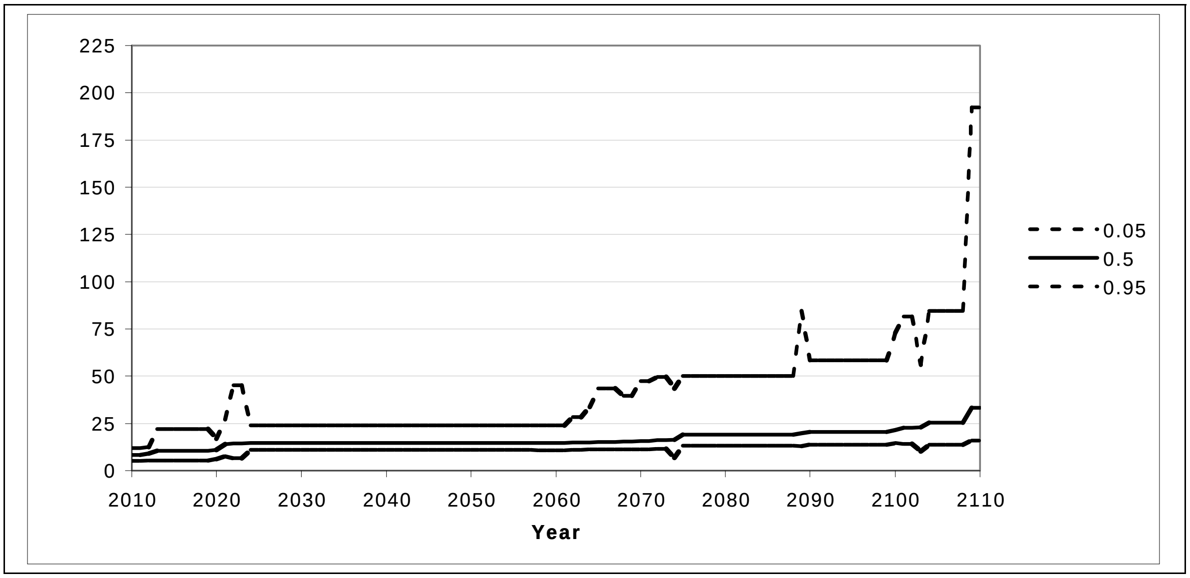

Figure 5. DRB Subset Average Cost Curve based on demand projected from EIA Data.

Discussion

The lack of uncertainty in the existing geological data affected the estimated mining costs. Among the most expensive seams to mine were those for which the least amount of overburden data was available – the Illinois Basin seams, and the Kittanning seam in Northern and Central Appalachia. The Illinois Basin report, which covers the coal available in the Danville, Springfield, and Herrin seams, did not report overburden depths as discretely as reports for western basin coal. The Colorado Plateau and Rocky Mountains and Great Plains coal reports reported maximum overburden depths through 11,000 feet. These reports also contain up to 7 overburden categories. The Illinois Basin report contains 3 overburden categories; these are 0-150 feet and >150 feet. In looking at the raw GIS coded data, one can see that there are additional categories; 200-1000 feet and "undetermined depth". Given the relative lack of quality, it is not possible to accurately represent these seams in the model. Alternately, there is no GIS coded data for the Lower Kittaning coalbed. After reading the NCRA reports for the Illinois Basin and Northern and Central Appalachia Basin, it is apparent that detailed analysis of these coalbeds was not completed because the researchers preconcluded that these seams would be expensive to mine. However, in order to more accurately estimate mining costs, more geological data is needed.

The total projected coal demand over the 100 year time period is 157 billion short tons. This demand is met, with 175 billion short tons of the total DRB and 120 billion short tons of the examined DRB subset remaining. At the rate of coal demand, coal resource will be exhausted in less than 250 years. Cost to mine the remaining coal will be more expensive. To estimate the mining cost for the remaining resource, more geological data is needed. Furthermore, it is necessary to measure all coal resources in all regions, not just those that are currently mined, in order to gain insight into the reliability of coal to provide energy for the long term. Our current estimated DRB will not last as long as is commonly believed.

Environmental Costs

Chapter 1 showed how the mining cost model was used to generate a lowest average cost curve for the National Coal Resource Assessment. The model's cost estimating ability will be expanded in this chapter to include environmental costs. The costs will be estimated following two methods. The first is a damage cost assessment – the magnitude of expected environmental impacts will be determined, and associated costs will be assessed. These costs will be estimated according to the "value" placed on the pollution or damage. The second method is to estimate the cost to prevent the environmental damage. This approach will examine the cost to implement a technology to mitigate environmental damage.

The process model is used to estimate the magnitude of environmental damages and the material needed to mitigate impacts. The mine workings, volumes, surface area can all be estimated by the model. The environmental costs are added to the model's calculated cost to obtain a total cost.

Environmental damage description

Coal mining releases pollutants to air, water, and soil, and can affect water availability, and ground stability. I will discuss these problems in terms of surface and underground mining. Both methods break the earth and result in soil degradation from aerating soils; both can affect surface water availability and quality, local air quality, and create large amounts of solid waste. My qualifier paper deals with acid mine drainage and subsidence, prevalent problems in underground mining in Appalachia.

Some of these problems, such as acid mine drainage and coal slurry discharge impoundments, must be perpetually managed for many years (possibly decades!) after the mine closes.

Environmental impacts are estimated by using techniques in the literature, which consist of rule-of-thumb calculations, and estimation factors based on fieldwork measurements.

Land impact footprint is estimated by calculating the total surface pit area, or area and depth of subsidence due to underground mining.

Water use is estimated by the amount of water used, based on water use factors.

Solid waste – not sure.

Energy consumption is estimated by using the model's current estimation of diesel and electricity use on an annual basis. Criteria and greenhouse gas emissions are calculated as a function of energy use. Methane emissions as a result of cutting coal and storing waste heaps will also be estimated.

Dust will not be estimated, because there is not enough data available.

Prior Work

Misiolek and Noser [54] estimate reclamation costs for surface mines ($/acre mined) in states that produced more than four million tons of bituminous and/or subbituminous coal in 1979. The study considers expected soil replacement, sales revenues, and the depreciation of equipment. Methods followed for depreciation calculations follow standards set by the Society of Mining Engineers, and industry. Mine and equipment lifetimes are not explicitly stated. Reclamation costs are determined as a function of overburden depth and estimated acreage mined. The authors find that reclamation costs range from $6500 - $8000/acre throughout the U.S. (1980 dollars).

Randall et al. [55] estimates the benefits to be gained by complying with Kentucky mining regulations and the Surface Mining Control and Reclamation Act. The results are presented as benefits derived from state regulations and from federal regulations. The cost of reclamation is extended beyond simple consideration of soil replacement and equipment costs. It evaluates acid mine treatment, costs of restocking game fish, lost water recreation as estimated by the Water Resources Council, increased flooding, damage to land and buildings, as well as aesthetic damages as valued by Kentucky residents. The results in 1976 dollars, are that state regulations offer benefits of $0.81 per ton of coal mined, and federal regulations result in benefits of $1.72 per ton of coal mined.

Land Impact

South African study has some land use estimates for collieries [56]

The surface footprint of surface and underground mining is estimated. In surface mines, this footprint is assumed equivalent to the pit mining area. In underground mines, the model assumes the surface footprint equals the subsidence area. After mining, surface pits are filled in and graded to an "approximate original contour". This approach is suitable in flat regions of the country. However, in Appalachia, where surface mining in mountains is nicknamed "mountain top removal" achieving the "approximate original contour" is impossible. The pit area is therefore the damaged land area from the mining activity. During underground mining, a seam of coal is extracted, thus removing a layer of earth. The overlying layers fall, causing subsidence. This can pose a safety hazard to overlying roads and buildings. It can also limit future growth, if building on land that overlies an area made unstable by underground methods.

Subsidence estimation

There are several accepted subsidence estimation methods. Typically, these were developed specifically for a region or coal seam, with empirically based subsidence factors. The subsidence factor may be estimated according to rock properties [7]. It is also possible to generate an approximate subsidence factor from empirical measurement of subsidence over time. There are several references that provide subsidence factors for various regions of the country [57], [7], and [58]. There are methods available to estimate the subsidence time lapse. However, these methods require field measurements and complicated mathematical modeling. For simplicity, and to use the limited information available, final total subsidence is estimated instead.

Subsidence for underground mining is estimated according to the size of the mine workings. In continuous mines, this consists of subsidence over rooms and pillar workings, and in longwall mines it is the longwall panels and the development sections. It is expected that subsidence area will be greater in longwall mines than continuous mines for the same pod. An empirical-based approach, applicable throughout the country [59] is used to estimate subsidence area and depth. This method estimates subsidence area and maximum subsidence depth as a function of overburden depth, seam height, panel width and pillar width. It estimates the subsidence factor, offset distance of inflection point, and major influence radius. These variables are shown in Figure 6 – Figure 8 and are estimated:

$$a = 1.9381(h + 23.4185)^{-0.1884}$$

$$d = h(0.382075 \times 0.999253^{h})$$

$$R = \frac{h}{\tan\beta}$$

where a = subsidence factor
h = overburden depth
d = offset distance of inflection point
R = radius of major influence or angle of major influence
$\tan\beta = 3$

Continuous mining subsidence variables are calculated:

$$a = \rho(0.7247 - 2.4733 \times 10^{-5} h = 1.9585 \times 10^{-7} h^{2})$$

$$d = h(0.380275 \times 0.999253^{h})$$

$$R = \frac{h}{\sqrt{\rho}\tan\beta}$$

where ρ = mine recovery ratio

The overburden depth, h, is input per each NCRA coal region as described in Chapter 2. The continuous mine recovery ratio, ρ, is estimated by the model as described in Chapter 1. For both underground mine types, maximum subsidence depth is calculated:

$$S_{\max} = a \times m$$

where m = mining height

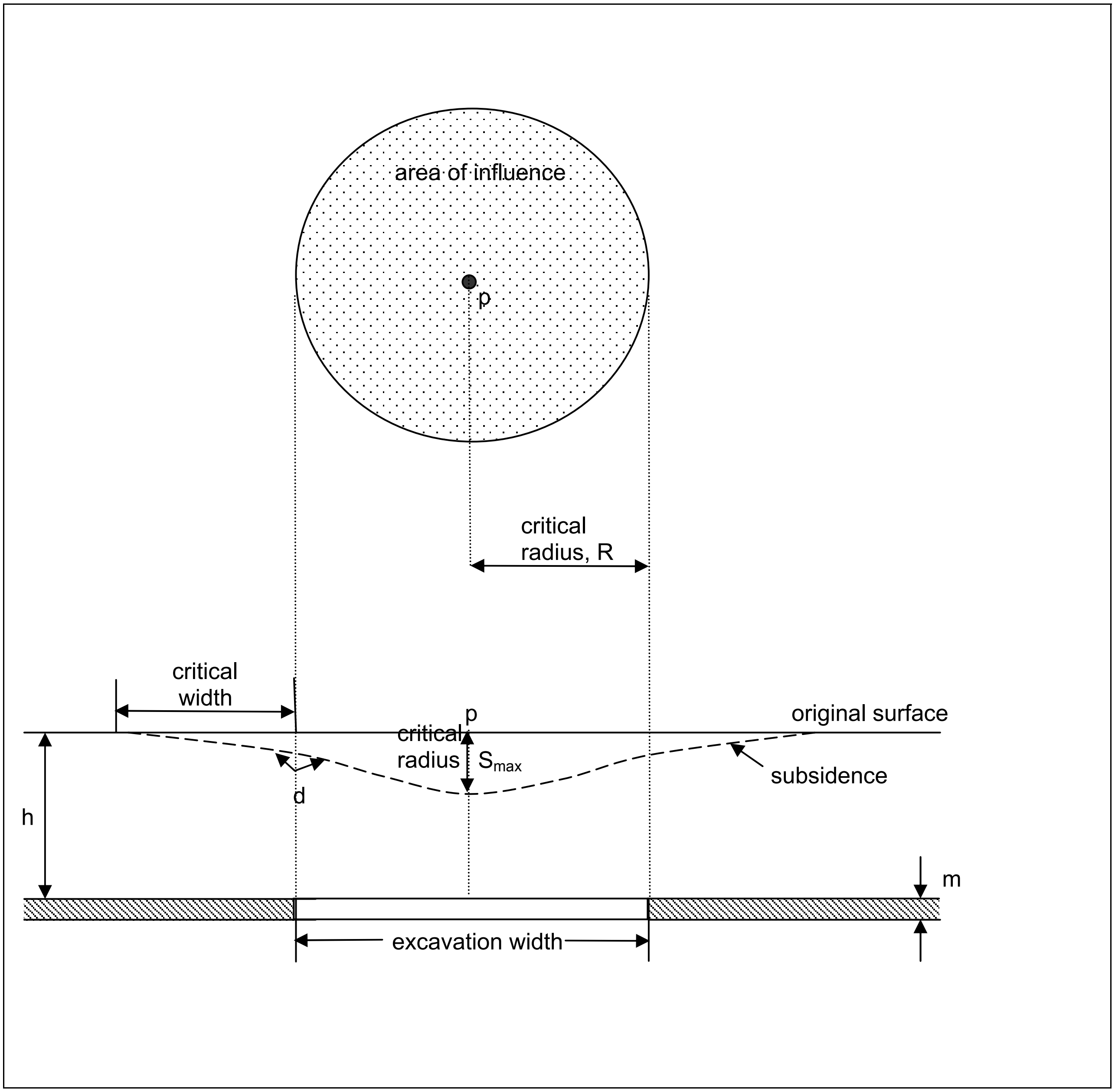

Figure 6. Subsidence variables. Diagram not to scale.

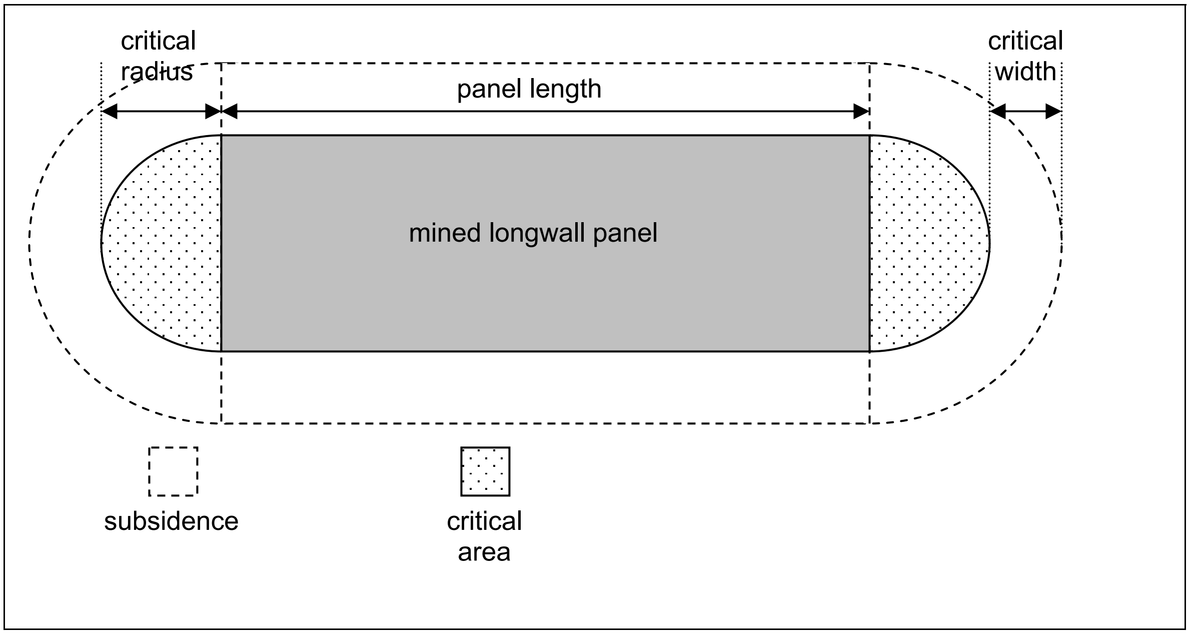

Figure 7. Longwall subsidence variables

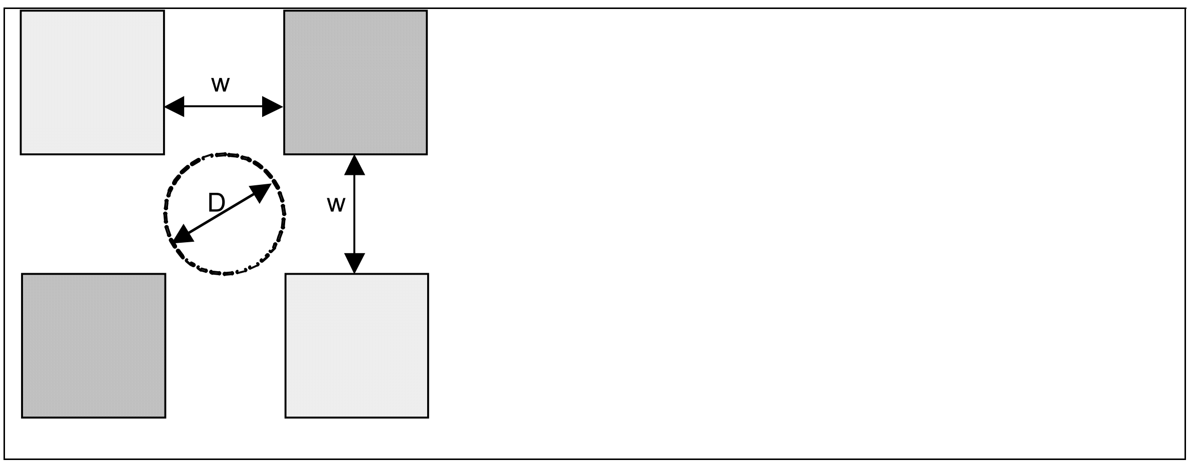

Figure 8. Continuous mine subsidence variables

As shown in Figure 8, for continuous mines and continuous mining support sections in longwall mines, it is assumed that chimney sinkholes will form with a diameter that ranges from w to w*sqrt(2) [57]. The number of these sinkholes is estimated as a function of the number of pillars left underground. It is assumed that these sinkholes will form where entries intersect and there is no adjacent coal to support the roof. The area of subsidence over longwall panels is determined by estimating the length and width of expected subsidence, based on panel dimensions, critical width, and critical radius. Although the diagram shows critical width as being half of the panel width, this may not always be the case, depending on how deep the seam lies.

Calculated maximum subsidence depth is shown in Table 37, and subsidence area per total lifetime production is shown in Table 38.

