ECONOMIC AND SYSTEM FEASIBILITY STUDY
OF MUNICIPAL WASTE STOWAGE IN UNDERGROUND COAL MINES

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ABSTRACT

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Chapter 1

Problem Introduction

1.1. Introduction

Production from the Southern Appalachian coal region totals approximately 30 million tons of coal per year [Combs, 1995]. If converted to electricity (see Appendix A), $2.2 \times 10^{17}$ kW-hrs would be produced, which can provide power for over 21.1 million homes annually. The energy supplied by just one kW-hr can light a 100-watt lamp for 10 hours, lift a ton 1,000 ft into the air, smelt enough aluminum for a six-pack of soda cans, or heat enough water for a shower [Fickett, et al., 1990]. Thus, it can be seen how vital the Southern Appalachian coal region is to the United States. However, coal production from this region is in danger of plummeting, since the remaining reserves are in coal seams that are thinner and more difficult to mine economically than those mined to date. To counter the economic impact of these geologically impaired seams, a method of generating additional revenue for underground mining companies, particularly underground coal mining companies in the Southern Appalachian area, is urgently needed. The proposed solution will benefit not only the mining industry, but also the waste disposal industry and the environment. This study is preliminary and does not involve a complete chemical or physical analysis.
1.2. Mining

The two major problems facing the Southern Appalachian coal mining industry are decreasing seam thicknesses and increasing subsidence expenses. Roughly 80 percent of the remaining coal mining reserves in this region lie in seams that are less than 3 feet in thickness [U.S. DOE, 1991]. Mining thinner seams can result in a considerable increase in mining costs because although conventional mining equipment exists that will mine as low as 28 inches [Tredway, 1995] and longwall shearsers economically as low as 48 inches [Aaron, 1995], development expenditures remain relatively the same for all seam heights. Consequently, as mining takes place in progressively thinner seams, the same development outlay is necessary for less coal.

Concern with the second problem, subsidence (which had affected about 2 million acres of U.S. land by 1985 [Dowding, 1985]), has escalated because of the growing environmental movement, not because it occurs more frequently. Subsidence is defined as the lowering of the ground surface after an area has been undermined by a continuous extraction method. The cause of damage experienced on the surface is not from the simple lowering of the ground, but from transient step changes. These step changes act in the horizontal plane by both extending and compressing the object resting on the surface above the excavation such as a building. After a period of time, stabilization is reached which results in a long shallow depression with a smooth surface that extends in all directions beyond the limits of mining [Munjeri, 1987 and Bowman, 1991]. Recently,
companies have been forced by the U.S. Congress to pay for damage caused by surface movement resulting from underground mining by enforcement through the Surface Mining Control and Reclamation Act (SMCRA) [dePretoro and Shostak, 1993]. However, the biggest source of subsidence averision has been the change in community attitudes toward coal mining and ground depressions created by underground mining. In past years, when coal mining was viewed as a great necessity, mine subsidence was tolerated. In the current social climate, many aspects of the mining industry are viewed as disruptive, and this reflects today's greater environmental awareness [Holla, 1992]. Therefore, two forces, government and society, contribute to the mounting concern over surface disturbances caused by caving excavations. In summary, the mining companies need to alleviate the costs accrued by thinner seams and larger subsidence expenses.

1.3. Waste Disposal

The waste disposal industry also faces certain difficulties, such as a reduction in available space, an increase in waste production, and negative public perception. Most municipal solid waste is landfilled, 72% in 1992 [U.S. OTA, 1989], while the remainder is either recycled or incinerated. However, landfill space is consistently decreasing due to site closures. Between the years 1988 and 1992, the number of landfills saw a 32.7% reduction [Steutville and Goldstein, 1993]. The dwindling amount of landfill capacity, coupled with a surmounting quantity of waste produced by a growing U.S. population,
poses the problem of insufficient disposal space [U.S. EPA, 1991 and Stana, 1994]. The United States and Canada combined had an average annual population increase of 1.32%, which equates to 2.8 million people per year between the years 1950 and 1985 [Demeny, 1988]. The U.S. alone accounts for 90% of this expansion, and is considered to be in stage 3 of demographic transition [Getis, et. al., 1994]. The quality of life at this stage motivates people to desire more land and consume more products, which results in less land for refuse disposal and more waste in need of disposal.

In addition to diminishing space and increasing waste, another formidable problem faced by the waste disposal industry is negative public perception. Most people view landfills as rat-infested piles of garbage; in reality, strict regulations ensure sanitary conditions and require a daily cover. Society, nonetheless, is often easily influenced when it comes to problems with the environment or its surroundings, because people hold aesthetics very high on their list of priorities for happiness. This is demonstrated by the “Not In My Back Yard” (NIMBY) phenomena, where citizens do not want any eyesores or possible sources of injury located near them, such as landfills or prisons. Consequently, no matter where municipal waste is sent, there will be civic outcry. Concern will not be as great, however, if the refuse can be hidden from public view.

In spite of the negative aspects associated with the waste disposal industry, there has been at least one positive program. Recycling jumped from 9% in 1989 to 14% in 1991 and again to 17% in 1992, resulting in a lower percentage of total garbage going to
landfills [Steuville and Goldstein, 1993]. Recycling programs have become more popular
with homeowners than predicted in the late 70's [DeCesare, 1995]. Although a lack of
secondary markets for the recycled material persists [Schultz, 1995], these markets have
expanded over the past years, due in part to a national goal targeting this problem [U.S.
EPA, 1991]. Nevertheless, the waste disposal industry could benefit from increased
space and inconspicuousness.

1.4. Joint Venture

It would be beneficial to find one solution to solve the problems of both the
mining and solid waste disposal industries. Such a solution is to stow non-hazardous
municipal solid waste underground in the openings produced by mining operations as the
voids are created. This would provide the additional income needed by mining companies
as well as added support for subsidence reduction. Furthermore, the waste disposal
industry would be given more space to utilize, and aesthetics would improve due to
subsurface waste disposal.

A large number of subterranean cavities (some created by mining) have been used
in many different ways. Some recent examples include growing mushrooms [Stauffer,
1974 and Bennett-Smith, 1978], cryogenic storage [Dreyer, 1982], radioactive waste
storage [Anon., 1995], and hydrocarbon storage [Bradt, 1972]. The latter was
specifically implemented at the former Leyden coal mine near Denver, Colorado. But, by
far, the most sizable use of underground excavations has been in Kansas City, Missouri, where subsurface warehouses cut in the limestone rock are common [Stauffer, 1974].

Thus, it is not unusual for subterranean cavities to be used successfully. Furthermore, these results suggest the viability of using underground voids to alleviate the previously mentioned difficulties of both waste disposal and high mining costs.
Chapter 2

Mined Space Previously Used for Disposal Purposes

2.1. Introduction

Underground waste disposal has been implemented in a number of places, so many case studies exist on the subject. This chapter includes examples of radioactive, surface, solution cavity, and longwall stowage areas, to illustrate some of the different methods previously used as well as site economics.