Table 37. Estimated maximum subsidence per underground mine type and NCRA coalfield (feet)

Region	Coalfield	Longwall Mine			Continuous Mine		
		0.05	0.5	0.95	0.05	0.5	0.95
Colorado Plateau	South Piceance	2.96	7.30	10.2	1.16	4.13	6.32
	South Wasatch	7.12	9.85	11.4	4.06	5.42	7.07
	Yampa	3.09	7.35	10.3	1.15	4.16	6.30
	Henry Mountains	3.19	6.31	8.1	1.33	3.83	5.42
	San Juan	3.45	8.45	11.3	1.22	4.55	7.07
Rocky Mountains and Great Plains	Ashland	11.07	32.59	66.0	5.64	17.90	36.61
	Colstrip	6.23	15.13	27.1	5.06	11.85	20.51
	Decker	21.96	62.35	103.4	14.32	43.21	70.72
	Gilette	25.08	73.29	134.7	15.17	42.14	85.68
	Sheridan	21.96	62.35	103.4	14.21	36.58	63.17
	Williston-Beulah-Zap	6.23	15.13	27.09	4.25	11.00	19.91
	Williston-Hagel	6.23	15.13	27.09	4.52	11.50	20.20
	Williston-Hansen	4.70	12.63	26.26	3.09	8.46	18.59
	Williston-Harmon	6.23	15.13	27.09	4.06	10.17	19.81
	Carbon-Johnson	9.28	23.92	32.12	7.04	17.69	25.60
	Green River-Dead Man	7.65	18.99	28.41	4.80	12.29	17.81
Gulf Coast	Wilcox	3.03	11.20	25.79	1.87	8.73	19.12
	Lower Wilcox	3.03	11.20	25.79	1.58	8.42	18.98
Appalachia	Pittsburgh	2.36	5.37	9.50	1.15	3.36	6.57
	Upper Freeport	4.05	6.54	9.89	1.84	4.18	6.96
	Lower Kittaning	1.35	2.13	2.66	0.42	0.66	0.85
	Pond Creek	2.20	5.08	9.41	0.87	2.96	5.42
	Fire Clay	2.36	5.37	9.50	0.99	3.35	6.11
	Pocohontas	2.36	5.37	9.50	0.82	2.99	6.08
Illinois	Springfield	1.42	2.30	2.98	0.55	1.07	1.65
	Herrin	1.14	3.54	6.69	0.41	1.64	4.26
	Danville	1.40	2.22	2.95	0.56	0.99	1.78

Table 38. Estimated subsidence area per underground mine method and NCRA coal region (ft^2/ton coal produced)

Region	Coalfield	Longwall Mine			Continuous Mine		
		0.05	0.5	0.95	0.05	0.5	0.95
Colorado Plateau	South Piceance	0.92	0.72	2.43	0.00	0.01	0.02
	South Wasatch	NAN	0.45	1.20	0.00	0.01	0.01
	Yampa	0.17	0.68	2.96	0.00	0.01	0.02
	Henry Mountains	0.13	0.73	2.37	0.00	0.01	0.02
	San Juan	0.59	0.62	1.96	0.00	0.01	0.01
Rocky Mountains and Great Plains	Ashland	0.61	0.40	1.04	0.00	0.00	0.01
	Colstrip	0.05	0.27	0.78	0.01	0.01	0.02
	Decker	0.43	0.26	0.81	0.00	0.01	0.02
	Gilette	0.05	0.31	0.83	0.00	0.00	0.01
	Sheridan	0.05	0.35	0.86	0.00	0.00	0.01
	Williston-Beulah-Zap	0.55	0.09	0.86	0.00	0.01	0.01
	Williston-Hagel	0.53	0.29	0.72	0.00	0.01	0.02
	Williston-Hansen	0.48	0.44	1.27	0.00	0.01	0.02
	Williston-Harmon	0.34	0.40	0.76	0.00	0.01	0.01
	Carbon-Johnson	0.05	0.35	0.69	0.00	0.01	0.02
	Green River-Dead Man	0.25	0.07	0.99	0.00	0.01	0.01
Gulf Coast	Wilcox	0.09	0.44	1.43	0.00	0.01	0.03
	Lower Wilcox	NAN	0.38	2.01	0.00	0.01	0.02
Appalachia	Pittsburgh	0.56	0.58	2.29	0.00	0.01	0.02
	Upper Freeport	0.95	0.60	1.61	0.00	0.01	0.02
	Lower Kittaning	3.77	2.10	6.60	0.00	0.01	0.02
	Pond Creek	1.84	1.01	2.80	0.00	0.01	0.02
	Fire Clay	0.92	0.74	2.20	0.00	0.01	0.02
	Pocohontas	0.79	0.53	2.84	0.00	0.01	0.01
Illinois	Springfield	3.76	2.10	5.01	0.01	0.01	0.04
	Herrin	1.74	0.17	4.33	0.00	0.01	0.03
	Danville	1.00	0.40	4.55	0.01	0.01	0.04

Surface pit area

As described in Chapter 1, the model estimates surface pit area. The total area of all mined pits is:

$$S_f = n_{pit} \times A_{pit}$$

where S_f = surface mine footprint

n_{pit} = number of pits mined
A_{pit} = pit area

Calculated land area per ton of coal produced is shown in Table 39.

Table 39. Surface mine land impact per NCRA coal region(ft^2/ton coal produced)

Region	Coalfield	0.05	0.5	0.95
Colorado Plateau	Danforth Hills	7.0E-03	6.7E-03	9.7E-03
	Deserado	0.7	0.4	0.4
	South Piceance	4.5	2.7	2.0
	South Wasatch	1.7	2.8	2.1
	Yampa	3.2	3.1	2.3
	Henry Mountains	4.9	3.5	3.8
	San Juan	2.5	2.4	2.6
Rocky Mountains and Great Plains	Ashland	1.0	0.8	0.5
	Colstrip	1.8	1.4	1.1
	Decker	0.7	0.3	0.2
	Gilette	0.4	0.3	0.2
	Sheridan	0.4	0.3	0.4
	Williston-Beulah-Zap	1.9	1.2	1.8
	Williston-Hagel	2.0	1.1	0.9
	Williston-Hansen	3.1	1.7	1.3
	Williston-Harmon	2.0	1.3	1.6
	Hanna-Ferris 23, 25,31,50,65	0.1	0.0	0.0
	Hanna-Hanna 77,78,79,81	0.2	0.2	0.1
	Carbon-Johnson	1.0	0.9	0.7
	Green River-Dead Man	1.6	1.1	1.0
Gulf Coast	Wilcox	2.3	2.0	1.7
	Lower Wilcox	2.7	1.8	1.8
Appalachia	Pittsburgh	2.5	3.8	5.2
	Upper Freeport	3.4	3.0	1.9
	Lower Kittaning	8.4	9.5	11.3
	Pond Creek	5.0	4.9	4.6
	Fire Clay	3.9	3.7	3.9
	Pocohontas	3.8	4.4	1.5
Illinois	Springfield	8.9	9.1	7.8
	Herrin	6.1	6.1	3.9
	Danville	13.4	7.2	7.4

Water Consumption

To estimate the amount of water used in a mine, the water volume and pressure needed to operate sprays and scrubbers, maintain and wash equipment, suppress dust, and provide shower and supply the support facilities must be quantified. Some water use rules of thumb are available for a few pieces of underground mining units[4] and surface dust control [60]; continuous mining units use 10 – 40 gallons/minute at working pressure of 200 – 300 psi, longwall shearers need 60 – 120 gallons/minute at 200 – 300 psi, belt lines use 5 – 10 gallons/minute, and dust control requires 5.2 gallons per ton of coal mined. However, a thorough water use estimate can't be calculated because extensive equipment water use data is not available. Instead, overall water consumption estimation factors are used: 344 acre-ft/million tons of underground coal mined, and 204 acre-ft/million tons of surface mined coal [61]. Based on these estimation factors, 0.1 gallons of water is consumed per ton of coal mined in the United States.

Acid Mine Drainage

Water quality can be affected by acid mine drainage, and runoff from pilings. Acid is formed when water and air contact sulfuric bearing rock in or around the mine. The acid can be formed in the mine void, and when discharged is called "acid mine drainage". A piling is a pile of waste material, which consists of low quality coal, dirt, and rocks, and provides another means of acid formation and runoff.

Site specific conditions necessitate site specific prediction. Detailed information needed to estimate potential acid mine drainage is not available. Even the experts acknowledge that predicting acid mine drainage is difficult. "prediction of base-levels of groundwater drainage is a complicated exercise." [62]. A few of the things that must be available in order to predict AMD are outcrop exposure measurements, drillhole logs, geological sections and core assays [63]. Although outcrop data is available for some of the National Coal Resource Assessment, an analysis of estimated potential distance from possible mines to outcrop is beyond the scope of this analysis. To predict acid mine drainage, EPA recommends collecting samples and determining acid generation potential from them [63]. The samples are to be drill samples collecting during mine planning. The two predominant methods of acid generation potential are static and kinetic testing. Static testing is a calculation based on the sulfur content, assuming complete reaction that produces two moles of acid for each mole of sulfur:

$$APP_{MAX} = \%S \times cf$$

where APP_{MAX} = maximum acid production potential, ton acid per ton rock
%S = percent of sulfur in coal
cf = conversion factor, 31.25

This calculation can be taken further with the maximum neutralization potential, or amount of carbon material needed to neutralize the sample. If the acid production potential is greater than the neutralization potential, then the material is acid producing. This rule of thumb assumes that all sulfur in the coal is converted to sulfuric acid; it is assumes the worst case, that all available sulfur will be exposed to water and oxygen to form acid. A shortcoming to choosing the static test is that metals release can't be estimated. The static test only estimates acid generation potential.

The kinetic test consists of a laboratory simulation of the acid reaction over time. Reaction rates and effluent concentrations are measured. These measurements are used as an empirical basis to estimate future acid formation. This method is time consuming.

In this report, the maximum acid production potential is calculated and used as a metric of mining impact on water quality. The NCRA reported regional coal sulfur content used to estimate the maximum acid production potential is shown in Table 40m and calculated maximum acid production potential is shown in Table 5.

Table 40. Arithmetic mean sulfur content in U.S. coal [64]

Region	Percent Sulfur
Powder River Basin	0.48
Williston	0.84
Colorado Plateau	0.83
Gulf Coast	1.09
Illinois Basin	3.55
Appalachian Basin	2.14

Table 41. Assumed sulfur content and acid production potential per NCRA coal region (tons acid/ton coal)

Region	Coalfield	Percent Sulfur	Acid Production Potential
Colorado Plateau	Danforth Hills	0.83	26
	Deserado	0.83	26
	South Piceance	0.83	26
	South Wasatch	0.83	26
	Yampa	0.83	26
	Henry Mountains	0.83	26
	San Juan	0.83	26
Rocky Mountains and Great Plains	Ashland	0.48	15
	Colstrip	0.48	15
	Decker	0.48	15
	Gilette	0.48	15
	Sheridan	0.48	15
	Williston-Beulah-Zap	0.48	15
	Williston-Hagel	0.48	15
	Williston-Hansen	0.48	15
	Williston-Harmon	0.48	15
	Hanna-Ferris 23, 25,31,50,65	0.48	15
	Hanna-Hanna 77,78,79,81	0.48	15
	Carbon-Johnson	0.48	15
	Green River-Dead Man	0.48	15
Gulf Coast	Wilcox	1.09	34
	Lower Wilcox	1.09	34
Appalachia	Pittsburgh	2.14	67
	Upper Freeport	2.14	67
	Lower Kittaning	2.14	67
	Pond Creek	2.14	67
	Fire Clay	2.14	67
	Pocohontas	2.14	67
Illinois	Springfield	3.55	111
	Herrin	3.55	111
	Danville	3.55	111

A more complex approach to model would be to estimate the rate of water recharge to the coal seam, and kinetic rate of acid formation and metals release. Studies of acid mine drainage in western Pennsylvania draw analogies between flow through flooded and leaking abandoned coal mines [65] [66], accounting for inlet water chemistry composition based on overburden qualities [67]. To apply this approach, more detailed information about rain fall, overburden composition and kinetic rates must be known for all the coal seams in this analysis. This information is not available on a widespread basis, and the computational work to estimate acid mine drainage using this method is beyond the scope of this analysis.

Solid Waste

Solid waste from coal mining includes infrastructure such as pipe and wires for mine support, as well as waste coal which includes low quality coal, rock, and dirt that is extracted with the coal that is mined. Waste, in itself, does not pose an environmental problem. However, current mine waste storage methods can be a concern. Impoundments, which store waste materials from mining pose a safety and environmental risk. If the impoundment wall fails, the local area can be flooded with water contaminated with wastes from the mine [68-71]. Refer to ICOLD, 2001, Margin and Davies, 2000; USCOLD, 1994; Vick 2000. Estimated failure rate for tailings dams is not available in the literature.

The model calculates waste rock created during mining. It does not include an analysis of consumed equipment, pipe, or electrical components because this kind of data is not available. Additionally, it is the storage or disposal of waste rock that is controversial. This waste is typically stored in impoundments that are not fail proof, or is pushed into valleys from mountain mining operations and is more familiarly known as “valley” fill” accompanying “mountain top removal.” Looking at the mining process represented by the model, waste rock is collected from the coal cleaning process – partings within the coal and ceiling or floor harvested in the coal cut must be separated and disposed of; surface mine spoil, which is the amount of overburden that will not fit back into the exhumed pit due to expansion; and material generated from underground shaft and hoist digging. The solid waste generated per mine type is assumed to be:

Longwall: Roof cut by longwall and continuous miner and partings
Ventilation shaft material
Hoist shaft material

Continuous: Roof cut by continuous miner and partings
Ventilation shaft material
Hoist shaft material

Surface: $T_{spoil} = \rho_{spoil}(V_{pit} - V_{OB}e_f)$

T_{spoil} = tons of spoil
ρ_{spoil} = soil density, 1.4 – 2.1 tons/yd^3 [4], chosen to represent all overburden component possibilities (limestone, sand, gravel, sandstone, clay, chalk, and shales which may be in overburden)
V_{pit} = pit volume
V_{OB} = extracted overburden volume

e_f = overburden swell factor, 1.15 – 1.65 [4], chosen to represent all overburden component possibilities

The calculated waste rock generated by each mine type and NCRA region is shown in Table 42.

Table 42. Estimated solid waste generated per mine type and NCRA coalfield (ton/ton coal produced)

Region	Coalfield	Longwall			Continuous			Surface		
		0.05	0.5	0.95	0.05	0.5	0.95	0.05	0.5	0.95
Colorado Plateau	Danforth Hills							4.E+03	7.E+04	5.E+05
	Deserado							7.E+03	4.E+05	6.E+06
	South Piceance	0.01	0.02	0.39	0.08	0.13	0.42	1.E+04	5.E+05	1.E+07
	South Wasatch	0.01	0.02	0.07	0.07	0.09	0.14	8.E+03	5.E+05	8.E+06
	Yampa	0.01	0.02	0.15	0.08	0.12	0.32	2.E+04	6.E+05	1.E+07
	Henry Mountains	0.01	0.03	0.13	0.09	0.15	0.26	7.E+03	3.E+05	7.E+06
	San Juan	0.01	0.02	0.13	0.07	0.11	0.26	8.E+03	8.E+05	1.E+07
Rocky Mountains and Great Plains	Ashland	0.03	0.34	1.63	0.00	0.00	0.10	5.E+03	2.E+05	3.E+06
	Colstrip	0.01	0.12	0.87	0.00	0.00	0.16	1.E+03	6.E+04	3.E+06
	Decker	0.15	0.44	2.01	0.00	0.00	0.01	7.E+01	6.E+03	2.E+05
	Gilette	0.17	0.47	1.77	0.00	0.00	0.01	1.E+02	3.E+03	1.E+05
	Sheridan	0.15	0.43	1.61	0.00	0.00	0.01	9.E+01	4.E+03	9.E+04
	Williston-Beulah-Zap	0.01	0.12	0.64	0.00	0.00	0.15	3.E+03	2.E+05	7.E+06
	Williston-Hagel	0.01	0.12	0.84	0.00	0.00	0.16	2.E+02	2.E+04	3.E+05
	Williston-Hansen	0.01	0.06	1.35	0.00	0.06	0.20	1.E+02	4.E+03	1.E+05
	Williston-Harmon	0.01	0.11	0.62	0.00	0.00	0.15	4.E+02	2.E+04	3.E+05
	Hanna-Ferris 23, 25,31,50,65							1.E+05	2.E+06	1.E+07
	Hanna-Hanna 77,78,79,81							1.E+03	5.E+04	5.E+05
	Carbon-Johnson	0.02	0.27	1.39	0.00	0.00	0.11	1.E+02	3.E+03	8.E+04
	Green River-Dead Man	0.01	0.17	0.70	0.00	0.00	0.13	8.E+02	7.E+04	2.E+06
Gulf Coast	Wilcox	0.01	0.05	0.76	0.00	0.08	0.31	3.E+01	5.E+03	3.E+05
	Lower Wilcox	0.01	0.06	0.80	0.00	0.07	0.29	3.E+02	9.E+03	2.E+05
Appalachia	Pittsburgh	0.01	0.03	0.12	0.09	0.17	0.40	4.E+02	7.E+04	5.E+06
	Upper Freeport	0.01	0.03	0.09	0.08	0.14	0.22	3.E+03	1.E+05	5.E+06
	Lower Kittaning	0.04	0.08	0.64	0.16	0.40	0.81	6.E+05	1.E+07	6.E+07
	Pond Creek	0.01	0.03	0.28	0.09	0.18	0.45	2.E+04	7.E+05	1.E+07

	Fire Clay	0.01	0.03	0.18	0.10	0.16	0.36	4.E+03	2.E+05	7.E+06
	Pocohontas	0.01	0.03	0.25	0.08	0.17	0.37	1.E+04	1.E+06	2.E+07
Illinois	Springfield	0.03	0.08	0.42	0.18	0.37	0.69	8.E+03	6.E+05	9.E+06
	Herrin	0.02	0.05	2.93	0.11	0.25	1.43	2.E+03	4.E+05	1.E+07
	Danville	0.03	0.08	0.86	0.14	0.39	0.80	4.E+03	5.E+05	1.E+07

Methane emissions

Methane is emitted throughout the mining process. It is released from the coal when it is broken. During surface mining operations, it is released to the atmosphere as the coal is cut and mined. During underground mining, it is released during development of the coal mine and via safety ventilation during mining. Methane emissions are estimated by using EPA methane emissions factors.

The U.S. EPA estimated methane emissions from coal mining in several reports [76] [77]. In their 2005 report, to estimate methane emissions, they used measured emissions data from underground mines and in situ coal quality data for surface mines. Quarterly MSHA safety measurements were the basis for their operating underground estimate. The dataset runs 1990-2003, excepting for 1997. Basin emissions factors for surface mining operations are based on in-situ methane content in coals. EPA assumes emissions factors are twice the in-situ content, but in the 1993 assessment, the assumed emissions factors to be three times the in situ content. Post mining emission factors are assumed to be 25-40% with a mean 32.5% in situ. The report does not explain why there is a difference between 1993 and 2003 emissions factors. Resulting emissions from surface mines are calculated:

$$Emissions = EF \times production \qquad (23)$$

where Emissions = methane emissions from mine in region i

EF = emissions factor of coal in region i

Production = production rate of mine in region i

Emissions factors are given in the 2005 report and summarized in Table 43. There are fourteen methane regions defined, that do not completely overlap the NCRA coal regions. Inclusion of coalfields within the methane regions was defined in order to apply the EPA emissions factors to the NCRA coalfields. The Northern Appalachia basin contains the Pittsburgh, Upper Freeport and Lower Kittanning coalfields while the Central Appalachia basin has the Fire Clay, Pond Creek and Pocohontas coalfields [78]. There is no NCRA coal data provided for the Warrior basin. The coalfields in the Illinois basin correspond to those defined by the NCRA. The Northern Rocky Mountains and Great Plains defined by the NCRA is assumed to be the same as the EPA defined "Northern Great Plains" except for Green River, which has its own assessment within the EPA defined "Rockies." The Rockies (Piceance Basin) is assumed to house the Danforth and South Piceance coalfields although the latter is found in both the Piceance and Uinta Basins. The Rockies (Uinta Basin) is assumed to house the South Wasatch, Henry Mountains, Deserado and Yampa coalfields. Although the Henry Mountains are not technically in the Uinta Basin, it is located close enough that it is assumed that it has the same methane quality. The San Juan coalfield is assumed to be in the Rockies (San Juan Basin), and no coal data is available for the West Interior or Northwest. Emissions factors as assigned are shown in Table 11.

EPA [77] has gob data and degasification data from MSHA.

Table 43. Coal surface and post-mining methane emissions factors [76] [79]

	Emissions Factors, ft^3/ton			
Basin	**Surface mine**	**Post-mining surface mine**	**Underground mine**[a]	**Post-mining underground mine**
Northern Appalachia	119.0	19.3	87.8	14.0
Central Appalachia (WV)	49.8	8.1	88.6	44.5
Central Appalachia (VA)	49.8	8.1	88.6	129.7
Central Appalachia (E KY)	49.8	8.1	88.6	20.0
Warrior	61.4	10.0	173.5	86.7
Illinois	68.6	11.1	45.2	20.9
Rockies (Piceance Basin)	66.2	10.8	76.22	63.8
Rockies (Uinta Basin)	32.0	5.2	76.22	32.3
Rockies (San Juan Basin)	14.6	2.4	76.22	34.1
Rockies (Green River Basin)	66.2	10.8	76.22	41.6
Rockies (Raton Basin)	66.2	10.8	76.22	41.6
Northern Great Plains	11.2	1.8	76.22	5.1
West Interior (Forest City, Cherokee Basins)	68.6	11.1	0	20.9
West Interior (Arkoma Basin)	149.0	24.2	0	107.6
West Interior (Gulf Coast Basin)	66.2	10.8	NA[b]	41.6
Northwest (AK)	11.2	1.8	NA	52.0
Northwest (WA)	11.2	1.8	NA	18.9

[a]Calculated from 1995 methane emissions and production data [79] and assuming methane density of 47,000 ft^3/ton [80]. The Rockies and Northern Great Plains coal basins are assumed to be in the "Western Coal Fields" region. The estimated overall methane emissions factor for all underground mines is 83.15 ft^3/ton.
[b]NA = Not available

Table 44. Methane emissions factors and calculated emissions rate (ton methane/ton coal produced) per mine type and region. Emissions rate assumes methane gas density of 47,000 ft^3/ton.

Region	**Coalfield**	**Emissions Factor (ft^3/ton)**				**Calculated Emissions Rate (ton Methane/ton coal produced)**			
		Surface mine	post mine surface	Under ground	post mine under ground	Surface mine	post mine surface	Under ground	post mine under ground
Colorado Plateau	Danforth Hills	66.2	10.8	76.22	63.8	1.E-03	2.E-04	2.E-03	1.E-03
	Deserado	32	5.2	76.22	32.3	7.E-04	1.E-04	2.E-03	7.E-04
	South Piceance	66.2	10.8	76.22	63.8	1.E-03	2.E-04	2.E-03	1.E-03
	South Wasatch	32	5.2	76.22	32.3	7.E-04	1.E-04	2.E-03	7.E-04
	Yampa	32	5.2	76.22	32.3	7.E-04	1.E-04	2.E-03	7.E-04
	Henry Mountains	32	5.2	76.22	32.3	7.E-04	1.E-04	2.E-03	7.E-04
	San Juan	14.6	2.4	76.22	34.1	3.E-04	5.E-05	2.E-03	7.E-04
Rocky	Ashland	11.2	1.8	76.22	5.1	2.E-04	4.E-05	2.E-03	1.E-04

Mountains and Great Plains	Colstrip	11.2	1.8	76.22	5.1	2.E-04	4.E-05	2.E-03	1.E-04
	Decker	11.2	1.8	76.22	5.1	2.E-04	4.E-05	2.E-03	1.E-04
	Gilette	11.2	1.8	76.22	5.1	2.E-04	4.E-05	2.E-03	1.E-04
	Sheridan	11.2	1.8	76.22	5.1	2.E-04	4.E-05	2.E-03	1.E-04
	Williston-Beulah-Zap	11.2	1.8	76.22	5.1	2.E-04	4.E-05	2.E-03	1.E-04
	Williston-Hagel	11.2	1.8	76.22	5.1	2.E-04	4.E-05	2.E-03	1.E-04
	Williston-Hansen	11.2	1.8	76.22	5.1	2.E-04	4.E-05	2.E-03	1.E-04
	Williston-Harmon	11.2	1.8	76.22	5.1	2.E-04	4.E-05	2.E-03	1.E-04
	Hanna-Ferris 23, 25,31,50,65	11.2	1.8	76.22	5.1	2.E-04	4.E-05	2.E-03	1.E-04
	Hanna-Hanna 77,78,79,81	11.2	1.8	76.22	5.1	2.E-04	4.E-05	2.E-03	1.E-04
	Carbon-Johnson	11.2	1.8	76.22	5.1	2.E-04	4.E-05	2.E-03	1.E-04
	Green River-Dead Man	66.2	10.8	76.22	41.6	1.E-03	2.E-04	2.E-03	9.E-04
Gulf Coast	Wilcox	66.2	10.8	NA	41.6	1.E-03	2.E-04		9.E-04
	Lower Wilcox	66.2	10.8	NA	41.6	1.E-03	2.E-04		9.E-04
Appalachia	Pittsburgh	119	19.3	87.8	14	3.E-03	4.E-04	2.E-03	3.E-04
	Upper Freeport	119	19.3	87.8	14	3.E-03	4.E-04	2.E-03	3.E-04
	Lower Kittaning	119	19.3	87.8	14	3.E-03	4.E-04	2.E-03	3.E-04
	Pond Creek	49.8	8.1	88.6	20-129.7	1.E-03	2.E-04	2.E-03	4.E-04 – 3E-03
	Fire Clay	49.8	8.1	88.6	20-129.8	1.E-03	2.E-04	2.E-03	4.E-04 – 3E-03
	Pocohontas	49.8	8.1	88.6	20-129.9	1.E-03	2.E-04	2.E-03	4.E-04 – 3E-03
Illinois	Springfield	68.6	11.1	45.2	20.9	1.E-03	2.E-04	1.E-03	4.E-04
	Herrin	68.6	11.1	45.2	20.9	1.E-03	2.E-04	1.E-03	4.E-04
	Danville	68.6	11.1	45.2	20.9	1.E-03	2.E-04	1.E-03	4.E-04

Energy Use

Energy consumption is the amount of diesel fuel and electricity needed:

$$E = \sum N_i J_i h$$

where E = energy consumed, BTU

N_i = number of equipment i

J_i = power rating of equipment i

h = total mine operating hours (Chapter 1)

Estimated energy use is summarized in Table 2.

Impacts not calculated

Water availability

Water availability can be affected when the ground subsides, or if a hole is dug to access underlying coals. If you dig a hole where a stream is, it will disappear. If you remove the seam under a water body, and allow the land to subside, the body will fall into the ground. This can be drastic, such as disappearance of a stream. It can also be subtle, such as slow drainage of a lake or pond. Subsidence of underground streams or bodies of water can also result from mining an underlying seam. Collapse of a large body of water may be a safety concern to underground mines, because it can cause flooding and potentially drown workers.

Because of the potential impact on surface streams, the SMCRA, in 30 C.F.R. § 816.57 states:

> (a) No land within 100 feet of a perennial stream or an intermittent stream shall be disturbed by surface mining activities, unless the regulatory authority specifically authorizes surface mining activities closer to, or through, such a stream. The regulatory authority may authorize such activities only upon finding that: (1) Surface mining activities will not cause or contribute to the violation of applicable State or Federal water quality standards, and will not adversely affect the water quantity and quality or other environmental resources of the stream.

The U.S. Bureau of Mines Information Circular 8741 (Babcock and Hooker, 1977) provides guidelines for mining under bodies of water, and to assess the potential impact if a breakthrough were to occur.

To evaluate how mining affects water availability, a detailed GIS analysis examining the location of water bodies relative to the coal and the likelihood that a mine will interrupt them should be performed. This data is not currently available in a ready to use format, as water flow data is not complete and there is no substantial groundwater dataset in the U.S.

Environmental impact cost or value

This section discusses the transformation of the damage quantified in section 2 into financial figures ($).

Land Value

There are several approaches to estimating land use impacts in life cycle assessment. The most common approach for evaluating life-cycle area has involved a metric that considers the surface area occupied multiplied by duration of use (e.g. acre-years) [82. In this analysis, however, a financial value to the land is estimated. U.S. land values by purpose are assumed to fall within the range of values assigned by the USDA {United State Department of Agriculture National Agriculture Statistics Service, 2008 #166]to farmland, and estimated ecological system values for grass and rangeland, marshes, and swamps [83]. These valuation data are reported in Table 45 - Table 48.

Table 45. U.S. Farm Real Estate Value by Coal Region [84]

Appalachia		Gulf Coast		Illinois		Colorado Plateau		Rocky Mountains and Great Plains	
State	$/Acre	State	$/Acre	State	$/Acre	State	$/Acre	State	$/Acre
PA	4220	LA	1770	IL	2610	AZ	2330	WY	370
OH	3180	TX	1030	IN	2770	CO	940	MT	410
KY	2000			KY	2000	NM	360	ND	505
VA	3200					UT	1460		
WV	1500								
Average	2820	Average	1070	Average	2500	Average	1170	Average	546

Table 46. U.S. Cropland value by Coal Region [84]

Appalachia		Gulf Coast		Illinois		Colorado Plateau		Rocky Mountains and Great Plains	
State	$/Acre	State	$/Acre	State	$/Acre	State	$/Acre	State	$/Acre
PA	4280	LA	1510	IL	3370	AZ	9000	WY	1070
OH	3230	TX	1070	IN	3150	CO	1170	MT	652
KY	2500			KY	2500	NM	1530	ND	546
VA	4100					UT	3060		
WV	3300								
Average	3482	Average	1290	Average	3007	Average	3690	Average	756

Table 47. U.S. Pasture value by Coal Region [84]

Appalachia		Gulf Coast		Illinois		Colorado Plateau		Rocky Mountains and Great Plains	
State	$/Acre	State	$/Acre	State	$/Acre	State	$/Acre	State	$/Acre
PA	2200	LA	1670	IL	1720	AZ	650	WY	280
OH	2100	TX	869	IN	1930	CO	620	MT	370
KY	1980			KY	1980	NM	250	ND	220
VA	3850					UT	690		
WV	1660								
Average	2358	Average	1270	Average	1877	Average	553	Average	290

Table 48. Ecosystem values ($/acre-year) [83]

Category	Low	High	Average
Temperate forests	104	138	121
Grassland/Rangeland	93	93	93
Tidal Marsh/Mangroves	442	9684	4412
Swamps/Floodplains	3946	12132	7832

Mined acreage per ton is so low that the maximum estimated land value is $0.50/ton. In order to use the ecosystem values in Table 48, the land area affected by mining is converted by multiplying it by the production rate:

$$LA_2 = LA_1 \times ML_i \times P_i$$

where LA_2 = land area, acre-year
LA_1 = land area, acre/ton

ML_i = mine life, years
P_i = total production over mine lifetime, tons

Table 49 –

Table 51 show estimated land impact costs per mine production. Continuous mine subsidence per ton of coal produced is so insignificant that the value of land impacted is $0/ton of coal mined. For longwall and surface mining, impact values are less than $1/ton.

Table 49. Estimated longwall mine land cost ($/ton coal produced)

Region	Coalfield	Farm Land	Cropland	Pasture	Temperate forest	Grassland/ Rangeland	Tidal Marsh/ Mangroves	Swamps/ Floodplains
Colorado Plateau	South Piceance	0.02	0.06	0.01	0.00	0.00	0.07	0.13
	South Wasatch	0.01	0.04	0.01	0.00	0.00	0.05	0.08
	Yampa	0.02	0.06	0.01	0.00	0.00	0.07	0.12
	Henry Mountains	0.02	0.06	0.01	0.00	0.00	0.07	0.13
	San Juan	0.01	0.01	0.01	0.00	0.00	0.06	0.11
Rocky Mountains and Great Plains	Ashland	0.00	0.01	0.00	0.00	0.00	0.04	0.07
	Colstrip	0.00	0.00	0.00	0.00	0.00	0.03	0.05
	Decker	0.00	0.00	0.00	0.00	0.00	0.03	0.05
	Gilette	0.00	0.01	0.00	0.00	0.00	0.03	0.05
	Sheridan	0.00	0.01	0.00	0.00	0.00	0.04	0.06
	Williston-Beulah-Zap	0.00	0.00	0.00	0.00	0.00	0.01	0.02
	Williston-Hagel	0.00	0.01	0.00	0.00	0.00	0.03	0.05
	Williston-Hansen	0.00	0.01	0.00	0.00	0.00	0.04	0.08
	Williston-Harmon	0.00	0.01	0.00	0.00	0.00	0.04	0.07
	Carbon-Johnson	0.00	0.01	0.00	0.00	0.00	0.03	0.06
	Green River-Dead Man	0.00	0.00	0.00	0.00	0.00	0.01	0.01
Gulf Coast	Wilcox	0.01	0.01	0.01	0.00	0.00	0.04	0.08
	Lower Wilcox	0.02	0.03	0.01	0.00	0.00	0.04	0.07
Appalachia	Pittsburgh	0.04	0.05	0.03	0.00	0.00	0.06	0.10
	Upper Freeport	0.04	0.05	0.03	0.00	0.00	0.06	0.11
	Lower Kittaning	0.14	0.17	0.11	0.01	0.00	0.21	0.38

	Pond Creek	0.07	0.08	0.05	0.00	0.00	0.10	0.18
	Fire Clay	0.05	0.06	0.04	0.00	0.00	0.08	0.13
	Pocohontas	0.00	0.00	0.03	0.00	0.00	0.05	0.10
Illinois	Springfield	0.00	0.00	0.00	0.01	0.00	0.21	0.38
	Herrin	0.00	0.00	0.00	0.00	0.00	0.02	0.03
	Danville	0.00	0.00	0.00	0.00	0.00	0.04	0.07

Table 50. Estimated continuous mine land cost ($/ton coal produced)

Region	Coalfield	Farmland	Cropland	Pasture	Temperate forest	Grassland	Tidal/ Mangrove	Swamps/ Floodplains
Colorado Plateau	South Piceance	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	South Wasatch	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Yampa	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Henry Mountains	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	San Juan	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Rocky Mountains and Great Plains	Ashland	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Colstrip	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Decker	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Gilette	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Sheridan	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Williston-Beulah-Zap	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Williston-Hagel	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Williston-Hansen	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Williston-Harmon	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Carbon-Johnson	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Green River-Dead Man	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Gulf Coast	Wilcox	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Lower Wilcox	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Appalachia	Pittsburgh	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Upper Freeport	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Lower Kittaning	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Pond Creek	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Fire Clay	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Pocohontas	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Illinois	Springfield	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Herrin	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Danville	0.00	0.00	0.00	0.00	0.00	0.00	0.00

Table 51. Surface mine land impacts ($/ton of coal produced)

Region	Coalfield	Farmland	Cropland	Pasture	Temperate forest	Grassland	Tidal/ Mangrove	Swamps/ Floodplains
Colorado Plateau	Danforth Hills	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Deserado	0.01	0.04	0.01	0.00	0.00	0.04	0.08
	South Piceance	0.08	0.23	0.03	0.01	0.01	0.28	0.49
	South Wasatch	0.08	0.24	0.04	0.01	0.01	0.28	0.50
	Yampa	0.09	0.26	0.04	0.01	0.01	0.31	0.55
	Henry Mountains	0.10	0.29	0.04	0.01	0.01	0.35	0.62
	San Juan	0.02	0.21	0.03	0.01	0.01	0.25	0.44
Rocky Mountains and Great Plains	Ashland	0.01	0.01	0.01	0.00	0.00	0.08	0.14
	Colstrip	0.01	0.02	0.01	0.00	0.00	0.14	0.25
	Decker	0.00	0.01	0.00	0.00	0.00	0.03	0.06
	Gilette	0.00	0.00	0.00	0.00	0.00	0.03	0.05
	Sheridan	0.00	0.01	0.00	0.00	0.00	0.03	0.06
	Williston-Beulah-Zap	0.01	0.02	0.01	0.00	0.00	0.12	0.22
	Williston-Hagel	0.01	0.02	0.01	0.00	0.00	0.11	0.20
	Williston-Hansen	0.02	0.03	0.01	0.00	0.00	0.18	0.31
	Williston-Harmon	0.01	0.02	0.01	0.00	0.00	0.13	0.23
	Hanna-Ferris 23, 25,31,50,65	0.00	0.00	0.00	0.00	0.00	0.00	0.01
	Hanna-Hanna 77,78,79,81	0.00	0.00	0.00	0.00	0.00	0.02	0.04
	Carbon-Johnson	0.01	0.02	0.01	0.00	0.00	0.09	0.16
	Green River-Dead Man	0.03	0.02	0.01	0.00	0.00	0.11	0.19
Gulf Coast	Wilcox	0.06	0.06	0.06	0.01	0.00	0.20	0.36
	Lower Wilcox	0.12	0.05	0.05	0.01	0.00	0.18	0.33
Appalachia	Pittsburgh	0.25	0.30	0.21	0.01	0.01	0.38	0.68
	Upper Freeport	0.19	0.24	0.16	0.01	0.01	0.30	0.53
	Lower Kittaning	0.61	0.76	0.51	0.03	0.02	0.96	1.71
	Pond Creek	0.32	0.39	0.26	0.01	0.01	0.50	0.88
	Fire Clay	0.24	0.30	0.20	0.01	0.01	0.38	0.67
	Pocohontas	0.00	0.35	0.24	0.01	0.01	0.45	0.79
Illinois	Springfield	0.00	0.00	0.00	0.03	0.02	0.92	1.63

Herrin	0.00	0.00	0.00	0.02	0.01	0.62	1.09
Danville	0.00	0.00	0.00	0.02	0.02	0.73	1.30

Methane emissions value

The value of methane emissions per ton of coal mined is calculated:

$$M_i = M_{EF,i}(GWP_{CH4})(P_{CH4})$$

where M_i = methane cost per ton of coal in region i ($/ton)
$M_{EF,i}$ = calculated methane emissions factor in region i, from Table 44 (ton/ton coal)
GWP_{CH4} = methane global warming potential, 19.1 $mtCO_2e$ [85]
P_{CH4} = U.S. methane emissions trading price, $3.80/ton [86]

The calculated cost of methane emissions is shown in

Table 52.

Table 52. Estimated methane emissions cost ($/ton coal produced)

Region	Coalfield	Surface mine	post mine surface	Underground	post mine underground
Colorado Plateau	Danforth Hills	0.10	0.02	0.12	0.10
	Deserado	0.05	0.01	0.12	0.05
	South Piceance	0.10	0.02	0.12	0.10
	South Wasatch	0.05	0.01	0.12	0.05
	Yampa	0.05	0.01	0.12	0.05
	Henry Mountains	0.05	0.01	0.12	0.05
	San Juan	0.02	0.00	0.12	0.05
Rocky Mountains and Great Plains	Ashland	0.02	0.00	0.12	0.01
	Colstrip	0.02	0.00	0.12	0.01
	Decker	0.02	0.00	0.12	0.01
	Gilette	0.02	0.00	0.12	0.01
	Sheridan	0.02	0.00	0.12	0.01
	Williston-Beulah-Zap	0.02	0.00	0.12	0.01
	Williston-Hagel	0.02	0.00	0.12	0.01
	Williston-Hansen	0.02	0.00	0.12	0.01
	Williston-Harmon	0.02	0.00	0.12	0.01
	Hanna-Ferris 23, 25,31,50,65	0.02	0.00	0.12	0.01
	Hanna-Hanna 77,78,79,81	0.02	0.00	0.12	0.01
	Carbon-Johnson	0.02	0.00	0.12	0.01
	Green River-Dead Man	0.10	0.02	0.12	0.06
Gulf Coast	Wilcox	0.10	0.02		0.06
	Lower Wilcox	0.10	0.02		0.06
Appalachia	Pittsburgh	0.18	0.03	0.14	0.02
	Upper Freeport	0.18	0.03	0.14	0.02
	Lower Kittaning	0.18	0.03	0.14	0.02
	Pond Creek	0.08	0.01	0.14	
	Fire Clay	0.08	0.01	0.14	
	Pocohontas	0.08	0.01	0.14	
Illinois	Springfield	0.11	0.02	0.07	0.03

Herrin	0.11	0.02	0.07	0.03
Danville	0.11	0.02	0.07	0.03

Impacts not assigned values

Acid mine drainage – there is no value for sulfuric acid.
Soil quality – no value for soil.

Environmental mitigation costs

There are several individual technologies that can be used to address environmental impacts at various stages of a mine's life.