Refuse is typically disposed in operating rather than abandoned mines, primarily because of health and safety issues. In the past, regulations required coal mine operators to seal shafts before deserting them; therefore, miners would dump refuse materials such as metal and wood down the openings to help fill them without regard for environmental consequences [Buttens, 1978]. Today, any type of disposal, particularly underground, is highly monitored. Even with the strict regulations, nuclear repositories are being constructed, and fluid injection into deep wells is being used. Fluid injection, which is showing great success, involves injecting liquid waste, predominantly industrial, through a borehole to depths up to 3,000 feet [Breeden, 1972]. The following examples, most from within the past twenty years, all relate in one form or another to the solution proposed in this paper, whether by geology, type of discarded material, or disposal technique.
2.2. Radioactive Waste Examples

The following two case studies address low, intermediate, and high-level waste disposal methods which have more stringent constraints than the municipal waste disposal being considered in this research. However, the stowage of radioactive material provides insight into the disposal of municipal refuse. Although the two materials do not compare economically, if selected stowage techniques are acceptable for nuclear waste, then it follows that they should be more than satisfactory for household trash. Radioisotopes are not as mobile as components in household waste, but radioactive disposal methods have been designed to keep any type of waste in place.

The first study was implemented in Germany in 1976 using a technique that addressed containerless disposal by utilizing a waste-binder mixture as depicted in Figure 2.1. Small percentages of radioactive substances, between 6.75% and 7.05% by weight, were combined with grout and pumped down a borehole into a cavity, which was solution mined (a brief definition of solution mining is found in Sect. 2.4.) for the sole purpose of low and intermediate-level nuclear waste disposal [Kraemer and Kroebel, 1989]. The study found that the process was feasible for radioactive matter, but that costly additives made it uneconomical for the disposal of municipal refuse. It may be more practical today due to increased environmental worth, but no studies have been completed on this subject.

The second example involves openings in bedded salt with a room and pillar
Figure 2.1 In-situ cavern system. (Kraemer and Kroebel, 1989)
layout, rather than salt domes that have been solution mined. Room and pillar mining leaves roughly 50 to 60 percent of the rock in place when pillars are unrecovered and continuous mining equipment is used [Lucas and Haycocks, 1973]. A designated grid pattern contains: pillars, which are columns of rock left in place; rooms, which are mined areas; and panels, which are a series of rooms used for production purposes. A typical room and pillar operation is depicted in Figure 2.2, which is a streamlined perspective of the Waste Isolation Pilot Plant (WIPP) in New Mexico, a test site for a nuclear repository, where TRU stands for transuranic [Dreyer, 1982]. As with all nuclear stowage areas, the test site is highly governed. The severe restrictions have caused development of elaborate mine plans and special waste storage containers. Acceptance of refuse has not yet started, because there remain some experiments to be conducted at the site, such as thermal alteration tests. Specifically at the Yucca Mountain high-level repository the mine and container designs account for the large thermal output expected from the radioactive material, which would be considerably less for municipal waste. Moreover, at the transuranic or low-level waste repository, WIPP, the containers have been designed to keep any radioactivity from escaping into the environment, which also included demanded assurance that no chemical alterations would occur. Although this should also be a concern for household garbage, the high costs associated with such an assurance would make it very impractical for a non-hazardous or non-radioactive substance. In addition, household waste, unlike radioactive substances, will not
Figure 2.2 Simplified view of the WIPP facility. (after Loomis and Wallace, 1993)
chemically alter its surroundings. One major difficulty has been transporting the waste matter to the final destination, since the public is deeply concerned about radioactive substances traveling through its communities. This will likewise be a consideration for municipal refuse, but public concern should not be as severe, due to the lower health risk posed.

2.3. Surface Examples

Mines which are geologically similar to the model site provide tentative disposal potential, regardless of whether they are surface or underground. The SMCRA has inspired numerous innovative approaches towards reclamation, including transforming excavations into school playing fields [Hodel v. Virginia Surface Mining and Reclamation, 1981], recreational lake areas [Short, 1983], and landfills [Carrier, 1978]. The following case study illustrates this initiative. A hypothetical model was proposed where municipal waste was stored in operating strip mines [Carrier, 1978]. Figure 2.3 shows an isometric view of an open cast mine. These open pit coal mines typically use draglines to remove overburden, i.e. material with no economic value, and shovels to load the coal into large haul trucks. The overburden can be pre-blasted to make removal easier for the dragline, which sits on the highwall. In addition, instead of hauling the overburden to a waste dump, it is simply placed in a spoil pile next to the dragline on top of the depleted coal area. These piles are in turn leveled and then reclaimed. Through mathematical
Figure 2.3 Main components of open-cast (strip) mining in a single-seam. (Hartman, 1987)
analysis, Carrier found that stowing waste in openings made by coal mining companies is a feasible alternative. Similarly, Emrich, in 1979, suggested that abandoned strip mines were suitable landfill sites which provided an option for solving the waste disposal problems of major cities. Consequently, underground coal mines with equivalent geology to open cuts may also prove to be economical for waste disposal, though emplacement costs are considerably higher.

2.4. Solution Cavity Examples

Solution cavities can provide valuable information on the economics of underground stowage of municipal waste. Solution cavities are salt domes that have been excavated by a simple and economical in situ technique portrayed previously in Figure 2.1. A borehole is drilled from the surface, and fluid, generally water, is pumped down the borehole and then recovered, thus the term “solution mining.” As the process is repeated, more salt is extracted from the ground, and an elongated cavity is created. Figure 2.1 also shows a cavity being filled with waste and a completely filled cavity. The drilling and solution mining steps are shown on the right side of the figure.

The following case study, conducted in Texas in 1976, is analogous to the one discussed in Section 2.2.; however, it is more appropriate for municipal refuse since it uses no additives. A cross section of the site is shown in Figure 2.4. As more waste was pumped into the cavity, the material below became compressed. Furthermore, a system:
Figure 2.4 Refuse disposal in solution-mined salt cavities. (after Rogers, 1974)
for methane recovery was installed. Because the material was sealed within an 
impervious salt dome, it was not considered hazardous. The cavity dimensions were 
2000 feet in height by 200 feet in diameter (i.e. 10 to 1 ratio), and the cavity was located 
500 feet below the top of the salt dome, as seen in Figure 2.5. A total of 6.08 million tons 
of municipal waste was stowed within four cavities, a fraction more than the average 
landfill capacity (refer to Chapter 3 for information on landfills), at densities equivalent to 
those found at landfills: 50 lbs/ft³ or 1,350 lbs/yd³; moreover, the site occupied only 33 
acres of surface area, which is an order of magnitude lower than that for a landfill. Not 
only was there a reduction in land use, but cost also declined. The economic analysis 
assumed that recycling of any valuable materials from the waste stream would offset any 
pre-disposal expenses; however, this analysis was made when more recyclables were 
present in municipal refuse. The cost of disposal was $122,212,600 (in 1994 dollars) for 
all four cavities [Rogers, 1974]. Figure 2.6 displays the comparisons of land usage and 
expenditures between a typical landfill and this solution cavity method. An analogy can 
be made since the volumes of waste involved in each disposal method are roughly equal. 
Cavity stowage costs are $44 million less than landfill expenses, while cavity stowage 
uses 94% less surface area. However, a more accurate comparison of the underground and 
surface operations is between the $/ton values of the two operations, because the volumes 
are not identical. The overall cost reveals a difference of 26.3%, whereas the $/ton values 
yield a difference of 27.7%. In summary, the volumes are close enough to yield an
CAVITY VOLUME $\approx 11.3$ Mill. Bbl.