	Premining	Mining	Postmining
Land	Catalog flora Store soil properly	Fill underground voids Stow-as-you-go	Backfill to approximate original contour Reclaim with plants
Water	Survey water bodies on land	Treat and release Increase water efficiency	Treat and release Rebuild streams
Solid waste	Design impoundment	Impoundment Coal piles	Maintain impoundment Vegetate pile
Air	Control dust and fuel use Reduce methane emissions	Control dust with water Reduce atmospheric release of methane via dilution Reduce fuel use	Cap waste piles to reduce methane emissions and metal leaching Seal mines Reduce fuel use during reclamation

Things that this environmental analysis will <u>not</u> address and why:
Water availability – not enough GIS data about the location of the water relative to coal resource, whether it is surface water or underground water. No information is available about underground water depths or flows. Surface intermittent stream data is also not available in a form that can be used.

Soil quality is not going to be evaluated because no data is readily available on mining-induced erosion.

Dust which is dependent on erosion qualities.

Energy efficiency – while this is a problem, the energy balance in coal extraction is not the focus of this study. Future technology's fuel efficiency is also not available to estimate future fuel use.

Of the technologies listed in the table, methane treatment is the most prevalently reported, because of safety issues. Lack of regulatory enforcement and a lot of exclusions allow these problems to be untreated.

Problem	Technology Available	Used?
Subsidence	Stowage	No
Valley fill	Concurrent reclamation	No
Acid mine drainage	Alkaline addition	?
Stream fill	Rebuild streams	?
Impoundment failure	Reduce waste	
Dust control	Water spray	Y
Methane control	Ventilation/dilution or capture	Y

Underground mine specific discussion

There are a few preventative measures that may be used to address acid mine drainage and several that can be used for subsidence prevention. Most notable is backfill, or grouting, to prevent subsidence. Technologies are chosen for this analysis partly because they are not a forever commitment. They prevent damage from occurring. Estimating perpetual costs has a high degree of uncertainty. Estimating prevention costs leads to less uncertainty because the time period of application is well defined.

Past and current technologies focus on addressing problems after they arise. All the technologies that aren't used regularly are those that are perceived to raise operating costs. These technologies are those that aren't required for safety, or are deemed to slow down productivity or too costly for additional materials. An overarching solution to these problems is grout injection to coat cut coal surfaces and to fill ground fissures, thus mitigating acid mine drainage and subsidence, respectively. The goal of the analysis in this section is to evaluate a technology that can mitigate environmental problems before they happen. That is, means of preemptively preventing damage. A method that comes to mind, that can address solid waste, subsidence and acid formation from underground mining is grout injection. For surface mining, concurrent reclamation, by which material from one pit is used to fill other pits as mining progresses, can reduce the mine footprint and exposed land.

Table 53 shows the combination of treatments that will be examined.

Table 53. Treatment options for underground mines and problems solved, highest cost to lowest cost

Treatment	Subsidence	Acid mine drainage	Air	Solid waste	GHG
Sealant and full grouting in cavity	X	X		X	
Full grouting in cavity	X			X	
Fissure grouting and sealant	X	X		X	
Fissure grouting and grout mine	X	X		X	

opening					
Fissure grouting only	X			X	
Seal mine only		X		X	
Premine methane development		X			X

Two types of materials will be assessed, ash or fines + cement, and cement only.

Surface mine specific discussion

Table 54. Treatment options for underground mines and problems solved, highest cost to lowest cost

Treatment	Pit footprint	Acid mine drainage	Air	Solid waste	GHG
Landfill liner and backfill	X	X		X	
Sealant and backfill	X	X		X	
Premine methane development	X		X	X	X

Landfill liner cost: geosynthetic clay liners appear to be the liners of the future. Geotextile layers with sodium bentonite clay in between are used in landfills throughout the U.S. The installed (1994) cost is \$0.42 - \$0.60/ft^2, but depends on shipping distance, area to be covered, market demand and season [87].

Avoiding Acid Mine Drainage

Some technologies are primarily for use in acid mine drainage prevention. To date, the majority of technologies that address acid mine drainage are treatment techniques. They address acid mine drainage after it happens. The same can be said of subsidence treatment. Grouting and fill is typically injected after mining. The technologies described here are not novel; their proposed application is unique because it may be that (a) they are not typically applied to coal mining, or (b) they are usually applied after the mine is closed and/or abandoned, or (c) both (a) and (b).

There are two general approaches to dealing with acid mine drainage. The first is to add alkaline material to reduce the acidity of the water draining from the mine. The second is to install a physical barrier that traps the water or prevents the acid forming material from having contact with air and water. The first approach requires perpetual treatment. The second approach is a one-time treatment that prevents environmental damage. Traditionally, physical barriers either prevent water and/or air from reacting with coal, or prevent water discharge from the mine. Water and air react with the sulfur in the coal to create hydrosulfuric acid (H_2SO_4), which then leaches metals from the coal in to the water. Historically, the approach is to seal the mine so that it floods. The water in the mine prevents air from touching the coal – only two ingredients are available for the reaction. The reactants are not all present for acid formation, so acid does not form. Furthermore, because water is sealed in the mine, there is no drainage.

Methods examined are sealants for all mine types and landfill liners for surface mine pits. The sealants are a penetrating coating that prevents water and air from reacting with exposed rock in oil and gas drillholes, and metal mines [88]. The cost to apply these materials to mines is calculated:

$$CC = \frac{SF \times CP}{P}$$

where CC = coating cost ($/ton)
SF = surface area of exposed workings (ft^2)
CP = coating price ($\$/ft^2$)

All mine dimensions are as defined in Chapter 1. In longwall mines, it is assumed that sealants can't be applied to the panel walls due fast collapse of the roof. The sealant is applied to the development sections, and the surface area is the total exposed gate pillar area. In continuous mines, the calculated surface area is the total surface area of all pillars. In a surface mine, the total area is:

$$SF_{pit} = 2n_{pit}L_{pit}W_{pit}\sum(h+m)L_{pit} + A_{pit}$$

where SF_{pit} = pit surface area (ft^2)
n_{pit} = number of pits
L_{pit} = pit length (ft)
W_{pit} = pit width (ft)
h = overburden and interburden depth (ft)
m = seam height (ft)
L_{pit} = pit length (ft)

Coating costs

Landfill liner cost: geosynthetic clay liners appear to be the liners of the future. Geotextile layers with sodium bentonite clay in between are used in landfills throughout the U.S. The installed (1994) cost is \$0.42 - \$0.60/ft^2, but depends on shipping distance, area to be covered, market demand and season [87].

Sealant costs: \$2-8/ft^2 [88].

Grouting costs: gunite or shotcrete is to be used. The application cost, including overhead and profit is $1.94 - $7.40/ft2 [89]. Because sealant and grout cost are so similar, it is assumed that the two would be used interchangeably.

The cost to avoid acid mine drainage by coating exposed surface areas is shown in Table 55 and Table 56.

Table 55. Calculated underground mine sealant or grout cost ($/Ton of coal produced)

Region	Coalfield	Longwall 0.05	Longwall 0.5	Longwall 0.95	Continuous 0.05	Continuous 0.5	Continuous 0.95
Colorado Plateau	South Piceance	1	4	8	0	1	3
	South Wasatch	1	4	8	0	1	3
	Yampa	1	4	8	0	1	4
	Henry Mountains	1	4	8	0	1	4
	San Juan	5	3	8	0	1	3
Rocky Mountains and Great Plains	Ashland	18	3	27	1	3	13
	Colstrip	11	4	14	0	1	5
	Decker	3	18	49	0	2	11
	Gilette	22	10	63	1	5	20
	Sheridan	2	17	44	1	5	17
	Williston-Beulah-Zap	6	3	12	0	1	2
	Williston-Hagel	1	4	11	0	0	2
	Williston-Hansen	3	3	11	0	1	2
	Williston-Harmon	3	4	13	0	1	2
	Carbon-Johnson	10	4	18	0	0	2
	Green River-Dead Man	7	5	12	0	1	4
Gulf Coast	Wilcox	4	4	10	0	0	2
	Lower Wilcox	5	3	14	0	0	2
Appalachia	Pittsburgh	2	4	8	0	1	2
	Upper Freeport	2	4	8	0	1	2
	Lower Kittaning	5	3	8	1	3	6
	Pond Creek	1	4	8	0	1	5
	Fire Clay	1	4	8	0	1	3
	Pocohontas	1	4	8	0	1	4
Illinois	Springfield	5	3	8	0	1	4
	Herrin	4	3	8	0	1	4
	Danville	1	4	8	0	1	4

Table 56. Calculated surface mine acid mine drainage avoidance costs ($/ton of coal produced)

Region	Coalfield	Sealant or Grout 0.05	Sealant or Grout 0.5	Sealant or Grout 0.95	Landfill Liner 0.05	Landfill Liner 0.5	Landfill Liner 0.95
Colorado Plateau	Danforth Hills	2	16	165	0	2	17
	Deserado	2	27	422	0	3	48
	South Piceance	5	203	2681	1	18	307
	South Wasatch	3	132	2230	0	17	249
	Yampa	7	215	5999	1	27	365
	Henry Mountains	2	134	1978	0	16	199
	San Juan	3	187	2782	0	21	324
Rocky Mountains and Great Plains	Ashland	2	50	983	0	6	89
	Colstrip	2	36	590	0	4	58
	Decker	1	5	184	0	1	20
	Gilette	1	16	162	0	2	18
	Sheridan	1	15	173	0	2	19
	Williston-	2	16	164	0	2	15

	Beulah-Zap						
	Williston-Hagel	1	7	126	0	1	13
	Williston-Hansen	2	21	233	0	2	21
	Williston-Harmon	1	6	89	0	1	11
	Hanna-Ferris 23, 25,31,50,65	2	40	428	0	4	34
	Hanna-Hanna 77,78,79,81	1	13	152	0	2	15
	Carbon-Johnson	2	77	669	0	8	74
	Green River-Dead Man	1	6	106	0	1	10
Gulf Coast	Wilcox	1	8	154	0	1	15
	Lower Wilcox	2	9	135	0	1	12
Appalachia	Pittsburgh	1	8	153	0	1	15
	Upper Freeport	1	8	130	0	1	15
	Lower Kittaning	1	9	118	0	1	13
	Pond Creek	1	8	151	0	1	18
	Fire Clay	1	9	130	0	1	11
	Pocohontas	1	9	116	0	1	16
Illinois	Springfield	1	8	124	0	1	10
	Herrin	1	8	187	0	1	15
	Danville	7	249	6573	1	29	519

Avoiding subsidence

I called Geo-Solutions and talked to Chris Ryan. He said that it costs \$20 per linear foot to drill. Rule of thumb is that it will cost \$40-50/cu yard to inject a 10:1 mixture of cement and fly ash. The cement material comprises a little more than one-half of this estimated cost. In new mines, 60% of the mined out volume is needed and in old mines 45% of the mined out volume is needed.

Two options were examined: partial fill and complete fill. The partial fill option assumes that grout will be injected into the fissures overlying the mine workings, as done by the Australians. The full fill option assumes that the total underground void will be filled with the 10:1 mixture of cement and fly ash. Both fill options assume the injections wells will be set up according to Australian experience. Wells are set 1969 ft (600 m) apart, covering a 5382 ft^2 (500 m^2) control area. These wells are moved ahead of panel development and can be reused from panel to panel. The total cost, according to Dr. Baotang Shen at CSIRO is \$2-3 million AUD for 2 wells (capital and operating cost). However, I use the geo-solutions rule of thumb.

The subsidence volume is calculated to determine grout volume:

Full fill	$V_{sub,i} = MV$ where $V_{sub,i}$ = subsidence volume in region i MV = mine void volume of mine type
Partial fill	$V_{sub,i} = S_{\max,i} \times A_{sub,i}$ where $V_{sub,i}$ = subsidence volume in region i $S_{max,i}$ = maximum subsidence depth in region i $A_{sub,i}$ = subsidence area in region i

The drilling depth for the well in the case of full fill is the total overburden depth, h; in the case of partial fill it is half the overburden depth.

Grouting cost with the geo-solutions grout for full fill and partial fill, are determined for the entire mine area based on the number of longwall panels for a longwall mine or number of rooms in a continuous mine. The results are shown in Table 57 and Table 58 for partial fill and full fill costs to avoid subsidence in underground mines. Complete fill also offers benefits in reducing acid mine drainage. When a mine is grouted, there can be as much as 90% inflow reduction using 10% compressible uncemented backfill [90]. The alkaline material in the cement can also balance the acids that form in acid mine drainage.

Table 57. Calculated costs for partial fill for subsidence avoidance for underground mines ($/ton of coal produced)

		Longwall			Continuous		
Region	**Coalfield**	0.05	0.5	0.95	0.05	0.5	0.95
	South Piceance	1.3	8.0	21.1	0.1	0.2	0.6
	South Wasatch	1.3	8.0	20.6	0.1	0.2	0.4
Colorado Plateau	Yampa	1.4	8.0	20.5	0.1	0.2	0.9
	Henry Mountains	1.2	7.2	20.3	0.1	0.2	0.7
	San Juan	1.4	8.2	23.1	0.1	0.2	0.7
	Ashland	3.9	21.0	83.5	0.1	0.2	0.6
	Colstrip	1.5	9.6	41.1	0.1	0.2	0.4
Rocky Mountains and Great Plains	Decker	4.2	34.9	107.7	0.2	0.5	1.5
	Gilette	5.4	42.2	125.2	0.2	0.4	1.2
	Sheridan	3.9	38.1	125.2	0.2	0.4	0.8
	Williston-Beulah-Zap	1.7	7.8	32.5	0.1	0.2	0.4
	Williston-Hagel	1.1	8.3	24.5	0.1	0.2	0.4
	Williston-Hansen	1.2	7.6	24.0	0.1	0.2	0.4
	Williston-Harmon	1.8	8.1	25.1	0.1	0.2	0.3
	Carbon-Johnson	2.2	9.3	37.9	0.1	0.3	0.5
	Green River-Dead Man	1.8	10.9	36.5	0.1	0.2	0.3
Gulf Coast	Wilcox	1.3	6.6	23.4	0.1	0.2	0.4
	Lower Wilcox	1.3	7.6	22.9	0.1	0.2	0.3
Appalachia	Pittsburgh	1.3	7.2	18.6	0.1	0.2	0.5
	Upper Freeport	1.1	7.2	16.2	0.1	0.2	0.4
	Lower Kittaning	1.5	9.4	22.1	0.6	1.2	2.5
	Pond Creek	1.1	7.7	18.7	0.1	0.3	1.0
	Fire Clay	1.0	7.1	19.2	0.1	0.2	0.7

	Pocohontas	1.4	7.6	21.5	0.1	0.3	1.0
Illinois	Springfield	1.1	7.5	17.5	0.1	0.4	1.2
	Herrin	1.1	6.9	18.9	0.1	0.3	1.2
	Danville	1.1	7.0	17.7	0.1	0.3	1.4

Table 58. Full fill subsidence avoidance cost for underground mines ($/ton of coal produced)

			Longwall			**Continuous**	
Region	**Coalfield**	0.05	0.5	0.95	0.05	0.5	0.95
Colorado Plateau	South Piceance	94.1	585.0	1259.9	47.3	53.5	61.6
	South Wasatch	94.1	585.0	1259.9	46.6	53.5	61.4
	Yampa	94.1	585.0	1259.9	47.0	53.6	61.5
	Henry Mountains	94.1	585.0	1259.9	47.0	53.4	62.3
	San Juan	94.1	585.0	1259.9	47.1	53.2	61.8
Rocky Mountains and Great Plains	Ashland	94.1	585.0	1259.9	46.9	53.2	61.5
	Colstrip	94.1	585.0	1259.9	46.9	53.3	61.4
	Decker	94.1	585.0	1259.9	46.5	53.3	61.4
	Gilette	94.1	585.0	1259.9	47.0	53.2	61.6
	Sheridan	94.1	585.0	1259.9	46.7	53.2	61.5
	Williston-Beulah-Zap	94.1	585.0	1259.9	46.6	53.1	61.3
	Williston-Hagel	94.1	585.0	1259.9	46.5	53.1	61.4
	Williston-Hansen	94.1	585.0	1259.9	47.0	53.2	61.4
	Williston-Harmon	94.1	585.0	1259.9	46.7	53.1	61.4
	Carbon-Johnson	94.1	585.0	1259.9	46.5	53.1	61.4
	Green River-Dead Man	94.1	585.0	1259.9	46.5	53.2	61.3
Gulf Coast	Wilcox	94.1	585.0	1259.9	46.6	53.1	61.4
	Lower Wilcox	94.1	585.0	1259.9	46.5	53.1	61.3
Appalachia	Pittsburgh	94.1	585.0	1259.9	47.1	53.5	61.4
	Upper Freeport	94.1	585.0	1259.9	46.7	53.3	61.4
	Lower Kittaning	94.1	585.0	1259.9	48.7	55.5	64.8
	Pond Creek	94.1	585.0	1259.9	47.0	53.8	62.5
	Fire Clay	94.1	585.0	1259.9	47.1	53.5	61.7
	Pocohontas	94.1	585.0	1259.9	46.6	53.9	62.6
Illinois	Springfield	94.1	585.0	1259.9	47.3	54.2	62.4
	Herrin	94.1	585.0	1259.9	47.5	53.7	62.6
	Danville	94.1	585.0	1259.9	47.4	54.2	61.8

Cement production is carbon intensive. Portland cement production emits 1,800 – 2,100 lb CO_2 per ton of cement [91], or about 1 ton CO_2 per ton of cement. Assuming Portland cement density is 0.02 ton/ft^3 [92], the total project emissions are calculated (**Table** 59). They are not insignificant, especially when full grouting is considered.

Table 59. CO_2 Emissions Associated with Portland Cement Fill Project (Million Tons CO_2), based on 50th percentile estimate of fill volume needed

		Half Grout		**Full Grout**	
Region	**Coalfield**	Longwall	Continuous	Longwall	Continuous
Colorado Plateau	South Piceance	21	0	1477	26
	South Wasatch	31	0	2163	34
	Yampa	22	0	1431	28
	Henry Mountains	16	0	1143	25
Rocky Mountains and Great Plains	San Juan	22	0	1868	30
	Ashland	89	0	2750	40
	Colstrip	39	0	2582	38
	Decker	148	0	2767	40
	Gilette	223	0	2764	40
	Sheridan	170	0	2764	40
	Williston-Beulah-Zap	39	0	2685	37
	Williston-Hagel	32	0	2482	39
	Williston-Hansen	30	0	2181	33
	Williston-Harmon	39	0	2422	38
	Carbon-Johnson	51	0	2726	40
	Green River-Dead Man	52	0	2728	40
Gulf Coast	Wilcox	21	0	1813	32
	Lower Wilcox	24	0	1712	33
Appalachia	Pittsburgh	12	0	1149	22
	Upper Freeport	16	0	1408	27
	Lower Kittaning	8	0	477	10
	Pond Creek	14	0	995	22
	Fire Clay	14	0	1231	23
	Pocohontas	16	0	1156	23
Illinois	Springfield	7	0	507	11
	Herrin	10	0	738	16
	Danville	6	0	476	11

To avoid these CO_2 emissions, a non-cement fill can be used. As an alternative to cement and coal fine backfill, rockfill can be used. Rockfill is sized or unsized material that is typical mixed with a binder such as Portland cement. Rockfill is not widely used. It accounts for only 6% of fill used in mines worldwide [93]. Rockfill cost estimates range from approximately Canadian$5/tonne [93] (assuming rockfill density is 1.88 tonne/m^3 [93] and a 1998 exchange rate of 1.4 CAD to 1 USD [94] this is $5/$yd^3$) to Australia$2 - $20/$m^3$ [95] (assuming 0.87 AUD to 1 USD [96], this is $2 - $21/$yd^3$). The cost of fill is assumed to be $2 - $21/$yd^3$. Because fill material cost is approximately half of the total injection cost, the revised backfilling cost if rockfill is to be used is $22 - $46/$yd^3$. The estimated rockfill costs, assuming complete fill because rockfill is less stable than cement-based fill, are shown in Table 60.

Table 60. Estimated Rockfill costs for full fill ($/ton of coal produced)

		Longwall			**Continuous**		
Region	**Coalfield**	0.05	0.5	0.95	0.05	0.5	0.95
Colorado Plateau	South Piceance	73.6	447.5	1238.6	29.5	43.8	60.5
	South Wasatch	73.6	447.5	1238.6	29.5	43.6	60.3
	Yampa	73.6	447.5	1238.6	29.9	43.7	60.3
	Henry Mountains	73.6	447.5	1238.6	29.4	43.6	60.3
Rocky Mountains and Great Plains	San Juan	73.6	447.5	1238.6	29.4	43.8	60.9
	Ashland	73.6	447.5	1238.6	29.5	43.6	60.2
	Colstrip	73.6	447.5	1238.6	29.4	43.4	60.1
	Decker	73.6	447.5	1238.6	29.3	43.5	60.1
	Gilette	73.6	447.5	1238.6	29.4	43.5	60.3
	Sheridan	73.6	447.5	1238.6	29.5	43.4	60.2
	Williston-Beulah-Zap	73.6	447.5	1238.6	29.3	43.3	60.1
	Williston-Hagel	73.6	447.5	1238.6	29.2	43.3	60.1
	Williston-Hansen	73.6	447.5	1238.6	29.3	43.3	60.1
	Williston-Harmon	73.6	447.5	1238.6	29.3	43.3	60.1
	Carbon-Johnson	73.6	447.5	1238.6	29.3	43.3	60.0
Gulf Coast	Green River-Dead Man	73.6	447.5	1238.6	29.5	43.4	60.2
	Wilcox	73.6	447.5	1238.6	29.2	43.3	60.2
Appalachia	Lower Wilcox	73.6	447.5	1238.6	29.3	43.3	60.1
	Pittsburgh	73.6	447.5	1238.6	29.4	43.6	60.9
	Upper Freeport	73.6	447.5	1238.6	29.5	43.5	60.2
	Lower Kittaning	73.6	447.5	1238.6	31.3	45.8	64.1
	Pond Creek	73.6	447.5	1238.6	29.9	43.5	61.1
	Fire Clay	73.6	447.5	1238.6	29.4	43.6	60.3
Illinois	Pocohontas	73.6	447.5	1238.6	29.9	43.7	61.3
	Springfield	73.6	447.5	1238.6	30.0	44.1	60.6

	Herrin	73.6	447.5	1238.6	29.6	44.2	60.8
	Danville	73.6	447.5	1238.6	30.0	44.4	60.6

Revegetation and reforestation costs

To estimate the cost of revegetation and reforestation, a base estimate of regarding and revegetation are used. Additional cost to restore forest land is added. The cost to restore the permitted area is calculated. Typically, bond requirements apply to the permitted area. This is the total acreage of a surface mine and the support facilities are for an underground mine.

Costs to regrade land, revegetate, and reforest are \$1,300/acre, \$1,350/acre [97], and \$120 - \$1400/acre [98], so the total estimated reclamation rate is \$2,750 - \$4,050/acre. Revegetation cost including reforestation cost per region and mine type is shown in Table 61.

Table 61. Calculated re-vegetation and reforestation cost per ton of coal produced

		Longwall			**Continuous**			**Surface**		
Region	**Coalfield**	0.05	0.5	0.95	0.05	0.5	0.95	0.05	0.5	0.95
Colorado Plateau	Danforth Hills							0	0	0
	Deserado							0.1	0.1	0.2
	South Piceance	0	0	0.1	0	0	0	0.2	0.3	0.7
	South Wasatch	0	0	0.1	0	0	0	0.2	0.2	0.3
	Yampa	0	0	0.1	0	0	0	0.2	0.3	0.6
	Henry Mountains	0	0	0.1	0	0	0	0.2	0.3	0.6
Rocky Mountains and Great Plains	San Juan	0	0	0.1	0	0	0	0.2	0.2	0.6
	Ashland	0	0	0	0	0	0	0	0.1	0.2
	Colstrip	0	0	0.1	0	0	0	0.1	0.1	0.3
	Decker	0	0	0	0	0	0	0	0	0.1
	Gilette	0	0	0	0	0	0	0	0	0.1
	Sheridan	0	0	0	0	0	0	0	0	0.1
	Williston-Beulah-Zap	0	0	0.1	0	0	0	0.1	0.1	0.3
	Williston-Hagel	0	0	0.1	0	0	0	0.1	0.1	0.3
	Williston-Hansen	0	0	0.1	0	0	0	0.1	0.1	0.4
	Williston-Harmon	0	0	0	0	0	0	0.1	0.1	0.3
	Hanna-Ferris 23, 25,31,50,65							0	0	0
	Hanna-Hanna 77,78,79,81							0	0	0
	Carbon-Johnson	0	0	0	0	0	0	0.1	0.1	0.2
Gulf Coast	Green River-Dead Man	0	0	0	0	0	0	0.1	0.1	0.3
	Wilcox	0	0	0.1	0	0	0	0.1	0.2	0.7
Appalachia	Lower Wilcox	0	0	0.1	0	0	0	0.1	0.2	0.6
	Pittsburgh	0	0	0.1	0	0	0	0.1	0.2	0.7
	Upper Freeport	0	0	0.1	0	0	0	0.1	0.2	0.7
	Lower Kittaning	0.1	0.1	0.2	0	0	0	0.1	0.2	0.7
	Pond Creek	0	0	0	0	0	0	0.1	0.2	0.6

	Fire Clay	0	0	0.1	0	0	0	0.1	0.2	0.7
Illinois	Pocohontas	0	0	0	0	0	0	0.1	0.2	0.6
	Springfield	0.1	0.1	0.2	0	0	0	0.1	0.2	0.7
	Herrin	0	0	0.2	0	0	0	0.1	0.2	0.6
	Danville	0	0	0	0	0	0	0	0	0

Mountaintop removal and valley fill

Mountaintop removal and valley fill can be avoided by not mining at all, or at least *not* surface mining. However, to mine by underground methods, it is arguably difficult and dangerous:

> The mining industry argues that mountaintop mining is essential to conducting surface coal mining in Appalachia. The poor stability of the soil surrounding the coal deposits in this region makes it impossible to mine the coal using underground mining techniques [99].

To investigate the cost of underground mining by safer measures, the cost of autonomous mining units is examined. Sensors and autonomous or remote controls on underground devices allow unmanned mining to be undertaken. These devices can also improve productivity by eliminating downtime and cutting error. According to the Australian Commonwealth Scientific and Industrial Research Organization (CSIRO), smart longwall sensing technologies steer the longwall perfectly straight, increasing productivity by 30 percent. The CSIRO is pioneering longwall automation by using U.S. Army autonomous tank driving technology. The Beltana mine in New South Wales is currently demonstrating the technology. CSIRO believes that within 10-15 years the robotic capabilities will be fully autonomous. At this time, human operators are still needed to oversee the machines. The longwall automation technology is being commercialized in a joint agreement with the Joy mining equipment company. Sensing technologies above and below the shearer decrease dilution that results from cutting into the ceiling and floor. On the left-to-right pass, it senses the lay of the seam. On the right-to-left pass, it cuts according to the profile sensed in the previous cut. Unmanned continuous miners are developed, and await commercialization [100-102].

To calculate likely cost of autonomous mining, the best capital cost estimate for autonomous longwall shearers and continuous miners was determined according to the additional cost of guiding technology. Additional manufacturer cost is the best estimate of cost. However, once these technologies are commercialized the manufacturer will probably charge a price that includes marketing, research and development, and sales markup. The additional cost to add U.S. tank driving technology to a longwall shearer is 100,000 AUD, which is worth $115,000 assuming 0.87 AUD to the U.S. dollar [96]. As there are no recorded instances of robotic continuous miners being used, and the technology has not been commercialized, the additional cost of automation was estimated by comparing conventional and unmanned ground vehicle prices. The typical cost of an army truck is $50,000 - $150,000 [103]. The cost of an unmanned ground vehicle ranges from $600,000 - $800,000 [104, 105]. The revised capital costs for a longwall shearer and continuous miner, based on the baseline capital costs in Chapter 1, are $1.82 million -

$2.62 million dollars and $1.68 – 4.00 million dollars, respectively. The operating costs are assumed to remain the same, and it is also assumed that the same number of miners will work at the mine, albeit in a different function – likely remote control of the machines from the surface with occasional underground maintenance. As a conservative estimate, it is assumed longwall production rates increase by 30%, and continuous mining productivity increases by 10%. The revised mining costs to underground mine using autonomous equipment is shown in Table 62. Comparing the 50th percentile estimated costs to the 50th percentile baseline costs reported in Chapter 1, using autonomous longwalls would result in mines that are on average 10% cheaper to operate. Autonomous continuous miner units would not result in a significant decrease in mining costs.

Table 62. Calculated autonomous underground mining cost by mine type and coalfield ($/Ton)

		Longwall			Continuous		
Region	**Coalfield**	0.05	0.5	0.95	0.05	0.5	0.95
Colorado Plateau	Danforth Hills						
	Deserado						
	South Piceance	12	18	46	26	44	77
	South Wasatch	10	15	29	23	35	53
	Yampa	12	18	61	28	42	82
	Henry Mountains	13	19	42	28	46	101
	San Juan	10	17	51	25	41	82
Rocky Mountains and Great Plains	Ashland	9	14	25	22	31	48
	Colstrip	9	14	31	22	33	54
	Decker	9	14	20	22	30	47
	Gilette	9	14	20	22	30	48
	Sheridan	9	14	20	22	30	48
	Williston-Beulah-Zap	9	14	39	23	32	57
	Williston-Hagel	9	14	33	23	32	52
	Williston-Hansen	9	15	35	23	34	63
	Williston-Harmon	9	15	24	23	32	56
	Hanna-Ferris 23, 25,31,50,65						
	Hanna-Hanna 77,78,79,81						
	Carbon-Johnson	9	14	21	22	30	49
	Green River-Dead Man	9	14	29	22	31	49
Gulf Coast	Wilcox	10	16	41	23	35	106
	Lower Wilcox	10	15	55	24	37	87
Appalachia	Pittsburgh	12	21	83	29	49	124
	Upper Freeport	12	19	36	28	44	69
	Lower Kittaning	26	39	86	66	100	173
	Pond Creek	12	22	86	29	54	126
	Fire Clay	12	22	63	29	49	115
	Pocohontas	13	22	57	30	53	115
Illinois	Springfield	24	38	87	54	100	186
	Herrin	15	28	180	36	69	204
	Danville	24	41	105	57	94	167

Coalbed methane

Methane release and capture are similar for surface and underground mines. For shallow seams that may be surface mined, there may not be much methane in the coal by the time that it is mined. Due to weathering, the methane will have leached out well before the coal is developed for mining. However, when the coal is broken, the methane stored in the coal will be released to the atmosphere. The best way to control methane emissions from surface mining is to drill and capture the methane from the coal before mining activity begins. For deeper seams that are to be underground mined, the methane concentration will be greater in the seam. This methane can be developed prior to and during mining operations. Current practice requires methane dilution in the ventilation air during mining for safety reasons. An alternative approach to draining methane from the mine during operation would include directional drilling to extract methane from the seam before it is cut.

Methane mitigation is focused on its capture and use. In an EPA methane degasification handbook [106], they mention water removal from seams to liberate the gas. This water must be treated. Gob wells can be set up prior to mining, then "mined through" in order to release the gas into the well. Gob gas is inconsistent, and the well has a short life. Gob wells are historically used as a safety measure, rather than for greenhouse gas reduction. Horizontal drill holes are also used, and can be 1000' – 4000' long. Benefits to methane development include:

- Reduced downtime of $50-$100/minute saved by a gas drainage system, such that ventilation requirements are always met.
- Ventilation power cost savings.
- Reduced development costs
    - Fewer development openings are needed
    - Fewer shafts are needed
- Increased resource – less is devoted to development sections
- Mine safety
- Reduced dust due to slower ventilation velocity
- Reduced seep water

EPA also provides well and pipe cost data. They base their research on vertical well spacing from 40-160 acres being optimal in Alabama. Gob well has higher density spacing at the end of the panel, but overall there are 2-6 wells per panel, and horizontal wells are drilled every 200-400 feet [107]. Reduction rates assumed are provided by EPA [107].

Cost and quantity estimation data provided by the USEPA for a coalbed methane development project are shown in Table 63.

Table 63. Coalbed methane project cost and size data [106] [107]

Component	Number or Size of Units	Cost per Unit	Operating Cost per unit ($/Year)	Recovery Efficiency[a]
Degasification system (cost to drill, install, and complete wells and boreholes)				
Gob wells	1 for every 200,000 – 500,000 tons coal/year 2-5 wells per panel	$307,900 - $535,000	20,000 – 40,000	Up to 50%
Pre-mining vertical wells	1 well for every 250,000 – 1,000,000 tons of coal over the lifetime. Well spacing 20-80 acres.	$320,000 - $640,000	20,000 – 40,000	Up to 70%
Longhole horizontal boreholes	1 longwall hole borehole drilled per year per 1 million tons of coal (1 per longwall panel)	$60,000 - $100,000 per 1 million tons of coal. Includes drilling cost of $50-$80/m for 1200 m hole	105,000 – 640,000	Up to 50%
Capital cost for water disposal for vertical premining degasification wells	1 disposal system per project [107]	$100,000 - $2,800,000		
Operating cost for water disposal for vertical pre-mining degasification wells	17-70 barrels per tcm of gas produced [107]	$0.02 - $2/barrel		
Coalbed methane water treatment/disposal technologies		28,000 – 1,872,000[b]		

[a]Percent of methane that would otherwise be emitted.
[b]Annual cost ($/Year) based on 20 year project life and 10% discount rate

The typical methane development scenarios [106], which are examined, are:
Option 1 Gob wells only, used during mine operation
Option 2 Vertical wells, used to drain methane 5 years prior to mining (this is the only option that can be applied to a surface mine.)
Option 3 Vertical wells + gob wells
Option 4 Vertical wells + gob wells + horizontal boreholes (drain seam 3 years prior to mining)

The cost to drill wells and operate them is estimated. Dehydrating the gas or enriching it to be input to a pipeline or for sale is not calculated. Because water production rates are not known, the cost to treat water is not calculated, although the capital cost of a water treatment project is included in methane abatement cost.

The cost for methane collection is calculated using the capital and operating costs in Table 63:

$$MC = \frac{\sum CapM_j + \sum NPV_{op,j}}{P}$$

where MC = methane abatement cost ($/Ton)
$CapM_j$ = capital cost of methane project equipment j
$NPV_{op,j}$ = net present value of operating methane project equipment j
P = total lifetime mine production

The period over which NPV for operating costs is determined is the amount of time that the equipment would be used, as summarized in Table 64.

Table 64. Operating period used to calculated operating cost NPV

Equipment	Operating period (years)
Gob well	Mine lifetime as calculated in model
Vertical wells	5
Longwall horizontal borehole	3
Water disposal in premining	5
Coalbed methane water treatment	5

The number of each piece of equipment is determined as shown in Table 65.

Table 65. Equipment quantity per methane reduction option. Key to variables is below table.

Equipment	Option 1	Option 2	Option 3	Option 4
Gob wells	$\frac{N_{panels}}{uniform(2,5)}$	0	$\frac{N_{panels}}{uniform(2,5)}$	$\frac{N_{panels}}{uniform(2,5)}$
Verticial wells	0	$\frac{P}{uniform(250\times10^3,10^6)}$	$\frac{P}{uniform(250\times10^3,10^6)}$	$\frac{P}{uniform(250\times10^3,10^6)}$
Horizontal boreholes	0	0	0	N_{panels}
Water disposal system for vertical wells	0	1	1	1
Water disposal system for coalbed methane water	1	0	1	1

N_{panels} = number of longwall panels in longwall mine or sections of rooms and pillars in a continuous mine
P = total lifetime mine production
Uniform(x, y) = uniform distribution between numbers x and y

The estimated costs using the configurations of these four "typical" options, using EPA equipment costs and project sizing parameters is shown in Table 29 – Table 30.

Table 66. Option 1 Methane mitigation costs for underground mines, $/ton of coal produced

Region	Coalfield	Longwall			Continuous		
		0.05	0.5	0.95	0.05	0.5	0.95
Colorado Plateau	South Piceance	10.1	15.5	27.0	11.0	16.4	27.1
	South Wasatch	10.1	15.4	27.0	10.6	16.2	27.1
	Yampa	10.1	15.7	27.0	10.6	16.6	27.1
	Henry Mountains	10.1	15.5	27.0	11.1	16.5	27.1
	San Juan	10.1	15.5	27.2	11.1	16.4	27.0
Rocky Mountains and Great Plains	Ashland	10.1	15.4	27.0	10.6	16.1	27.0
	Colstrip	10.1	15.4	27.2	10.8	16.2	27.0
	Decker	10.1	15.4	27.0	10.6	16.1	27.0
	Gilette	9.7	15.8	28.3	10.0	15.9	27.6
	Sheridan	9.7	15.8	28.3	9.9	15.9	27.6
	Williston-Beulah-Zap	9.7	15.8	28.3	10.0	16.0	27.7
	Williston-Hagel	9.7	15.8	28.3	10.4	15.9	27.6
	Williston-Hansen	9.7	15.8	28.3	10.0	16.2	27.8
	Williston-Harmon	9.7	15.8	28.3	9.9	16.0	28.2
	Carbon-Johnson	9.7	15.8	28.3	9.9	15.9	27.6
	Green River-Dead Man	9.7	15.8	28.3	9.9	15.9	27.6
Gulf Coast	Wilcox	9.7	15.9	28.3	10.4	16.3	27.8
	Lower Wilcox	9.8	15.8	28.3	10.7	16.2	28.4
Appalachia	Pittsburgh	9.8	15.9	28.8	11.1	16.6	28.0
	Upper Freeport	9.7	15.9	28.8	10.4	16.3	27.9
	Lower Kittaning	10.2	16.2	29.2	11.8	18.2	29.2
	Pond Creek	9.6	16.0	28.7	10.8	16.9	28.0
	Fire Clay	9.6	15.9	28.7	10.8	16.8	28.5
	Pocohontas	9.8	15.8	28.8	11.1	16.7	28.2
Illinois	Springfield	10.1	16.4	29.1	11.8	18.1	30.0
	Herrin	9.9	16.1	28.9	11.3	17.6	28.9
	Danville	10.0	16.5	29.0	11.9	18.1	29.5

Table 67. Option 2 Methane mitigation costs for all mine types ($/ton of coal produced)