DISPOSAL CAPACITY = 65 Mill Tons
(for ultimate density = 50 lb/cu. ft)

TOTAL DEPTH = 3000 ft

SALT OBERBURDEN = 3 ft

CAVITY RADIUS

33 Acres

CAVITY DIMENSIONS - 200' De x 2000' High
CAVITY VOLUME = 11.3 MILLION BARRELS
DISPOSAL CAPACITY (4 Cavities) = 6 MILLION TONS
(for ultimate density = 50 lb/cu. ft)

Figure 2.5 Cross section and plan view of refuse disposal project. (Rogers, 1974)
Figure 2.6 Land and cost comparisons between surface and underground waste disposal systems.
accurate comparison without conversion. Thus, the benefits of subsurface stowing can clearly be seen.

2.5. Longwall Examples

Longwall mining, the largest tonnage extraction technique used in the Appalachian region, uses a shearer that cuts approximately a two foot slice of material with each pass across the coal face [Peng, 1986]. The shearer rides along an armored conveyor that connects to a series of shields which separate the caving region from the face workings. An isometric cutaway of a panel of coal being mined is shown in Figure 2.7. A simplified cross section that shows a shield supporting the immediate roof is seen in Figure 2.8, and it can be seen that the immediate roof crumbles behind the shields, whereas the first and second main roofs fail in large blocks. The failure mode is very site specific, but all sites have immediate and main roofs. Figure 2.9 details the plan view of a longwall operation with several panels and panel entries where pillars of coal are left in place to support the area until mining of the section has commenced. After the shearer makes a traverse, the shields that support the roof are moved forward behind the machine. The shields move forward individually or in small groups (3-4 sections at a time) and are shown in Figure 2.10. The longwall mining method is highly automated and has greatly increased the productivity of underground coal mines. Due to the widespread use of this mining technique, the following examples are the most useful in the development of the proposed
Figure 2.7 Isometric cutaway view of longwall mining. (Peng, 1986)
Figure 2.8 Cross section of longwall mining. (Peng, 1986)
Figure 2.9 Plan view of longwall mining. (Peng, 1986)
Figure 2.10 Longwall shields being moved forward. (National Coal Association, 1988)
model.

In 1991, British Coal constructed an experimental disposal system in which waste rock and tailings were injected behind a longwall using two different techniques: full height stowing and trailing pipes [Astle, 1991a and Astle, 1991b]. The trailing pipe method involves extending pipes into the gob region after caving has occurred and then pumping backfill through them to the injection point. British Coal determined that bentonite in water was the optimum carrier of particles. More important however, was that there was sufficient water absorption by the broken material to make the sludge solidify quickly; therefore, there should not be a problem with water after emplacement takes place in the gob region, the area where caving occurs. As a result, no expenditures will be required for pumping excess water from the underground workings.

A second example, from Sill and Mez, 1994, concerns the disposal of residual material in the gob region. The system contains hydraulic transport, to utilize gravity, and trailing pipe emplacement, which is designed to fill the voids in the debris and not the gob directly. A cross section and plan view of trailing pipe stowage is shown in Figure 2.11. This disposal technique was also implemented in the previous example; however, in this case, the trailing pipe lays roughly 0.3 m above the floor and extends around 15 to 20 m into the gob. In the model proposed in this study, hydraulic transport will be fundamental, and the complete void behind the longwall is expected to be filled, not just the voids in the fallen material. Sill and Mez also reported that, in Germany, residues
Figure 2.11 Cross section and plan view of trailing pipe emplacement technique.
(after Sill and Mez, 1994)
from coal combustion have been approved for cavity filling, but only if specifications set
by the mining authority (i.e. no toxic, radioactive, or organic contaminants, etc.) are
satisfied. In addition, since early 1993, other residuals, such as used foundry sands and
material from the incineration of household waste and sewage sludges, have been allowed
to be included in stowage suspensions. Nevertheless, the model proposed in this study is
concerned only with raw domestic refuse.

In 1975, Singh and Courtney completed a feasibility analysis on the pneumatic
stowing of mining wastes behind longwall shield supports. Brattice cloth and retaining
screens, termed “discharge walls”, were viewed as utilization mechanisms to keep the
emplaced amalgamation from invading the face machinery. A discharge wall is seen in the
cross section of Figure 2.12, which also includes a plan view of lateral pneumatic stowing
with the location of the brattice cloth. As the shields advance, the stowing pipeline will
move forward as well to keep pace with the longwall operation. The study of Singh and
Courtney more closely resembles the proposed method of disposal than the preceding
example, because the waste mixture completely fills the void created from the coal
removal. The former is designed to stow material before the roof collapses, whereas the
latter method carries out emplacement after caving, which does not provide significant
strata control.

It is important to note here that the pneumatic emplacement used in Singh and
Courtney’s study can not be used in the model proposed in the current work because it
Figure 2.12 Cross section and plan view of lateral pneumatic stowing. (after Singh and Courtney, 1975)
may create sparks which could lead to a methane explosion in coal mines. An adequate alternative is to use hydraulic means. Although pneumatic was first applied in Germany in 1924 because it was more economical than hydraulic [Singh and Courtney, 1975], now that the causes of methane explosions are known, coal mines must take safety into consideration and use hydraulic stowing. This technique not only eliminates sparks, but also provides roof support and purges gas pockets. Even though Singh and Courtney, 1975 discussed the possibility of including urban wastes in mine excavations, when the study by Sill and Mez, 1994 was completed, only residue from the incineration of household wastes was being accepted into the mines. However, this shows that some form of municipal refuse disposal has been contemplated.

Whereas the previous examples in this section address the methods of disposal, the following two studies primarily examine economics. The first case [Adam, 1982] deals with an emplacement system that contains both hydraulic transportation and backfilling. Unfortunately, the study was completed only through above ground trials when lack of funding halted subterranean implementation. However, with the information gathered, the possibility of discarding mine waste underground at the site appeared to be technically and economically feasible [Adam, 1982]. Another point made in the analysis was that backfill is protected in the subterrane from oxidation; consequently, it should not react to form acid mine drainage (AMD) or create burning coal piles which can occur on the surface. The motivation of a similar study [HRB-Singer, Inc., 1980] was to
eliminate environmental problems faced by the coal mining industry due to coal refuse
surface dumping. An initial study was performed to target the troubled areas, and it was
found that the most serious environmental problems existed in the eastern United States,
especially in the Appalachian region. Burning coal piles are the main environmental
problem with surface disposal of coal refuse. As the problems of surface disposal grow,
the feasibility of underground emplacement increases; hence, subsurface stowage seems
more viable in this region. Although subterranean waste emplacement is not required by
any state or federal agency through regulations, the rules and regulations for it have been
promulgated in the *Federal Register* 44, no. 50, 13 March 1979. Specifically, Section
784.25 of Surface Coal Mining & Reclamation Operations Permanent Regulatory Program
in Pennsylvania [HRB-Singer, Inc., 1980]
Chapter 3

Municipal Waste

Municipal waste is a very complex substance, as several researchers, including Rathje [Rathje, 1991] and Emcon Associates [Emcon Associates, 1980], have mentioned. Therefore, this chapter differentiates between various wastes and defines the characteristics of the waste being used in this study. Waste tonnages, disposal methods, disposal costs, and site development costs are also discussed.

3.1. Classification

The focus in this research is specifically on non-hazardous municipal solid waste which is generated from households, even though many other waste types exist. Different types of waste include: municipal (non-hazardous), hazardous, and radioactive; gaseous residue, liquid, sludge (semi-solid), and solid.

A thorough explanation of solid waste is furnished by the Environmental Protection Agency (EPA) in Federal Register, 50 Fed. Reg. 614, Jan. 4, 1985, which takes 54 pages to complete [Percival, et al., 1992]. In general, according to the EPA, materials are considered solid wastes if they are “abandoned by being disposed of, burned, or incinerated; or stored, treated, or accumulated before or in lieu of those activities” [Percival, et al., 1992]. Furthermore, solid waste, defined more completely by EPA in Section 1003 (27) of the Resource Conservation and Recovery Act (RCRA), is:
any garbage, refuse, sludge from a waste treatment plant, water supply treatment plant, or air pollution control facility and other discarded material, including solid, liquid, semi-solid, or contained gaseous material residue from industry, commercial, mining, and agricultural operations, and from community activities (42 U.S.C. § 6903 (27)), but [this discarded material] doesn’t include solid or dissolved material in irrigation return flows or industrial discharge that are point sources subject to permit under the Federal Water Pollution Control Act (33 U.S.C. 1342), or source, special nuclear, or by-product material as defined by the Atomic Energy Act of 1954, as amended (6.8 Stat. 923) [U.S. EPA, 1991; Robinson, 1986; Percival, et al., 1992].

As previously stated, municipal waste, which is solid waste not regulated as hazardous waste, will be used in the proposed model. Hazardous waste is covered under Subtitle C of RCRA, which prohibits discharges that may harm “human health or the environment,” whereas municipal waste is encompassed in Subtitle D of RCRA, which has the same basis as the Subtitle C criterion, but allows for “consideration of the practicable capabilities of such facilities” [U.S. EPA, 1991 and Carra, 1990].

Hazardous waste has the following characteristics: ignitability (40 CFR § 261.23); corrosivity (40 CFR § 261.21); reactivity (40 CFR § 261.22); toxicity (40 CFR § 261.24) [Friedman, 1979; Percival, et al., 1992]. Hazardous waste is defined by section 1004 (5) of RCRA and 3001 of the Solid Waste Disposal Act (SWDA) as:

a solid waste, or combination of solid waste, which because of its quantity, concentration, or physical, chemical, or infectious characteristic may - (A) cause or significantly contribute to an increase in mortality or an increase in serious irreversible, or incapacitating reversible, illness; or (B) pose a substantial present or potential hazard to human health or the environment when improperly treated, stored, transferred, or disposed of, or otherwise managed [Percival, et al, 1992; Robinson, 1986].
In addition, a hazardous substance, defined in section 101 (14) of the Act, 42 U.S.C. \( \S \) 9601 (14), is:

(A) any substance designated pursuant to section 1321 (b) (2) (A) of title 33, (B) any element, compound, mixture, solution, or substance designated pursuant to section 9602 of this title, (C) any hazardous waste having the characteristics identified under or listed pursuant to section 3001 of the Solid Waste Disposal Act [42 U.S.C. \( \S \) 6921] (but not including any waste the regulation of which under the SWDA [42 U.S.C. \( \S \) 6901 et seq.] has been suspended by Act of Congress), (D) any toxic pollutant listed under section 1317(a) of title 33, (E) any hazardous air pollutant listed under section 112 of the Clean Air Act [42 U.S.C. \( \S \) 7412], and (F) any immediately hazardous chemical substance or mixture with respect to which the Administration has taken action pursuant to section 2606 of title 15 [Percival, et al., 1992].

Furthermore, a regulated hazardous waste means "a solid waste that is a hazardous waste, as defined in 40 CFR 261.3 that isn't excluded from regulations as a hazardous waste under 40 CFR 261.4(b) or was not generated by a condition exempt small quantity generator as defined in \( \S \) 261.5" [U.S. EPA, 1991]. Part 258 of RCRA, which encompasses municipal solid waste landfills (MSWLFs), contains Appendix I which lists constituents for detection monitoring and Appendix II which lists hazardous inorganic and organic constituents [U.S. EPA, 1991]. The concentration of substances in MSWLFs should not exceed the maximum contamination levels contained in Appendix I. Both Appendix I and Appendix II to part 258 of RCRA are supplied in Appendix B of this paper.

As mentioned in the previous section, some materials that contain hazardous components are exempt from hazardous waste controls and regulations. By Part 261 of
RCRA, exclusions include: “household waste, fertilizer used in agricultural operations, and certain categories of high-volume waste that Congress had directed EPA to study (for example, mining wastes like fly ash)” [Robinson, 1986; Percival, et al., 1992]. However, not all fly ash is hazardous. Another exemption is for generators of small amounts (<1,000 kg/month) and very small amounts (<100 kg/month) of hazardous waste.

MSWLFs allow a generation of <100 kg of hazardous waste per month (new in October 1986 [Carra, 1990]), but those that accumulate >100 kg/month are “subject to regulation under subtitle C.” Landfills that generate 100 - 1,000 kg/month are:

not subject to all the regulations, under Parts 262-265 concerning requirements upon those who transport, own, or operate a hazardous waste treatment, storage, or disposal facility. However, if the generator generates > 1,000 kg of hazardous waste per month or accumulates > 1,000 kg of hazardous waste on site, such wastes become subject to all EPA hazardous waste regulations [Robinson, 1986; Percival, et al., 1992].

Disposal of these exempted wastes can have “significant environmental consequences, [due to] millions of gallons of hazardous material [being predominantly] unregulated” [Percival, et al., 1992].

Municipal solid waste, which itself consists of many components, is just one component of solid waste. An MSWLF unit is “a discrete area of land or an excavation that receives household waste and other Subtitle D wastes, and is not a land application unit, surface impoundment, injection well, waste pile, or construction/demolition waste landfill,” which are all categorized under industrial solid waste disposal facilities along
with industrial landfills (all terms are defined in section § 257.2) [U.S. EPA, 1991].

Generally, wastes found at municipal solid waste landfills in the U.S. include: household (72%), commercial (17%), construction/demolition (6%), industrial (2.73%), institutional, sewage sludge, incinerator ash, dead animals, infectious, household hazardous, and very small quantity generator (VSQG) hazardous. Only significant percentages of wastes, which were determined by the EPA in a 1987 MSWLF Survey [Carra, 1990], are noted. The waste stream exclusive of household wastes amounts to 28% of the total and consists of waste types that may contain large quantities of hazardous constituents (82 constituents were found in the leachate—63 organics and 19 inorganics) [Carra, 1990; Percival, et al., 1992; U.S. EPA, 1991]. The following definitions from part 258 of RCRA indicate the difference between the three main waste types found in MSWLF units:

*Household waste* means any solid waste (including garbage, trash, and sanitary waste in septic tanks) derived from households (including single and multiple residences, hotels and motels, bunkhouses, ranger stations, crew quarters, campgrounds, picnic grounds, and day-use recreational areas.)

*Commercial solid waste* means all types of solid waste generated by stores, offices, restaurants, warehouses, and other nonmanufacturing activity, excluding residential and industrial wastes.

*Industrial solid waste* means solid waste generated by manufacturing or industrial processes that is not a hazardous waste regulated under subtitle C of RCRA.


As indicated, household waste is the largest component of municipal waste (72%) [Carra,
1990]; therefore, household waste is targeted, because it will help the municipal solid waste disposal industry the most, and will be safer for the mining companies than accepting all municipal solid waste, because it "eliminates the key potential sources of regulated hazardous waste (such as commercial and industrial waste generators which account for roughly 20% of waste stream)" [U.S. EPA, 1991].

MSWLFs that are non-hazardous are covered in Part 258 of RCRA. An average volume-based composition for a non-hazardous municipal solid waste in the U.S. is: 50% paper, 10% plastic, 6% metal, 1% glass, 13% organic, and 20% miscellaneous (including construction and demolition debris, tires, textiles, rubber, and disposable diapers) [Rathje, 1991].