		Longwall			Continuous			Surface		
Region	**Coalfield**	0.05	0.5	0.95	0.05	0.5	0.95	0.05	0.5	0.95
Colorado Plateau	Danforth Hills							7.8	17.2	58.7
	Deserado							9.9	21.3	87.6
	South Piceance	6.0	10.3	23.9	7.2	11.2	24.4	12.5	70.8	451.3
	South Wasatch	5.8	10.1	23.8	6.9	11.3	24.0	14.3	70.6	257.3
	Yampa	6.0	10.2	23.9	7.2	11.6	24.6	12.6	81.3	465.0
	Henry Mountains	6.2	10.3	24.0	7.1	11.4	24.0	10.9	55.0	228.5
	San Juan	5.9	10.3	23.9	7.2	11.4	25.1	12.2	77.5	381.4
Rocky Mountains and Great Plains	Ashland	5.8	10.0	23.8	6.7	11.0	23.5	10.5	25.2	112.0
	Colstrip	5.8	10.1	23.8	7.0	11.1	23.8	8.8	23.3	146.3
	Decker	5.8	10.0	23.8	6.7	11.0	23.5	6.2	11.9	36.4
	Gilette	6.1	10.8	24.6	6.7	11.8	22.7	7.4	15.4	32.5
	Sheridan	6.1	10.8	24.6	6.7	11.8	22.7	8.4	16.5	44.8
	Williston-Beulah-Zap	6.1	10.8	24.7	6.7	11.8	22.8	7.9	16.2	36.3
	Williston-Hagel	6.1	11.2	24.6	6.8	11.8	22.7	7.2	12.9	30.3
	Williston-Hansen	6.1	11.3	24.6	6.8	12.2	22.7	8.2	16.3	41.0
	Williston-Harmon	6.1	10.8	24.6	6.8	11.9	22.9	7.4	13.5	25.6
	Hanna-Ferris 23, 25,31,50,65							9.6	21.3	52.0
	Hanna-Hanna 77,78,79,81							7.5	14.7	29.6
	Carbon-Johnson	6.1	10.8	24.6	6.8	11.8	22.7	9.7	31.9	213.1
	Green River-Dead Man	6.1	10.8	24.6	6.8	11.8	23.2	6.8	12.9	25.5
Gulf Coast	Wilcox	6.1	11.5	24.7	6.8	12.3	23.4	7.2	13.7	29.8
	Lower Wilcox	6.1	10.8	24.6	6.8	12.0	23.2	6.3	14.5	38.7
Appalachia	Pittsburgh	6.1	10.9	22.5	6.8	12.3	23.2	6.2	13.5	35.7
	Upper Freeport	5.9	10.9	22.5	6.7	11.9	22.7	6.8	14.6	39.8
	Lower Kittaning	6.5	11.4	23.0	8.0	14.5	25.4	7.1	14.0	38.7
	Pond Creek	5.9	10.9	22.4	7.0	12.6	23.0	6.6	13.5	36.8
	Fire Clay	6.1	10.9	22.5	6.9	12.4	22.8	7.3	13.5	31.1
	Pocohontas	6.1	10.9	22.4	7.2	12.3	23.1	7.1	14.8	38.2
Illinois	Springfield	6.3	11.4	23.0	8.2	13.9	27.0	6.2	13.9	31.9
	Herrin	6.1	11.4	25.6	7.0	13.6	23.7	6.6	13.4	30.6
	Danville	6.4	11.3	23.0	7.9	14.2	24.3	14.1	92.9	699.5

Table 68. Option 3 Methane mitigation costs for underground mines ($/Ton of coal produced)

		Longwall			Continuous		
Region	**Coalfield**	0.05	0.5	0.95	0.05	0.5	0.95
Colorado Plateau	South Piceance	18.9	26.4	47.3	19.6	28.6	39.6
	South Wasatch	18.9	26.2	47.3	19.3	28.3	40.3
	Yampa	18.9	26.2	47.4	19.5	28.2	39.7
	Henry Mountains	18.9	26.4	47.4	19.4	28.7	40.4
	San Juan	18.9	26.3	47.5	19.6	28.1	39.9
Rocky Mountains and Great Plains	Ashland	18.9	26.1	47.3	19.2	27.8	39.2
	Colstrip	18.9	26.2	47.3	19.4	27.8	39.3
	Decker	18.9	26.1	47.3	19.2	27.8	39.2
	Gilette	17.4	26.1	46.4	18.2	26.6	43.5
	Sheridan	17.4	26.1	46.4	18.2	26.6	43.5
	Williston-Beulah-Zap	17.5	26.1	46.4	18.2	26.7	43.8
	Williston-Hagel	17.4	26.2	46.4	18.2	26.6	43.5
	Williston-Hansen	17.5	26.1	46.4	18.2	26.6	43.5
	Williston-Harmon	17.4	26.2	46.4	18.2	26.6	43.8
	Carbon-Johnson	17.4	26.1	46.4	18.2	26.6	43.5
	Green River-Dead Man	17.4	26.1	46.4	18.2	26.6	43.5
Gulf Coast	Wilcox	17.5	26.1	46.4	18.3	27.3	43.9
	Lower Wilcox	17.4	26.3	46.7	18.2	27.2	43.8
Appalachia	Pittsburgh	17.4	26.9	43.1	18.3	28.9	40.6
	Upper Freeport	17.4	27.2	42.4	18.4	28.3	40.6
	Lower Kittaning	18.2	27.6	43.1	21.4	31.2	44.2
	Pond Creek	17.6	27.1	42.4	19.9	29.6	42.0
	Fire Clay	17.4	27.1	42.5	18.7	28.9	41.8
	Pocohontas	17.3	27.2	42.4	18.2	28.8	41.7
Illinois	Springfield	18.0	27.6	43.0	20.6	30.7	43.5
	Herrin	17.6	27.1	42.7	19.6	29.5	44.1
	Danville	18.0	27.6	42.9	20.6	31.2	43.6

Table 69. Option 4 Methane mitigation costs for underground mines, $/ton of coal produced

Region	Coalfield	Longwall			Continuous		
		0.05	0.5	0.95	0.05	0.5	0.95
Colorado Plateau	South Piceance	18.9	27.8	46.4	19.8	28.6	44.2
	South Wasatch	18.6	27.6	46.0	19.2	28.4	44.1
	Yampa	18.7	27.6	45.8	19.3	28.6	44.3
	Henry Mountains	18.7	27.6	46.3	19.8	28.7	44.2
	San Juan	18.6	27.7	46.6	19.2	28.4	44.2
Rocky Mountains and Great Plains	Ashland	18.6	27.6	45.6	19.2	28.2	44.1
	Colstrip	18.7	27.6	45.6	19.2	28.2	44.1
	Decker	18.6	27.6	45.6	19.2	28.2	44.1
	Gilette	19.0	27.9	43.9	19.7	28.9	39.7
	Sheridan	19.0	27.9	43.9	19.7	28.9	39.7
	Williston-Beulah-Zap	19.0	27.9	43.9	19.8	28.9	39.8
	Williston-Hagel	19.0	27.9	43.9	19.7	29.3	39.7
	Williston-Hansen	19.1	27.9	44.0	20.0	28.9	39.7
	Williston-Harmon	19.0	27.9	43.9	19.7	29.1	39.7
	Carbon-Johnson	19.0	27.9	44.1	19.7	28.9	39.7
	Green River-Dead Man	19.0	27.9	43.9	19.7	28.9	39.7
Gulf Coast	Wilcox	19.1	27.9	44.1	19.9	29.3	39.7
	Lower Wilcox	19.0	27.9	44.1	19.8	29.3	40.1
Appalachia	Pittsburgh	18.5	28.0	49.7	20.2	29.8	43.3
	Upper Freeport	18.4	27.7	49.6	20.1	29.8	43.7
	Lower Kittaning	19.3	28.6	49.9	21.2	32.0	48.5
	Pond Creek	18.4	27.8	49.6	20.3	29.9	43.3
	Fire Clay	18.5	27.9	49.6	19.8	29.8	43.7
	Pocohontas	18.4	27.7	49.7	19.8	29.6	42.7
Illinois	Springfield	18.8	28.5	49.9	21.1	31.6	45.6
	Herrin	18.6	28.1	49.8	20.9	31.2	46.5
	Danville	18.9	28.5	49.9	21.5	31.9	45.6

Appendix 1

INFORMATION AND ASSESSMENT TECHNIQUES FOR ESTIMATING MINING COSTS

INFORMATION AND ASSESSMENT TECHNIQUES FOR ESTIMATING MINING COSTS

Project: Assessing Future Supply Curves for Coal in Light of Economic, Technological Uncertainties
Subtask: 404.03.01

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Introduction

This subtask is fully funded by the Department of Energy's National Energy Technology Laboratory. The project is managed by the Research and Development Solutions, LLC (RDS) at Morgantown, WV.

As it has been envisioned by many that coal will continues to play a predominant role in the Nation's diverse energy supply in the future. A good inventory of the national economically and technologically recoverable coal resources can support this envision. This work assesses the implications of continued coal use, as resources are consumed, and mining is undertaken under more challenging conditions than those that exist today. The analysis will evaluate the cost of mining coal in consideration of 1) more advanced technologies and 2) techniques to mitigate environmental impacts that may arise from extraction in the future. This research project is a continuation of a collaborative effort undertaken by Carnegie Mellon University (CMU) and the National Energy Technology Laboratory (NETL). In the second year, the Department of Mining Engineering at the West Virginia University (WVU) has joined the team. The WVU team provides their mining professional knowledge for determining the mining costs. In whole, this effort is developing strategies/models to estimate the cost to extract coal from myriad U.S. coal regions.

During the project period, the WVU party provided needed cost information, developed methodologies that can be programmed in the cost modeling program, discussed cost issues with CMU party, and reviewed reports and draft technical papers. In this report, these research efforts are briefly listed and some of the results are presented.

[1] Associate Professor, Ph.D. and P.E.
[2] Graduate Research Assistant

Some Basic Cost Data

In the early stage of the project, the authors provided CMU party with the following cost data for implementing in the cost simulation program:

- Some typical capital costs of longwall and continuous mining equipment. Typical costs for longwall as well as room and pillar mines. Typical production cost per ton for the two mining methods. Typical capital costs for coal preparation plants and the preparation cost per ton. The data are shown in Appendix A.
- Subsidence related issues: Basics about subsidence influence zones as well as the typical costs for mitigating common surface structures (residential property and interstate highway). Based on our subsidence experience with various surface structures and features, suggestions to remove some of the restraints to mine under some of the highways and small streams have been made. The average costs for mitigating and litigating subsidence effects are shown in Appendix B.

Development of Programmable Methodology to Estimate Recovery Ration for Room and Pillar Mining Operations

In underground coal mines using room and pillar mining method, the pillars should be adequately designed in order to prevent underground mine structural failures and to avoid surface subsidence. In order to have the highest pillar strength as well as the highest recovery ratio at a given geological and mining condition, the best way is to design square pillars in plan view. Improvement on method to determine the recovery ratio of room and pillar mines. In the original mine cost estimation program, a fixed but unrealistic 44% recovery ratio for room and pillar mines was assumed. To provide a more realistic method to determine the recovery ratio, the coal mine pillar design method is simplified for the ease in programming. The recovery ratio is mainly a function of the mining depth and height. The pillar The empirical equations for determining the recovery ratio for mining heights from 3 to 15ft, the range of mining heights in the US underground coal mines, have been provided to CMU team.

The size (i.e., the side length, W, ft) of the pillars depends on the in-situ strength of the coal (σ_i, psi), the overburden depth (h, ft), room width (W_r, ft), mining height (H, ft) and the pillar safety factor (SF). The pillar strength is estimated using Bieniawski formula. The size of the pillar can be obtained by solving the non-linear equation. The derived method is shown in Appendix C.

Method to Estimate Mine Ventilation Cost

The cost to ventilate a coal mine is normally the second highest among the auxiliary operation as the cost for the ground control being the highest. The original CMU model estimated the ventilation costs using the methods in the Hard Rock Miner's

Handbook Rules of Thumb, 3rd edition (2003). However, the ventilation requirements for hard rock mines are very different from that for coal mines. Suggestion has been made to employ the ventilation cost estimation method listed in the SME Handbook. However, the SME method does not consider the differences in ventilation requirements for different rank of coal and mining depth. The WVU team modified the SME estimation method for ventilation requirements to consider the rank of the coal seams to be mined and the depth of the coal seams - two important factors related to methane emission rates into the mine workings. In developing this method, it is assumed that the ventilation requirement is proportional to the gas content. The measured gas content vs. depth curves for a number of major coal seams have been used to derive the empirical formulae for the correction factor to required ventilation air quantity for various coal seams. The developed method is shown in Appendix D.

Method to Estimate Mine Reclamation Cost Related to Mine Subsidence

In the effort to address the environmental impacts from mining, CMU group was planning to develop a finite element approach to predict mining subsidence. Extensive discussion about this plan was made between WVU researcher and CMU counterparts. WVU researcher stated his own experience and capabilities in subsidence studies. WVU researcher explained the various subsidence prediction methods and their respective advantage and disadvantages.

To further explain the topic of mine subsidence, WVU researcher also hosted Ms. Melissa Chan from CMU on April 23 for her inquiry about the current state of subsidence research. WVU researcher showed the parts of subsidence course materials that might be useful for the coal-cost model. A light version of our subsidence prediction model CISPM-LT has been shared to CMU party. Lecture materials of my graduate course "Surface Subsidence Engineering" have also been shared.

Reviewing Technical Reports and Papers

WVU team has participated in the validation work of the coal cost simulation model developed by the CMU team including reviewing of a number of preliminary and intermittent reports. After reviewing the preliminary reports from CMU team, suggestions have been made on the strategy and major mining parameters to be simulated with the model. After the CMU team has made the necessary changes in the models, the WVU team reviewed and commented the intermittent reports.

- After reviewing the preliminary CMU reports, it is found that some of the selected cases to demonstrate the coal cost simulation model were not well suitably chosen. The thicknesses of the coal seams used were too thin to be mined with underground mining methods and too deep to be mined economically with surface mining method. Through a number of email messages, suggestions on the simulation strategy and considerations in selecting the simulation sites have been communicated to the CMU team.

- Three CMU intermittent reports titled as “Continuous Mining Validation”, “Longwall Validation” and “Surface Mining Validation” were sent to the WVU team. The CMU reports have been critically reviewed, heavily edited and commented. The review placed emphasis on the correctness of the input information and on the explanations for some of the disagreements between the actual and simulation results. Based on the review, it seems that the coal cost models for underground mining methods are reasonably accurate. Some of the significant differences are caused by incorrect input data such as mining height and the number of continuous miner units or longwall united used in the mines. Therefore, by inputting correct mining parameters, the coal cost models for underground mining methods should produce fairly accurate simulation results. It seems that some of the simulation results from the model for surface mining could deviate, to significant degrees, from the actual results. However, in the light of the large ranges of surface mining machines available for surface mining operations, the differences can be considered reasonable.
- Reviewed two articles and other supplementary materials prepared by CMU researchers for publication in professional journals. The first article was planned to submit to International Journal of Mining, Reclamation and Environment. Extensive editing was performed on this paper. After an unsuccessful submission to that journal, WVU made recommendation to submit a revised version to the Journal of Computers and Geosciences. Suggestions on structure and contents for the paper have been made and editing of the revised paper has been made.
- A report of sensitivity study has been prepared by CMU researchers to address some of the inaccurate predictions on longwall mines and surface mines costs. Reviewed and inspected simulation results conducted by CMU researchers to fix the problems in the continuous miner and longwall simulation models. Some of the root causes for the inaccuracy of these two models have been found and the findings have been fed back to CMU researcher to fix the problems.

Acknowledgement

The authors of this report would like to appreciate the opportunities and financial support provided by the U.S. Department of Energy’s National Energy Technology Laboratory (DOE/NETL) to conduct the research works in the project. Thanks are expressed to Research and Development Solutions, LLC for their management of the project activities and funds. Appreciations should also go to CMU researchers for the friendly co-operations in conducting this project.

Appendix A: Mine Capital and Operating Costs

Single Section Capital Cost (387,000 CTPY = 430,000 RTPY)[3]

1-Continuous Miner	- $1,300,000
1-Continuous Haulage System	- $1,000,000
2-Shuttle Cars	- $350,000 (2) = $700,000
1-Roof Bolter	- $400,000
2-Scoops	- $350,000 (2) = $700,000
2-12 Passenger Mantrips	- $12,500 (2) = $15,000
2-2 Passenger Mantrips	- $5,000 (2) = $10,000
1-Belt Drive	- $250,000
1-Stacker Belt	- $100,000
1-Substation	- $250,000
1-Fan	- $150,000
1-Rescue Chamber	- $50,000
50-SCSR's	- $710 = $35,500
2-Utility Trucks	- $25,000 = 50,000
2-Rock Dusters	- $2,000 = $4,000
Feeder Breaker	- $200,000
Average roof bolt[4]	- $12
Total	**- $4,214,500**
Surface Facilities	- $3,175,000
Total Mine	**- $7,400,000**

Longwall Capital Cost (5.3 mm CTPY = 6.2 mm RTPY)[5]

Longwall System	-$60,000,000
3 Continuous Miner Sections	-$12,643,500
Total	**-$73,000,000**
Surface Facilities	-$327,000,000
Total	**-$400,000,000**

[3] Single section mine implies a drift operation operating a single section with a single continuous miner with pillar retreating.
[4] Roof bolt implies a fully grouted resin bolt of average length including resin and bolt plate.
[5] Longwall mine implies a mine utilizing one longwall shear supported by three continuous miner sections for development with surface facilities including rail spurs, loadouts, overland conveyors, and high capacity preparation plants not typically associated with small mines.

Single Section Operating Cost

Labor Cost	-$8.25 / ton
Supply Cost	-$10.00 / ton
Power Cost	-$2.00 / ton
Taxes and Information	-$4.00 / ton
Misc.	-$0.75 / ton
Total FOB Mine[6]	**-$25 / ton**

Longwall Mine Operating Cost

Average Total FOB Mine	**-$16 / ton**

Capital as a Function of Annual Production

Single Section Mine	-$17.2 / RTPY
Longwall Mine	-$64.5 / RTPY

Prep Plant based on 10 Million Tons / Year and $55 Million Capital Cost

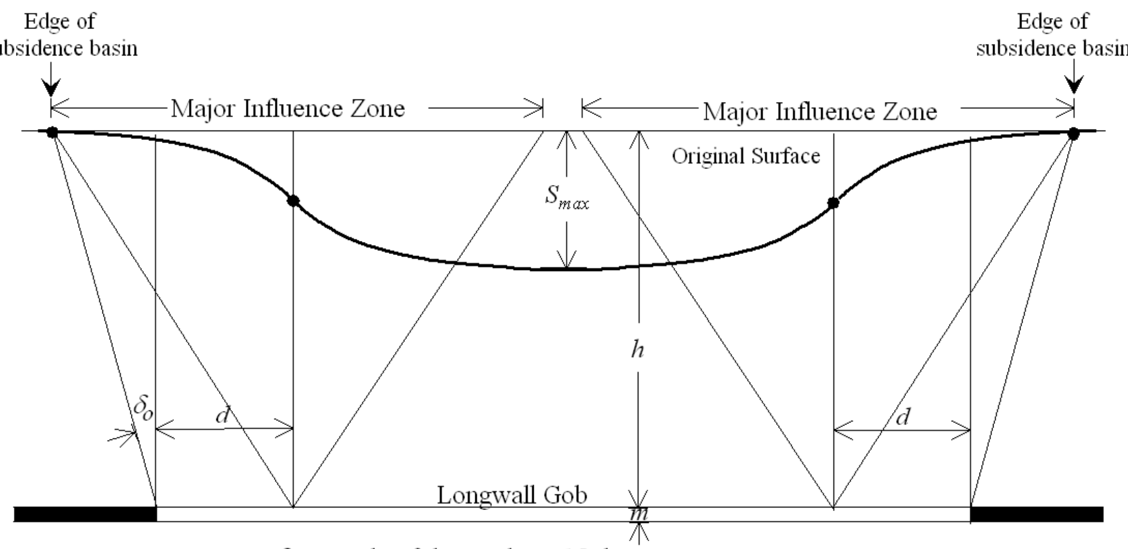
30 Year Mine Life	-$0.20 / ton
10 Year Mine Life	-$0.55 / ton
Plant Operating Cost[7]	-$2.00 - $5.00 / ton

[6] Total operating cost for single section mines utilizing continuous haulage will involve a lower operating cost than ones utilizing shuttle cars in a functionally identically mine environment.
[7] Total plant operating cost includes refuse disposal, magnetite replacement, and maintenance

Appendix B: Longwall Mine Subsidence and Associated Mitigation and Litigation Costs

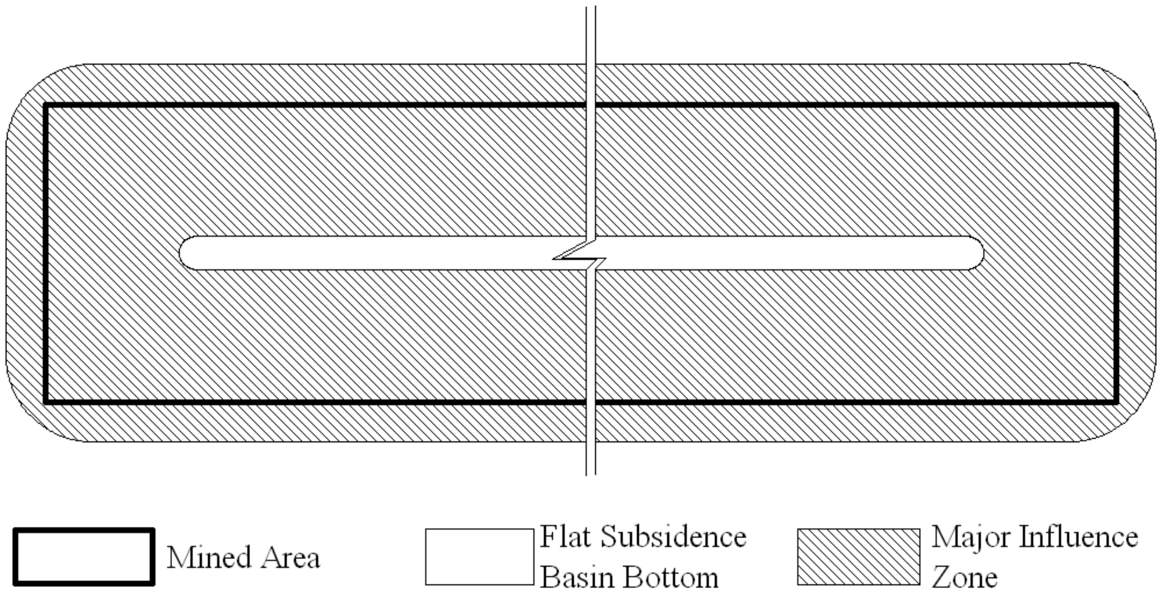
LONGWALL SUBSIDENCE

1. Conceptual Drawings



δ_o - angle of draw, about 15 degrees
h - overburden depth
S_{max} - maximum possible subisence
d - offset of inflection point, about 0.2 h
Width of major influence zone is about 0.6 h

Typical Final Subsidence Profile over Longwall Panel in Illinois Coal Basin



Typical Subsidence Basin over Longwall Panel in Illinois Coal Basin

2. **Average Subsidence Costs for Residential Structures (Based on Southwestern PA Experience)**[8]
   - **a.** Mitigation costs: $25,000 - $30,000 /home
   - **b.** Mitigation + repair + compensation: $50,000 - $60,000 /home (need to confirm)
   - **c.** Total budgeted costs for damages to home, disturbances to water, land etc. for each property: $250,000

3. **Average Subsidence Costs for Mining Under Interstate Highway**
   - **a.** About $70,000 per 100 ft linear length of highway (two-way, two-lanes each direction). For example, the estimated cost to mitigate and fix I-79 over each longwall panel with a distance of about 1,500 ft is about $1,000,000.