MSWLF units are required to have detection and prevention systems for the disposal of regulated hazardous waste [U.S. EPA, 1991]. The monitoring parameters chosen were "believed to provide a reliable means of detecting the possible presence of releases from municipal solid waste landfills while avoiding unnecessary analysis costs to the regulating community." The 46 volatile organic compounds (VOCs) were determined to be the best thing to measure (see Appendix B-I) [U.S. EPA, 1991].

3.2. Tonnages and Disposal Methods

Of the entire waste stream, municipal solid waste accounted for almost 1.4% in 1990, whereas industrial non-hazardous waste was the largest component with about
66.7% and mining wastes ranked third after oil and gas wastes with roughly 12.3% [Carra, 1990]. In 1992, the U.S. yielded 291,742,000 tons of municipal solid waste, of which Virginia alone generated 7,600,000 tons and West Virginia 1,700,000 tons [Steuteville and Goldstein, 1993]. As mentioned previously, landfills are suffering serious capacity shortages, especially in highly industrialized areas. This is due in part to the increasing amount of waste and stricter citing regulations for landfills [Carra, 1990; U.S. EPA, 1991]. This capacity issue has been targeted by the EPA as one of three national goals for municipal solid waste management.

The entire United States averaged a 72% landfill disposal rate [Steuteville and Goldstein, 1993]. Virginia used several different methods to dispose of its municipal solid waste: 58% was landfilled (in 254 landfills), 24% recycled, and 18% incinerated. Meanwhile, West Virginia landfilled 90% (in 40 landfills) and recycled only 10% [Steuteville and Goldstein, 1993]. Therefore, since most waste is disposed in landfills, the solution proposed in this work must be cost competitive with landfilling to make any significant impact on the waste problem or to benefit mining companies.

There are many human health and environmental regulations (as defined in 40 CFR 257 and amended by the Hazardous Solid Waste Amendments in 1984) that must be followed by an MSWLF. The requirements stated that facilities must establish the following, but also take into account the "practicable capability" of the facility:

1. Flood plain controls,
2. Protection for endangered species or their critical habitats,
3. Surface water and wetland discharge controls,
4. Groundwater
protection practices, (5) Agricultural land application restrictions, (6) Disease vector controls (includes rodents, flies, and mosquitoes capable of transmitting diseases to humans [Robinson, 1986]), (7) Prohibition on open burning, (8) Controls for explosive gases, fires, bird hazards to aircraft, public access [Carra, 1990].

Two other controls to be established at an MSWLF that were not given in the source above include erosion controls and hazardous waste detection and prevention controls [U.S. EPA, 1991]. Groundwater protection should include “groundwater monitoring as necessary to determine contamination, establish location standards for new or existing facilities and provide for corrective actions as appropriate” [Carra, 1990]. In addition to human health and environmental regulations, there are also location restrictions for landfills. Landfills are not allowed within set distances of the following: airports, 100-year floodplains, wetlands, fault areas, seismic impact zones, and unstable areas [U.S. EPA, 1991].

To prevent environmental contamination, landfills utilize synthetic or natural impermeable layers, since the waste is above the groundwater level. When underground disposal is conducted, the waste is stored below the groundwater level, thus making the impermeable rock the primary isolation, and stabilization the secondary. A study conducted in 1986 looked at this “multibarrier disposal system” for non-hazardous industrial solid wastes [Simonen, 1992]. It was concluded that the waste should be non-flammable and non-explosive, as well as either have a low leachability of harmful elements, or be stabilized to prevent leaching [Simonen, 1992].
3.3. Disposal and Site Development Costs

It is necessary to keep in mind that this research is based on the assumption that people will still pay the typical landfill tip fee to dispose of their trash. Therefore, included here are some typical municipal waste disposal cost ranges for the United States as well as for the individual states of Virginia and West Virginia. The U.S. has an average tip fee of $29/ton with a state high of $74/ton in New Jersey. Virginia has an average tip fee of $25/ton; and, West Virginia, $40/ton [Steuteville and Goldstein, 1993]. Tip fees, as well as haulage fees, which will be discussed later, will vary within states, similarly between states and regime; thus, all costs related to the proposed solution will be very site specific.

In an attempt to find an alternative method of industrial solid waste isolation from the biosphere, underground waste disposal was addressed. A stowage case study revealed that underground mine disposal was 17.4% less expensive than industrial non-hazardous waste landfill disposal [Simonen, 1992]. Stabilized waste disposal underground was estimated to cost between $100/m³ - $125/m³, when the capacity was at least 80,000-100,000 m³ [Simonen, 1992]. These values which equate to about $140/ton are consistent with typical industrial solid waste disposal costs.

A typical site development cost for a landfill holding 6 million short tons of waste at 1,350 lb/yd³ would be roughly $167 million on 560 acres of land [Walsh, 1990]. Therefore, the proposed solution must be cost competitive with this amount. In addition,
the solution should minimize land usage due to land availability problems. Of course, economic feasibility also depends on the area where the disposal will take place, because there are regional variations in all associated costs. The aforementioned landfill was compared to underground solution cavity storage in a previous section (see Figure 2.6).
Chapter 4

Waste Preparation

After household waste is collected from residents, it will be transported to a recycling plant before it travels to the mine to be emplaced underground in an environmentally safe manner. Recycling will reduce the overall emplacement costs; however, the fluctuating recycling markets will limit the savings. By taking recycling away from landfills, public tip fees will increase, because the profit gained by landfills will be lessened. Some of the different types of recycling include: (1) public amenities drop-off, which is a stationary-system for materials including aluminum, glass, and yard-wastes; (2) designated route, which is a mobile-system such as curbside collection; (3) materials recovery facilities (MRFs), where all recyclables that are received at the recovery site are mixed. The MRF category consists of both dirty MRFs, where people pick through materials including cardboard, ferrous, and newspapers, and mechanized MRFs, which are process plants that include a shredder, magnets, and other processing equipment to separate out non-ferrous (i.e. brass, copper, aluminum) and ferrous materials [Schultz, 1995].

A mechanized MRF, which included a flail mill, air classifiers, magnetic separators, cyclones, mineral jigs, flotation circuits, secondary shredders, and high-tension separators (see Figure 4.1), was discussed in a study done in 1976 by the United States Bureau of Mines (USBM) that found recycling to be profitable [Stanczyk and
Figure 4.1 Recycling plant flowsheet. (Stanczyk and DeCesare, 1985)
DeCesare, 1985]; however, it assumed that curbside recycling would not be successful and that the entire waste stream would be recoverable and marketable. The products from the recycling plant used for this study had the following output: 4-8 wt% ferrous metals; less than 1 wt% nonferrous metals; 4-17 wt% glass; and, 63-83 wt% combustibles [Stanczyk and DeCesare, 1985]. Combustibles included paper, plastics, and organics. Perhaps Rathje’s breakdown [Rathje, 1991] of a typical waste stream is more realistic, because it includes unmarketable and non-recyclable material. The USBM study does, nevertheless, provide a starting point for our proposed model. Their plant was able to process 1,000 short tons/day with an operating cost of only $19.24/short ton and a recovery profit of $30.31/short ton, converted into 1994 dollars [Stanczyk and DeCesare, 1985]. It should be noted that this study was conducted at a pilot plant and costs may vary if a larger scale plant is used. Moreover, considering today’s volatile recycling market, this important step in the proposed model can lead to either the success or failure of the project.

Before recycling is implemented, complete knowledge of the recycling market is necessary, because justification for the capital investment is essential, and assurance of the viability of the process is relevant to the success of the proposed method [Robinson, 1986]. The material markets are site specific, like transportation costs and tip fees. Even though some materials have high recycling values, others are not as valuable. The lack of secondary markets is a large concern, which has been recognized by the EPA, as indicated
by one of their three national goals for municipal solid waste management: improving secondary material markets [U.S. EPA, 1991]. Another problem with recycling is that it can be odorous and difficult to process, where paper can be soiled or colored, and plastic can be colored too [Schultz, 1995]. Therefore, even though the USBM makes recycling look extremely attractive, this view may be misleading.
Chapter 5

Emplacement

5.1. Introduction

To understand the idea of backfilling behind longwall shields, first the method of longwall mining must be understood. Therefore, the first section in this chapter discusses and explains that particular mining technique; the second section treats hydraulic backfilling.

5.2. Longwall Mining

A brief overview of longwall mining is found in section 2.5 which gave some examples of the highly automated mining method. One concept which was not mentioned is that as weight on the coal face is lessened by using shields and utilizing other support techniques, or from the main roof caving, the coal becomes easier to cut [Adamczyk, 1995]. Section 2.5 contains Figure 2.8 which showed different zones within the caving zone in cross section; likewise, Figure 5.1 shows in cross section the three overburden zones, including the caved, fractured, and continuous deformation zones. The caved zone in plan view reveals various fall zones in the horizontal direction in addition to the vertical zoning that was already evidenced. As can be seen in Figure 5.2, it takes some time before the first caving occurs. It is not until about the fifth caving sequence that the gob
Figure 5.1 Cross section of three zones in overburden due to longwall mining.  
(Peng and Chiang, 1984)
Figure 5.2 Plan view of fall zones at longwall panel start-up. (after U.S. DOE, 1981)
region resembles an equal rate of caving parallel to the shields and coal face. However, there is still a slight deformation of the caving line, because the material abutting the entries will fall first because less support is provided by the surrounding rock. There should be sufficient time to emplace backfill behind the shields before caving begins. When the backfill is added before degradation starts in the region behind the shields, support will be provided so that failure never has a chance to begin. This means that a failure zone that resembles the first fall outline will be conveyed the entire length of the panel, as long as an appropriate strength backfill is used.

The rate of face advance in the U.S. is between 30 ft and 60 ft (10 m - 20 m)/day, which is 2 to 5 times greater than the advance achieved in Europe [dePretoro and Shostak, 1993]. Some other typical U.S. longwall characteristics include panel widths ranging between 250 ft and 1,000 ft, with the most common 500 ft to 600 ft, and panel lengths from 1,000 ft to 9,000 ft, with the most common between 3,000 ft and 5,000 ft [Peng, 1986]. These distances usually depend on the type of geology that exists at the mine site, as well as the equipment capabilities.

The roof geology is of greatest concern when dealing with caving. Geology sets the criterion for what type of roof support to choose and the speed of shield movement, and can also determine stowing viability. Three immediate roof types include unstable, medium stable, and stable, which has three different categories, too. An unstable roof is like the one found at the case study site addressed in Chapter 7 regarding the economic
model. An unstable immediate roof (see Figure 5.3a) has “weak or soft carbonaceous shale and well-jointed or fractured sandy shale” [Peng and Chiang, 1984]. If supports are not advanced within 10 minutes of the shearer’s passing, then the unsupported material will fall between the faceline and the shields. Likewise, as soon as the supports are advanced, the gob material will fail immediately [Peng and Chiang, 1984]. However, this is only after the mining has progressed past the first few fall sequences; thus, it should not cave immediately in the first fall step, so it should stand up long enough to emplace the waste material. If caving is allowed, then after the caving has equalized, the caving line is near the front half of the shield [Peng and Chiang, 1984]. A medium stable immediate roof (see Figure 5.3b) possesses “hard shale, sandy shale, and weak sandstone” [Peng and Chiang, 1984]. The unsupported roof at the face can remain standing through the return trip of the shearer, and the material behind the shields remains stable longer than that with the unstable roof [Peng and Chiang, 1984]. The caving line is now moved to the rear edge of the shields and sometimes into the gob [Peng and Chiang, 1984].

The final kind of immediate roof, which contains three different types, is stable. Type 1 (see Figure 5.3c) is a “thick and strong sandy shale or sandstone” [Peng and Chiang, 1984]. This type can remain stable for more than 5 to 8 hours without being supported, and the rock in the gob region will drop in large pieces [Peng and Chiang, 1984]. Type 2 (see Figure 5.3d) can also contain sandstone, but a “very thick and hard” sandstone, or it can contain a conglomerate [Peng and Chiang, 1984]. This type of roof
Figure 5.3 Longwall cross section with (a) unstable, (b) medium stable, (c) stable-type 1, (d) stable-type 2, (e) stable-type 3 immediate roof. (Peng and Chiang, 1984)
can remain uncaved for long periods of time; therefore caving must sometimes be induced, so failure does not occur rapidly and forcefully, which can cause damage to the supports [Peng and Chiang, 1984] and is considered a safety hazard. Type 3 (see Figure 5.3e), which remains uncaved for very long periods of time, contains either “hard limestone or sandstone and is thicker than the mining height” [Peng and Chiang, 1984]. Generally “joints and fractures are well developed and become the weak planes along which the rock breaks; thus, causing the rock to break in blocks. Sometimes sags will form a semi-arc, with the gob end eventually resting on the gob” [Peng and Chiang, 1984].

The time in which the gob material remains stable was addressed in the previous paragraph, but no real definite times were given. Table 5.1 displays the Russian and Polish cavability classifications, which are based on rock strength and time exposure. In addition to the three main categories discussed previously, there is also a category for low stable and very stable immediate roofs. $C_a$ is the average uniaxial compressive strength of the core, $w$ is between 4.6 ft and 6.6 ft (1.4-2.0 m), and P stands for the Polish classification number [Peng and Chiang, 1984]. This table gives the Russian time values ranging from 5 minutes to 4 hours, whereas the Polish time values are very general.

5.3. Backfilling

After the waste has passed through the recycling plant and to the mine site, most likely by train, it must be transported underground. The waste, probably combined with
Table 5.1 Cavability classification based on rock strength and time exposure.  
(Peng and Chiang, 1984)