[8] These are the average cost numbers. The actual costs vary considerably depending on many factors.

Appendix C: Development of Methodology for Estimating Recovery Ratio in Room and Pillar Mine

ESTIMATION OF RECOVERY RATIO IN ROOM AND PILLAR MINE

by

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Department of Mining Engineering

West Virginia University

Updated: December 6, 2007

In underground coal mines using room and pillar mining method, the pillars should be adequately designed in order to prevent underground mine structural failures and to avoid surface subsidence. In order to have the highest pillar strength as well as the highest recovery ratio at a given geological and mining condition, the best way is to design square pillars in plan view.

The size (i.e., the side length, W, ft) of the pillars depends on the in-situ strength of the coal (σ_i, psi), the overburden depth (h, ft), room width (W_r, ft), mining height (H, ft) and the pillar safety factor (SF). The pillar strength is estimated using Bieniawski formula. The size of the pillar can be obtained by solving the non-linear equation in Eq. 1.

$$1.1 \cdot h \cdot \frac{(W+W_R)^2}{W^2} = SF \cdot \sigma_i \left(0.64 + 0.36\frac{W}{H}\right) \tag{1}$$

Once the size is determined, the recovery ratio (η) can be determined using Eq. 2.

$$\eta = 1 - \frac{W^2}{(W+W_R)^2} \tag{2}$$

Using the two equations above, the pillar size and the recovery ratio for a range of overburden depth have been determined and plotted in Figs. 1 and 2. The following common design parameters are used:

- Mining height: H = 3, 4, 5, 6, 7, 8, 10, 12 and 15 ft
- Room width: W_r = 20 ft
- Overall safety factor: SF = 1.2
- Coal in-situ strength: σ_i = 900 psi

Determination of Pillar Size

By solving Eq. 1, the required sizes of the square pillar (W) for each of the mining height (m) at varying depth (h) are plotted in Fig. 1. It should be noted that the minimum size of the pillar should be at least 3 times of the mining height in order to avoid slim pillars that forms the condition for cascading pillar failure (CPF). A CPF event is a rapid failure of pillars in a large area that could cause serious safety problem to a mining operation. In order to make the method to estimate the pillar size easily programmable, regression is performed on each of the derived curves. It is found that a power function (Eq. 3) will fit each of the curves the best. The derived coefficients *a* and *b* in the power function for each of the selected mining height are listed along with the R^2 value of the regression in Table 1. The high R^2 values show that the derived regression functions can accurately represent the data.

$$W(h) = A \times h^B \qquad \text{and} \qquad W(h) \geq 3 \cdot m \tag{3}$$

Table 1. Derived Coefficients *A* and *B* for the Power Function in Pillar Size Design with Safety Factor of 1.2

	m = 3'	m = 4'	m = 5'	m = 6'	m = 7'	m = 8'	m = 10'	m = 12'	m = 15'
A	0.4775	0.4193	0.3694	0.3273	0.3033	0.2715	0.2223	0.1920	0.1648
B	0.5994	0.6396	0.6750	0.7066	0.7302	0.7569	0.8039	0.8406	0.8826
R^2	0.9976	0.9979	0.9981	0.9984	0.9984	0.9987	0.9991	0.9993	0.9995

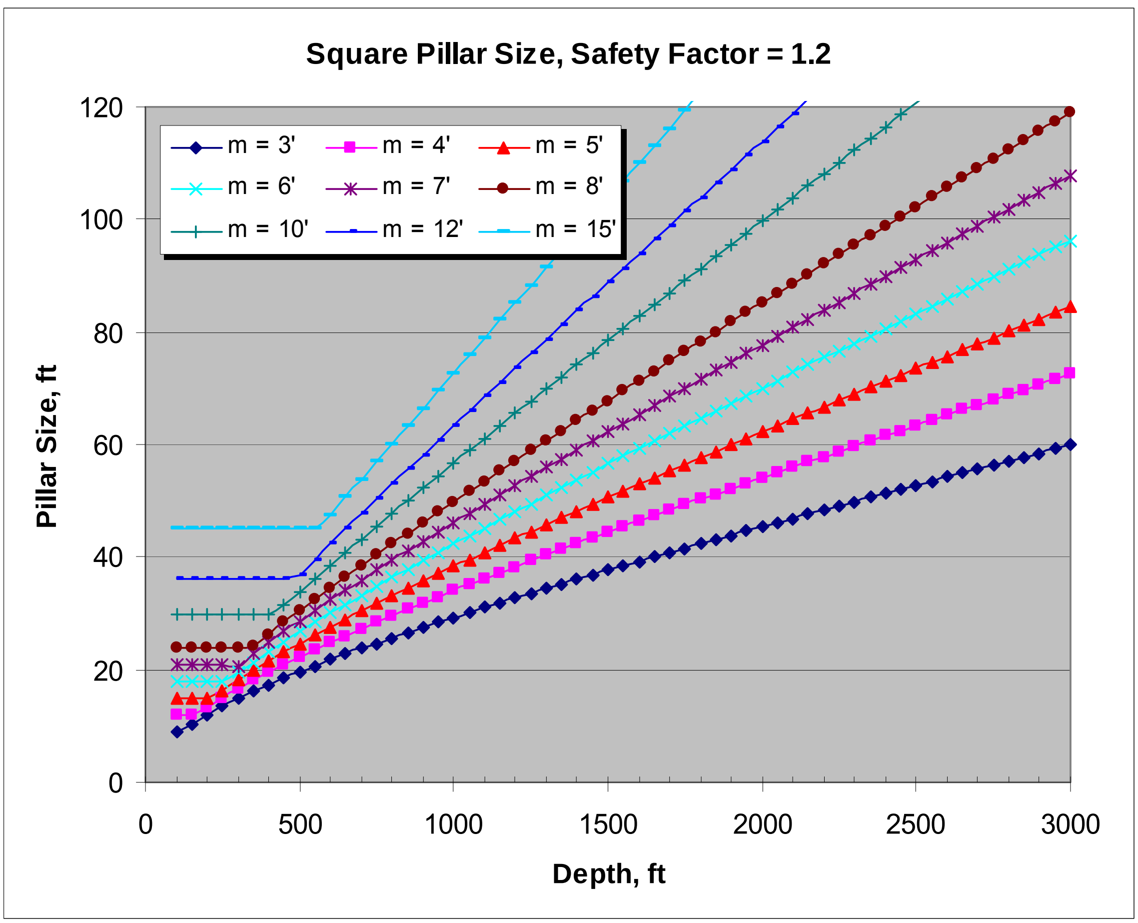

Figure 1. Sizes of the Coal Pillar at Difference Depths and Mining Heights

In order to make programming easier, the relationships between the mining height (*m*) and the coefficients *A* and *B* in the power function (Eq. 3) have been studied. It is found that the relationship between *A* and *m* can be well represented with the following vapor pressure model.

$$A(m) = e^{\left(a + b/m + c \cdot \ln(m)\right)} \tag{4}$$

An exponential association function can be used to accurately represent the coefficient *B*.

$$B(m) = a \times \left(b - e^{-c \times m}\right) \tag{5}$$

The coefficients for each of the two empirical functions are listed in Table 2 and the relationships are plotted in Figs. 2 and 3.

Table 2. Results of the Regression Studied for the Coefficients *A* and *B* in the Power Function

Regression Studies for the Coefficients	*Coefficient A*		*Coefficient B*	
	Vapor Pressure Model:		Exponential Association:	
	A=exp(a+b/m+c ln(m))		B=a(b-exp(-c m))	
	a =	0.94906	*a* =	0.54774
	b =	-1.84890	*b* =	1.84267
	c =	-0.97667	*c* =	0.09773
	R =	0.99940	*R* =	0.99980

m =	Original	Fit	Error	% Error	Original	Fit	Error	% Error
3	0.4775	0.4770	0.0005	0.10%	0.5994	0.6008	-0.0014	-0.23%
4	0.4193	0.4202	-0.0009	-0.21%	0.6396	0.6388	0.0008	0.13%
5	0.3694	0.3706	-0.0012	-0.33%	0.6750	0.6733	0.0017	0.25%
6	0.3273	0.3299	-0.0026	-0.79%	0.7066	0.7046	0.0020	0.29%
7	0.3033	0.2965	0.0068	2.23%	0.7302	0.7329	-0.0027	-0.38%
8	0.2715	0.2690	0.0025	0.91%	0.7569	0.7587	-0.0018	-0.23%
10	0.2223	0.2266	-0.0043	-1.92%	0.8039	0.8032	0.0007	0.09%
12	0.1920	0.1956	-0.0036	-1.85%	0.8406	0.8398	0.0008	0.10%
15	0.1648	0.1622	0.0026	1.59%	0.8826	0.8828	-0.0002	-0.03%

Example: Determine pillar size for a room and pillar coal mine that extracts a coal seam of 5.4 ft thick at a depth of 720 ft.

At the mining height of 5.4 ft, the coefficients *A* and *B* for the power function (Eq. 3) are determined, using Eqs. 4 and 5, to be 0.3533 and 0.6862, respectively. By substituting these coefficients in Eq. 3, the determined pillar size at the depth of 720 and a safety factor 1.2 will be 32.3 ft.

Determination of Recovery Ratio

Once the size of square pillar is determined, the recovery ratio in the mine can be determined using Eq. 2. However, it should be noted that a recovery ratio greater than 75% may not be practical. The estimated overall recovery ratios for various mining height (*m*) and depth (*h*) are plotted in Fig. 4. Using the previous example, the recovery ratio will be about 62%.

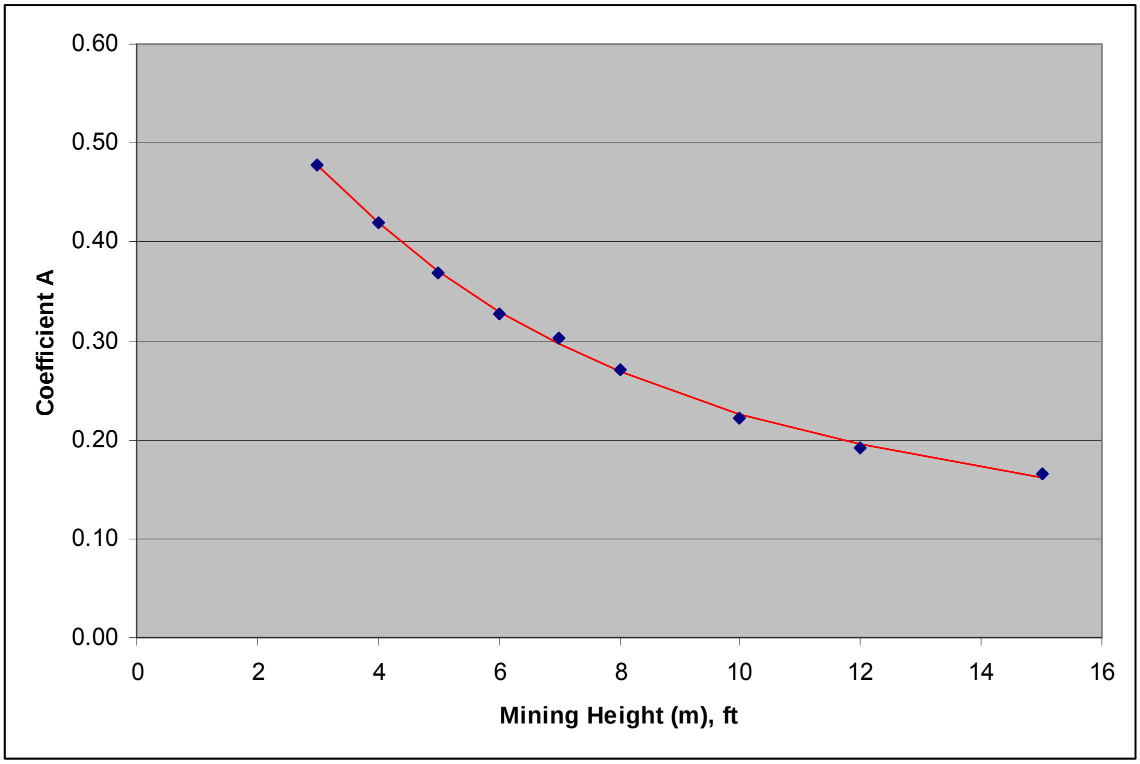

Figure 2. Relationship between Coefficient *A* and Mining Height (*m*)

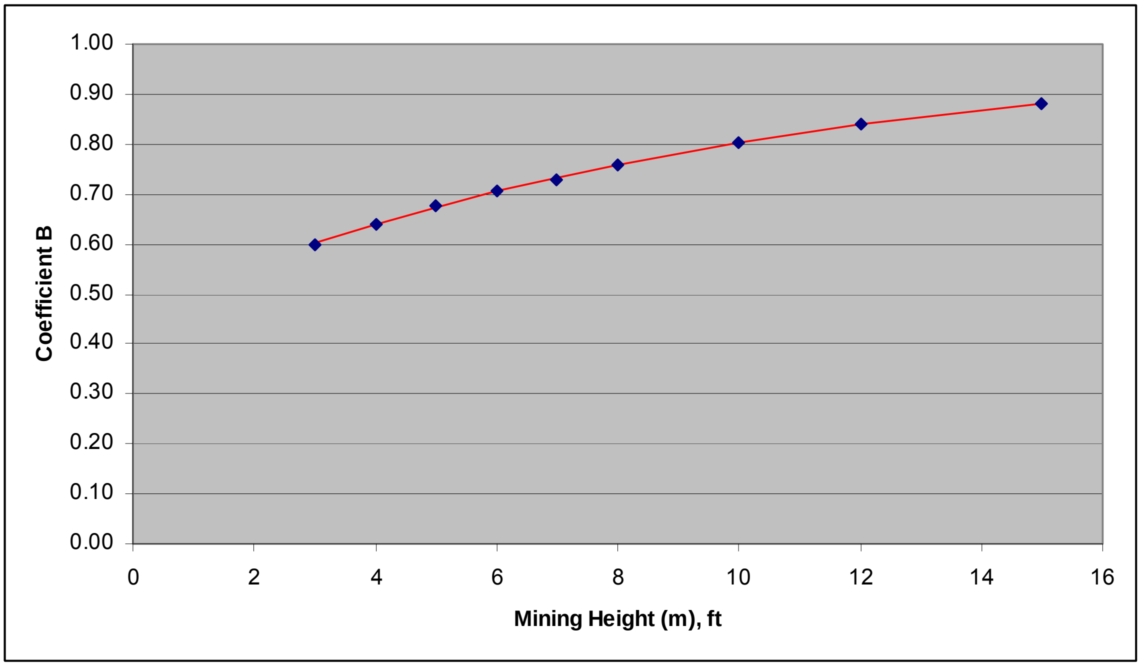

Figure 3. Relationship between Coefficient B and Mining Height (*m*)

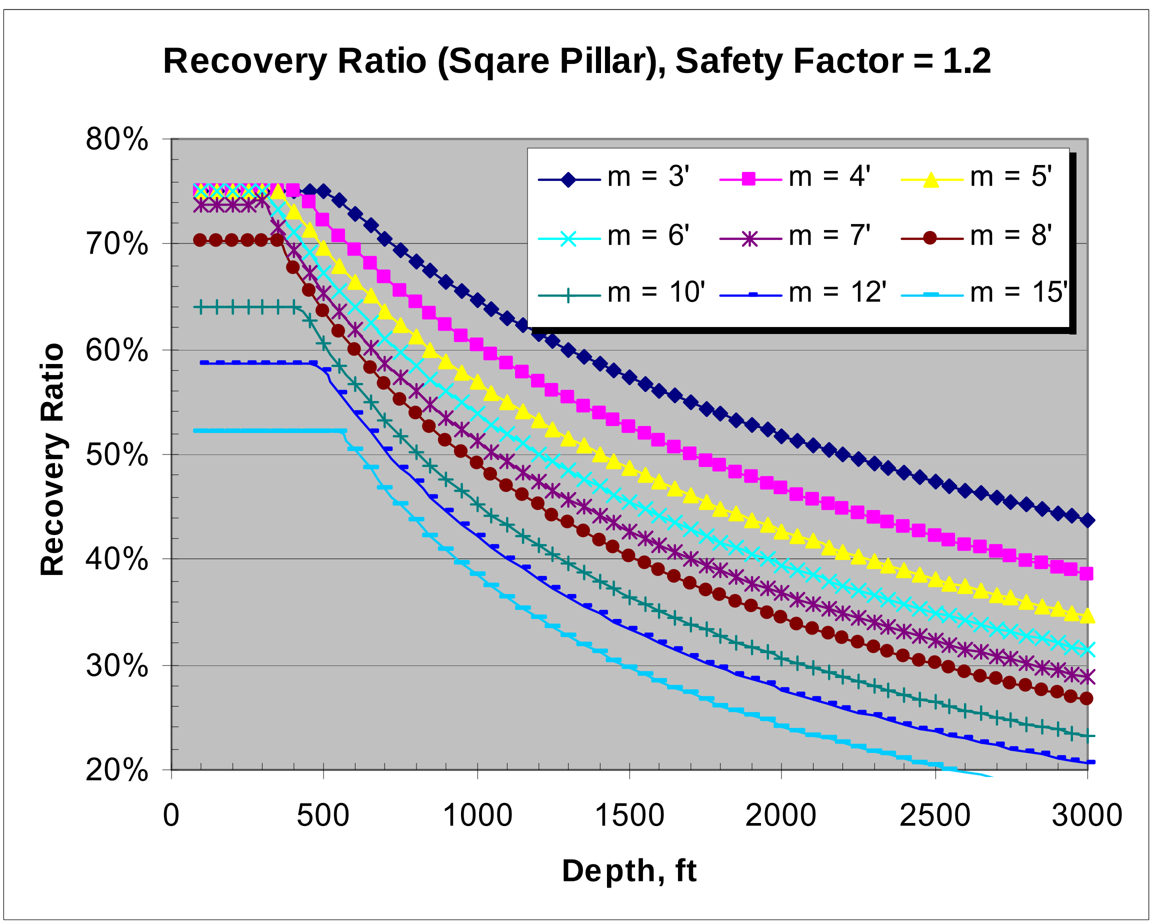

Figure 4. Recovery Ratios for Various Mining Height (*m*) and Depth (*h*)

Appendix D: Development of Method to Estimate Mine Ventilation Costs

ESTIMATION OF VENTILATION COSTS

by

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Updated: December 6, 2007

Mine ventilation cost could be one of the large cost items in a coal mine operation. The methods in SME handbook for estimating the ventilation costs are used as the base methods. The base costs are then adjusted by the coal seam (rank) and mine depth. Both capital and operating costs for ventilation system are based on the horsepower requirement for the mine which in turn is dependent on the gas emission and fan head needed.