<table>
<thead>
<tr>
<th>Roof Type</th>
<th>Russian Classification</th>
<th>Polish Classification</th>
</tr>
</thead>
<tbody>
<tr>
<td>I, unstable</td>
<td>$C_a \leq 2133$ psi (1471 MPa). After shearer’s cutting, an exposed area $w$ wide can stay unsupported for less than 5 min.</td>
<td>$P = 0$-18. After exposure, roof fall from immediately (low $P$) to a short delay (high $P$). Head coal is needed for protecting the roof.</td>
</tr>
<tr>
<td>II, low stable</td>
<td>$2133$ psi (1471 MPa) $\leq C_a \leq 3555$ psi (2452 MPa). An exposed area $w$ wide by 13 ft (4.0 m) long can stay unsupported for 5-20 min.</td>
<td>$P = 18$-35. Roof very difficult to control. Full of cavities, fractures, and fissures, friable, easily caving.</td>
</tr>
<tr>
<td>III, medium stable</td>
<td>$3555$ psi (2452 MPa) $\leq C_a \leq 4977$ psi (3432 MPa). An exposed area $w$ wide by 13-82 ft (4-25 m) long can stay unsupported from 20 min to 2 hr.</td>
<td>$P = 35$-60. Easily caving. From fractured roof with local falls (low $P$) to fairly good roof (high $P$).</td>
</tr>
<tr>
<td>IV, stable</td>
<td>$4977$ psi (3432 MPa) $\leq C_a \leq 6399$ psi (4413 MPa). An exposed area $w$ wide by 82 ft (25 m) long can stay unsupported for longer than 2 hr.</td>
<td>$P = 60$-130. Good roof with excellent caving (low $P$) to hardly caving (high $P$).</td>
</tr>
<tr>
<td>V, very stable</td>
<td>$C_a &gt; 6399$ psi (4413 MPa). An exposed area $w$ wide by 262-394 ft (80-120 m) long or whole face width can stay unsupported for 3-4 hr.</td>
<td>$P &gt; 130$. Very strong and very stable. Artificial caving is necessary.</td>
</tr>
</tbody>
</table>
additives, is emplaced using a backfilling system like those implemented in Germany, the former Soviet Union, France and Poland [Munjeri, 1987]. In 1975, the National Science Foundation conducted a study on the underground storage of coal refuse using several different backfilling techniques [National Academy of Sciences, 1975]. Only two of the backfilling operations, lateral pneumatic and hydraulic, appear to be suitable for the proposed model based on their automation and application to operating longwall mines. As discussed previously, even though pneumatic is more economic than hydraulic, liquid emplacement is the only safe method due to the possibility of sparks. In addition, two other factors that favor hydraulic stowing over pneumatic are its high emplaced density and mitigation of pipeline erosion. Hydraulic backfill possesses the highest emplaced density, whereas pneumatic is slightly lower than hydraulic and has a greater density than typical cement [Vorobjev and Deshmukh, 1966]. A higher density means better subsidence control and better isolation of wastes from the environment. Pipeline erosion, which is more noticeable in pneumatic, can be alleviated by using smaller particles [Carlson and Saperstein, 1989], or by using fly ash combined with hydraulic stowing, because its fineness will develop a thin film inside the pipeline which will prevent excessive wear [Sinha, 1989]. Replacing pipeline can become an expensive solution.

Lateral pneumatic backfilling, shown previously in cross and plan section in Figure 2.12, operates by having each deflector down the line open when the section of gob being stowed is completed and the corresponding deflector is closed [Singh and Courtney,
1975]. Although a plow and a shearer are shown, a shearer is the more commonly used cutting machine today. Furthermore, U.S. longwall operations typically use retreat mining, rather than advance mining. Pneumatic backfilling closely resembles hydraulic backfilling, the only difference being the medium which is used for emplacement; thus, the same figure may be used to demonstrate hydraulic stowing. However, Figure 5.4 shows a cross section of hydraulic stowing, where it can be seen that a pipeline is suspended, free to swing, from the roof supports and advances with them [Munjari, 1987 and Voss, 1983]. Figure 5.4 also shows the surface site, which includes silos or storage bins for both municipal refuse and coal refuse, a water reservoir, and a mixing tank. Once the waste reaches the bottom of the shaft or borehole, it can continue to the face or be diverted to an underground bunker which can account for surges or shutdowns on the longwall. Either a surge tank must be used, or there must be a way to vary the speed of the waste stream [McWilliams, 1991]. Also seen in Figure 5.4 are telescopic pipes and ball joints which have been developed for hydraulic stowage by being movable under pressure. The ball joints allow up to a 15 degree deflection [Voss, 1983]. Prior discussions also stated that discharge walls prevent the stowed material from interfering with face operations, because the shields were designed as “tight and rugged stowing screens which advanced with the supports” [Voss, 1983]. No noticeable drainage problem should occur as long as enough solids or cementing agents are used in the backfill mixture. Experience in Poland indicates that hydraulic backfill drains within 20 minutes
Figure 5.4 Cross section of hydraulic stowing showing both surface and underground operations.
of its placement [HRB-Singer, 1980], so this time would be a minimum. Some of the additives being addressed in this research can solidify and may be able to support load quicker than the backfill used in Poland. Hydraulic backfill reduces subsidence from 90% of seam height without any fill [McWilliams, 1991] to only 5% - 15% of seam thickness with implementation [Bowman, 1991], which equates to only a few inches in thin seam mining. If the stowed material can support weight more quickly, then the subsidence may even be less than this value.

Hydraulic transport is considerably better than either pneumatic or mechanical. Not only is this system quicker in transporting material, but it is also better for ventilation and easier to install and maintain [Sill and Mez, 1994]. A study done on cable supports stated that by using cable bolts instead of timber cribs, less air resistance was provided [McDonnell, et al., 1995]; therefore, this should remain true in using hydraulic transport instead of mechanical and in using support provided by the backfill instead of alternative methods like cribbing. In addition, it has been found that when stope widths reach 1.6 m, backfilling is more economical than using timbers [Spearing and Steward, 1992]. Another advantage of hydraulic transport is that the material can be transported by using gravity alone. It was noted that slurry fill (density equal to 1.70, which is slightly larger than some other possible additives) can be transported horizontally without pumping for 7,000 m from the force of gravity acting over a 2,000 m drop [Spearing and Steward, 1992]. This is a ratio of 7:2; it can travel horizontally 3.5 times
the distance over which it dropped. Thus, in the case study used in the developed economic model, a drop of 320 ft can transport the material 1,120 ft. Since longwall faces are progressively receding away from shafts, if the pipeline is sent down the shaft, then some intermediate pumping stations might be needed. The main disadvantage of hydraulic stowing is that it requires large amounts of water [Bowman, 1991]; thus, this method can be difficult to implement where water is difficult to acquire.
Chapter 6

Environmental Precautions

It was discussed in a previous section how societal views have changed positively towards the environment since the early 70’s. There is an increased environmental awareness and more people and organizations are taking actions to protect the environment, which is noticed for example by the overwhelming acceptance of recycling. However, municipal solid waste landfills have suffered from “intense public scrutiny,” because of the public’s “demand for a cleaner, safer environment” [Carra, 1990]. Therefore, one of the main concerns with the proposed method is how it will affect the environment. Thus, precautions must be taken to assure that no harm befalls the natural surroundings. The method also must not negatively affect the mine environment. Both of these issues are addressed in the following subsections.

6.1. Groundwater

Waste is allowed to be mixed with the groundwater, on a case by case basis, as stated in Section 1.09 of the Water Quality Standards of the Virginia State Water Control Board [Adam, 1982]. Nevertheless, to guarantee approval of the proposed stowage method, no groundwater contamination should occur. Three options exist to prevent contamination of the groundwater: eliminate the water from the system, do not allow interaction between the water and waste material, and control the water in the system.
Eliminating the water can be done by using pneumatic backfilling; however, this option has been disqualified for safety reasons in coal mines. Moreover, there can be a natural influx of water regardless of the stowage method.