Base Method for Estimating Ventilation Requirement

The ventilation requirement includes the quantity of the ventilation air (Q) and fan head (H). In SME handbook, these two values for coal mines are estimated using the following two equations based on daily production:

$$Q = 500 \times T^{0.8} \quad \text{cfm} \tag{1}$$

$$H = 2.4 \times T^{0.1} \quad \text{inches of water} \tag{2}$$

The installed fan horsepower (HP) is then estimated as

$$HP = \frac{Q \times H}{3800} \quad \text{hp} \tag{3}$$

Ventilation Capital Cost

In general, the most reliable method to estimate the capital cost for an installed ventilation system is the total installed horsepower (HP) of all ventilation fans in the system. For underground coal mines, the capital cost (C_c) is estimated by the following equation:

$$C_c = 7500 \times HP^{0.6} \tag{4}$$

Ventilation Operating Cost

The ventilation operating cost is mainly the electricity cost. Since the ventilation system will operate year around, the annual operation cost for a mine ventilation system will be:

$$C_o = 0.75 \times HP \times 24 \times 365 \times c \tag{5}$$

In this equation, c is the electricity cost per kilowatt-hour.

Based on this base method, the capital and annual operating costs for a coal mine ventilation system are calculated for normal range of underground coal mine production and plotted in Fig. 1.

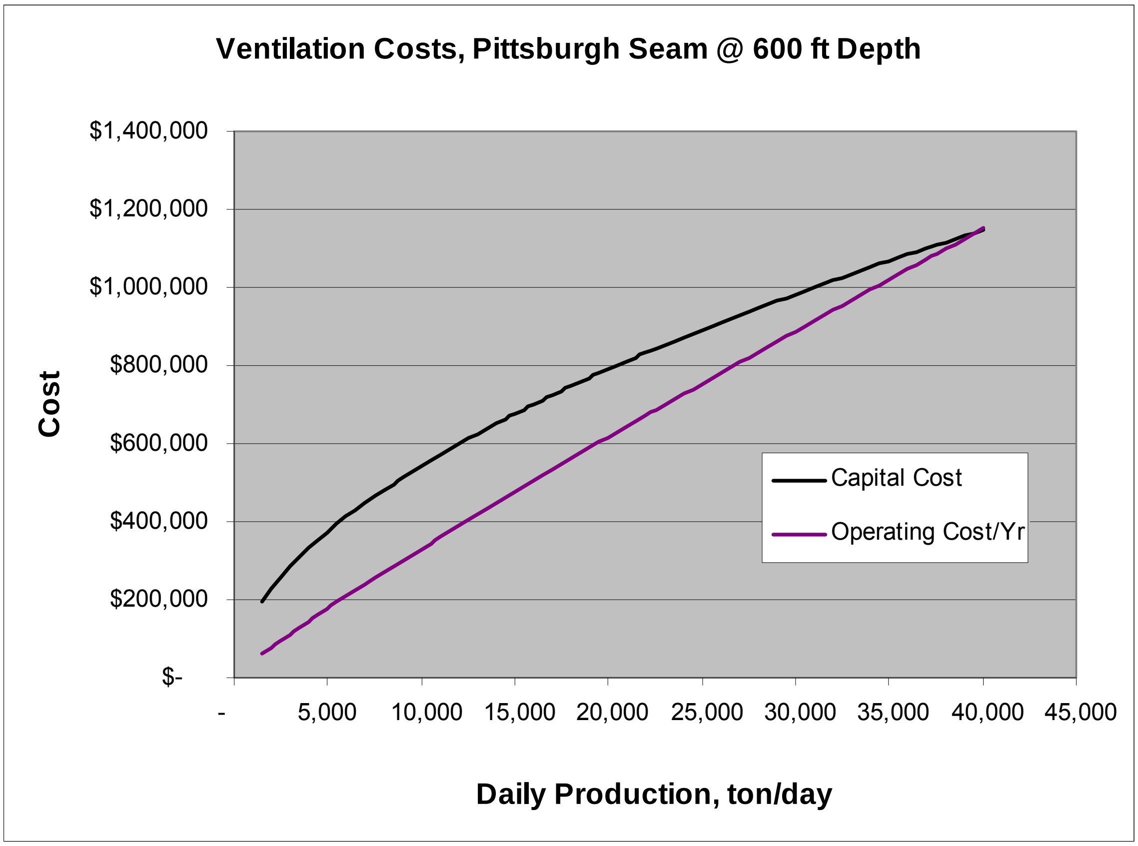

Figure 1. Base Capital and Annual Operating Costs of a Mine Ventilation System

Adjustments to the Ventilation Requirement

Underground coal mines are operated in difference coal seams and at different depth. These differences make the gas content in a ton of the mined coal vary considerable. The required ventilation air quantity is to dilute the emitted gas to the underground to safe levels. It is assumed in this study that the gas emission is proportional to the gas content. It has been demonstrated by studies that the gas content in coal is function of the coal rank and mining depth (related to the reservoir pressure). Therefore, the required ventilation air quantity (Q) should be dependent on the coal rank and mining depth.

Indirect Method to Estimate Gas Content

Figure 2 shows the gas contents of a number of major US coal seams as the symbols. An indirect method to estimate gas content (Y) in coal seam at a particular depth (h) is shown in Eq. 1.

$$Y = \frac{Y_c bh}{1 + bh} \tag{6}$$

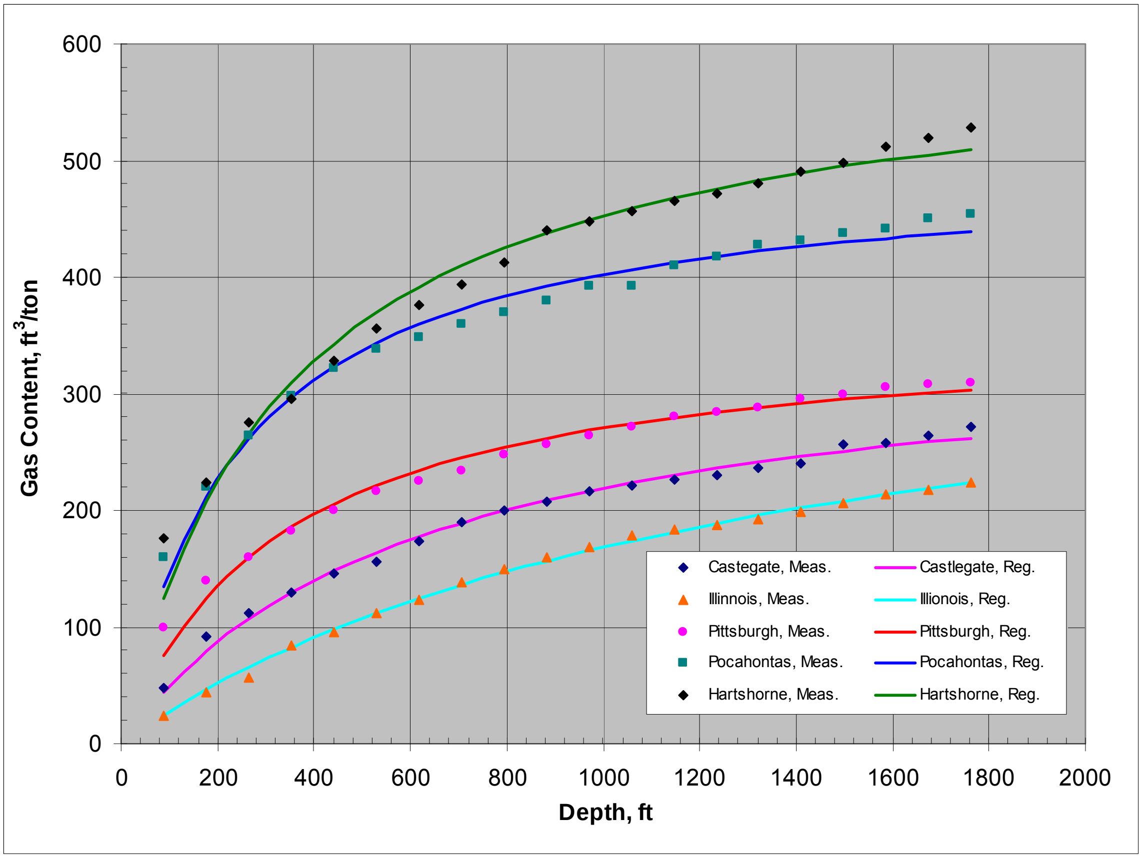

Figure 2. Gas Content in Major US Coal Seams

Nonlinear regressions have been performed on the measured data and the coefficients for the coal seams are listed in Table 1 and the regression curves are plotted back into Fig. 2. The high r values indicate good fitting to the data.

Table 1. Determined Regression Coefficients for the Selected Coal Seams

Seam	Castlegate Seam	Illinois No. 6	Pittsburgh Seam	Pocahontas No. 3	Hartshorne Seams
Y_c=	352.77	392.44	360.58	498.29	607.76
b=	0.00164	0.00075	0.00301	0.00419	0.00293
r =	0.9961	0.9990	0.9990	0.9907	0.9873

Correction Factor to Ventilation Air Quantity

It is assumed that the empirical equation for estimating the air quantity in the SME handbook is for the Pittsburgh coal seam at a depth of 600 ft. A correction factor should be applied for mining conducted in the other coal seams and at different depth. Based on this assumption, the empirical equation for the correction factors is shown in Eq. 7 and the coefficients in the empirical equation are listed in Table 2. The correction factors for different coal seams (S) and depth (h) are plotted Fig. 3. The adjusted ventilation quantity is then determined using Eq. 8.

$$C(S,h) = \frac{abh}{1+bh} \tag{7}$$

Table 2. Coefficients for Correction Factor for the Selected Coal Seams

Seam	Castlegate Seam	Illinois No. 6	Pittsburgh Seam	Pocahontas No. 3	Hartshorne Seams
a =	1.58	1.76	1.62	2.23	2.73
b=	0.00164	0.00075	0.00301	0.00419	0.00293
r =	0.9961	0.9990	0.9990	0.9907	0.9873

$$Q' = Q \times C(S,h) \tag{8}$$

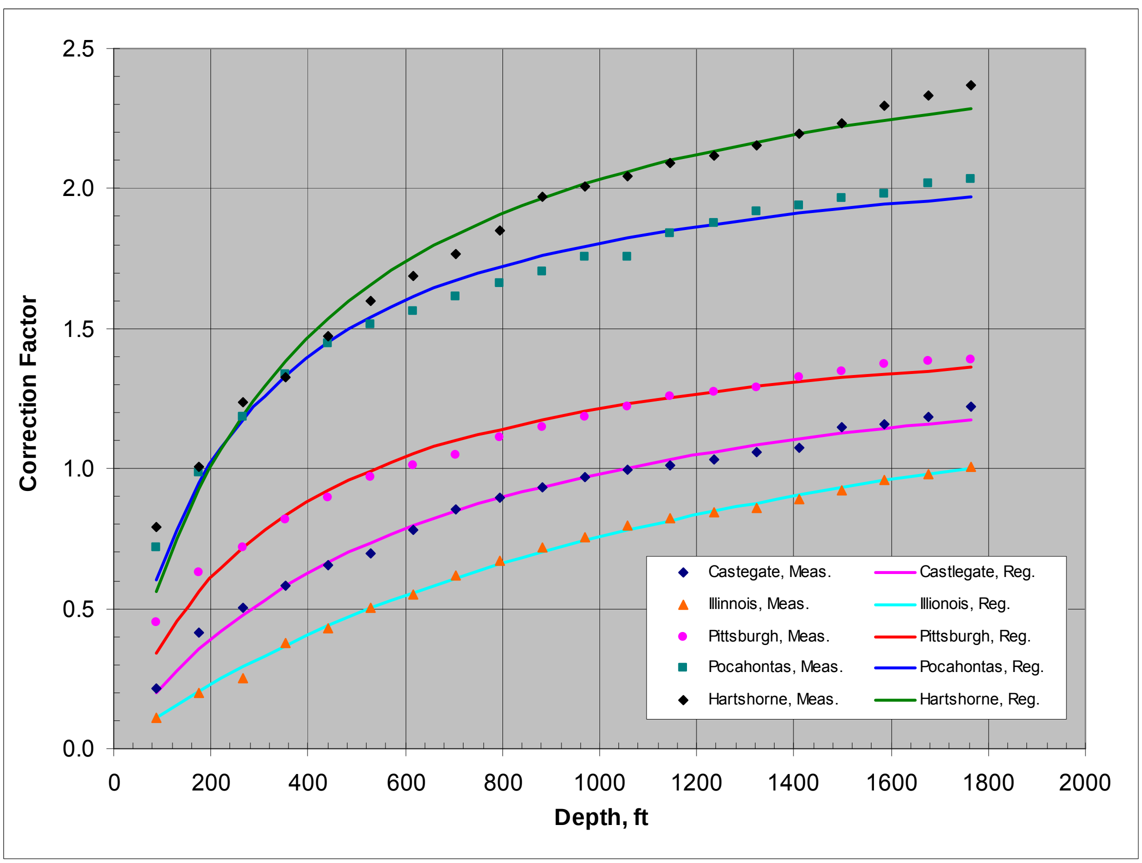

Figure 3. Correction Factors for Required Ventilation Air Quantity

Example: Determine the ventilation requirement and ventilation costs for a coal mine extracting in the Pocahontas No. 3 seam at a depth of 2,000 ft. The daily production is 10,000 tons and electricity cost is \$0.04/kw-hour.

- Based on SME handbook method, the ventilation requirement is determined

  Q = 792,500 cfm   H = 6.03 in. water
  $C(S, h)$ = 1.99
  Q' = 1,577,000 cfm
  HP = 2,502 hp
- Capital cost C_c = \$820,500
- Annual operating cost: C_o = \$657,630

Suggested Selection of the Models

Figures 2 and 3 show that data of gas contents are only available for five coal seams and five models have been developed for the six seams. Generally, the higher the rank a coal seam is, the higher is the gas content. In order for the users to select proper empirical model for the coal seams other than those five presented in Figs. 2 and 3, a model selection chart is shown in Fig. 4. The chart is complied from coal rank information from various sources for the four major coal producing regions in the US. In this chart, the left side shows the rank of the coal, middle portion shows the four major US coal fields and some of the major coal seams in the fields. The right side shows the suggested application ranges of the five empirical models.

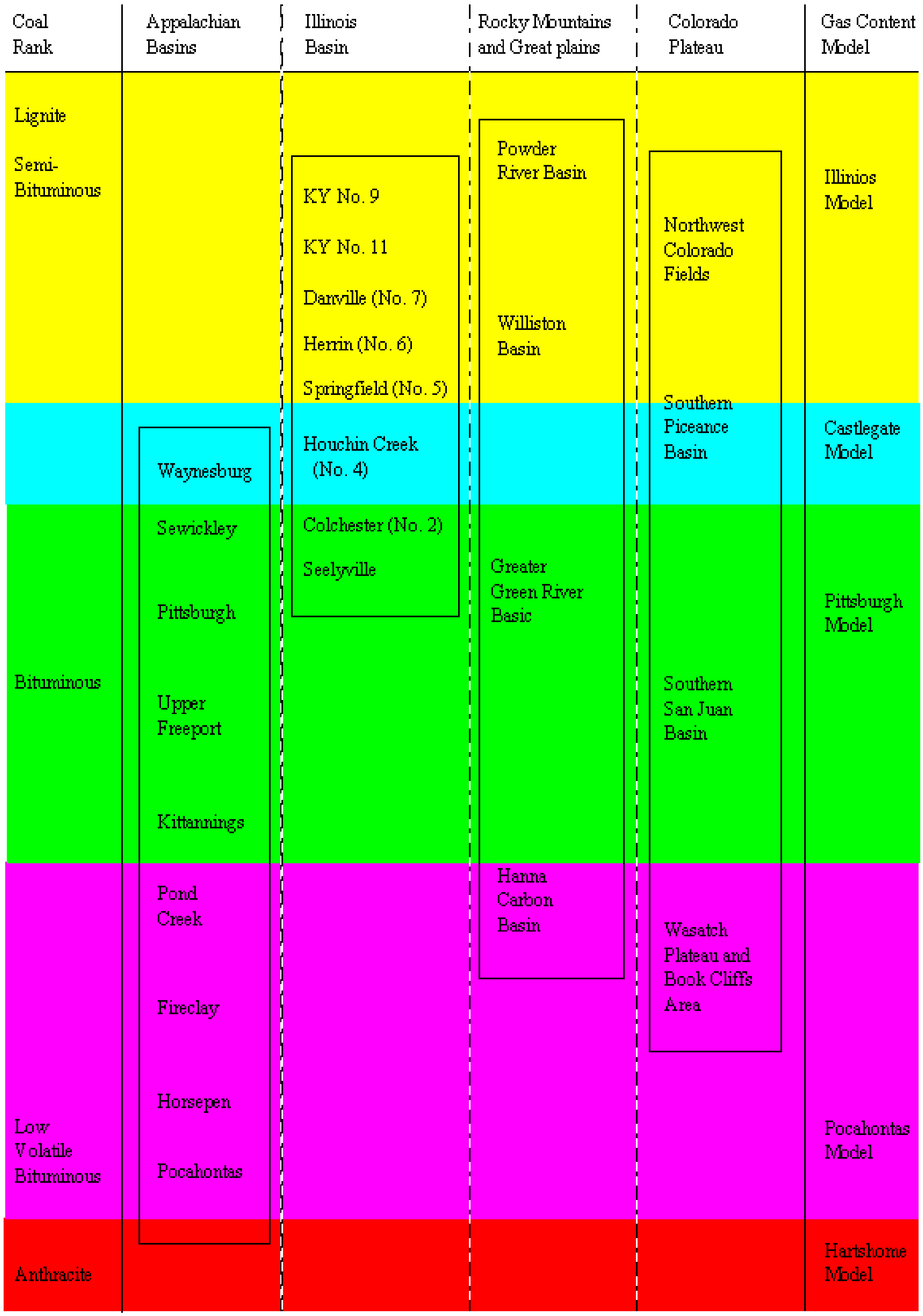

Figure 4. Model Selection Chart for Gas Content and Correction Factors

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