The second method, allowing no interaction, can be achieved by using either artificial or natural stabilizers combined with the waste when it is stowed. This technique also allows for extracting 100% of the coal, which eliminates the problem of acid mine drainage [Sinha, 1989]. The stabilizer must be impervious, quick setting, and inexpensive, and it must have a suitable swell factor and an appropriate strength. A material is being developed in China that possesses all these necessary additive requirements [Henghu, 1993]. It is an inexpensive, quick setting cement that can be used with a water content of 75% to 90% by volume. For further information regarding this innovative material, see Appendix C. Other artificial stabilizers include Portland cement, backfill, and an organophilic clay-based medium. Cement can immobilize inorganic wastes [Glasser, 1993]. A study found that cements higher in sodium, which were alkaline, could react with and coagulate many metals when mixed with acidic wastes [Glasser, 1993]. Table 6.1 gives crystallochemical incorporation of toxic waste materials in crystalline cement phases [Glasser, 1993]. Natural stabilizers include alkaline fly ash, quarry fines, and wood fiber. The advantage of using natural stabilizers is that they are generally considered waste materials and would, therefore, be more economical than man-made stabilizers, as well as better for the environment if disposed in a safe manner. This is true
Table 6.1 Crystallochemical incorporation of toxic waste materials in crystalline cement phases. (Glasser, 1993)

<table>
<thead>
<tr>
<th>Mode of Incorporation</th>
<th>Example</th>
</tr>
</thead>
<tbody>
<tr>
<td>Substitution for calcium</td>
<td>Sr, Ba, Pb</td>
</tr>
<tr>
<td>Substitution for hydroxyl</td>
<td>F-, Cl-, Br-, I-</td>
</tr>
<tr>
<td>Substitution for SO$_4^{2-}$ in AF$_t$ and AF$_m$</td>
<td>IO$_3^-$, etc.</td>
</tr>
<tr>
<td></td>
<td>CrO$_4^{2-}$, SeO$_4^{2-}$, etc.</td>
</tr>
<tr>
<td>Substitution for Al, Fe</td>
<td>M$^{3+}$, Cr$^{3+}$, etc.</td>
</tr>
<tr>
<td>Occupancy of channel sites in AF$_t$</td>
<td>Small organic molecules</td>
</tr>
</tbody>
</table>
only if the producers do not decide that their material is a marketable item. In addition, alkaline fly ash and quarry fines have a neutralizing capability. They have both been used to control acid mine drainage and surface runoff from coal refuse piles [Daniels, et al., 1993 and Kumar and Hudson, 1992].

The final method of preventing groundwater pollution is to control the water either by utilizing favorable site geology or using man-made retainment. The waste is not combined with additives, so does not provide much ground support. However, a stabilizer could be used to eliminate subsidence, but the method would become infeasible. An impermeable layer of rock in the roof will prevent water from entering the mine. An impermeable layer of rock in the floor will facilitate water removal by allowing water to be channelled in the appropriate direction to be pumped from the mine. This water can be treated and then recycled either back into the mine or through the processing plant. In the more realistic case of unfavorable geology, artificial retainers must be used. This option includes the use of barrier walls, such as grout curtains, or liners, similar to those used in landfills. However, artificial retainers are difficult as well as expensive to implement, so they probably will not be a viable option.

6.2. Heat

The second environmental interest deals with the mine environment. It can be expected that if a stabilizer such as cement is used to stow the municipal refuse, there will
be heat generated in the curing process. In addition, heat will be generated from the decomposition of organics in the waste mixture. On the other hand, air will no longer be flowing through the gob because it is backfilled, so the air will not be heated due to its passage through the gob. Consequently, there should not be a significant change in the overall temperature of the air given the assumption that these are equal temperature changes. It was found in a study that air temperatures were 4°-6° lower on pneumatically stowed faces than on similar caving faces, in addition less dust was present in the air [Voss, 1983]. This means that faces using hydraulic stowing would have similar effects on the mine environment and may have fewer airborne particulates.

6.3. Methane and Odors

The final environmental interest is also one concerned with the mine environment. Methane generated from the decomposing organics could be recovered by the methane collection system already in place in most underground coal mines, yet the odors generated may not be as easy to alleviate. Fortunately, there will be an increased air flow at the face because the air no longer travels through the gob. This increase, however, may not be enough to appease the workers at the face. It is not foreseen as a big problem, but it is something that will need to be considered more thoroughly before implementing the proposed stowage scheme. The elimination of stray air currents also decreases the chance of spontaneous combustion [Reinshagen, 1986]. Spontaneous combustion is an
increasing concern for underground coal mines as they operate at greater depths, because at greater depths there are finer particles and the temperature and humidity are greater. As depths increase, the confining pressure in the coal also increases which causes the material to break into smaller particles. A critical point is reached when the water vapor in the air exceeds the water vapor present in the coal, because the phenomenon know as the heat of adsorption increases the temperature of the coal [McPherson, 1993]. Spontaneous combustion incidents have increased dramatically over the past decade due to increased depths and changes in extraction processes, which now incorporate machines that break the coal into smaller particles. The most recent spontaneous combustion accident occurred on December 1, 1995 in Alabama at one of the mines owned by Jim Walters Resources.
Chapter 7

Economic Model

7.1. Introduction

Economic analysis of the proposed stowage method included creating a flowsheet, formulating an equation, and formatting a spreadsheet that allowed a mine operator to make changes for their specific mine and waste stream. A model transforms a complex system so that a normal person can understand it. Models take the form of graphs, mathematical equations which use average characteristics, and computer programs which take into account real values [Samuelson and Nordhaus, 1989]. There are many types of models such as general equilibrium, production, market, input-output, and mathematical models [Palm and Smit, 1991]. The model used in this research is actually a combination of a flowchart, a mathematical equation, and a computer spreadsheet. The flowchart can also be manipulated into an economic model by itself.

The flowchart, which is considered the “Work Breakdown Structure (WBS)” technique, “serves as a framework for defining all project work elements and their interrelationships, collecting and organizing information, developing relevant cost and revenue data, and integrating project management activities” [Degarmo, et al., 1993]. The mathematical equation follows the “factor” technique which includes summing the product of several quantities or components and adding these to any components.
estimated directly. That is,

\[ C = \sum_d C_d + \sum_m f_m U_m \]  \hspace{1cm} (7.1)

where

- \( C \) = cost being estimated
- \( \sum_d \) = the sum of all components \( d \)
- \( C_d \) = cost of the selected component \( d \) that is estimated directly
- \( \sum_m \) = the sum of all components \( m \)
- \( f_m \) = cost per unit of component \( m \)
- \( U_m \) = number of units of component \( m \)

[Degarmo, et al., 1993]

For example, suppose a miner earned $30,000 annually; in addition, the miner earned $5 for every foot of advance in excess of 10 feet per day and $1 for every bolt installed. If the miner averaged an extra 2 feet per day and installed 12 bolts per day for 300 days, then this would be a total of 600 feet and 3600 bolts. Therefore, the total earned by the miner would be the sum of the base salary plus the sum of all the component benefits:

\[ 30,000 + [($5/ft)(600 \text{ ft})+($1/bolt)(3600 \text{ bolts})] = 36,600. \]

The computer spreadsheet is a tool, rather than a model, that helps companies easily use the mathematical model in their cost and price estimating.

7.2. Economic Model and Equation

A partial model is shown in Figure 7.1, where the total cost includes public waste disposal profits, transportation costs, recycling profits, and stowage costs. Aesthetic
Total Cost = - Waste Disposal Income + Transport + Recycling + Stowage

TC = -WD + T + R + S

T = [(TR(dist)r)(w tons) + (TM(dist)m)(r tons)]

Figure 7.1 Partial economic model showing transportation breakdown.
values and complete environmental costs and benefits have not been included in the
calculations due to lack of information. Transportation has been broken down to show
that this cost includes transportation to the recycler and the mine. Each box can be
broken down, and some even have further expansions. This initial step made it easier to
formulate the equation shown below:

\[
TC = -[(LF + TL(dist)_{LF}(w\ tons)) + [TR(dist)_{R}(w\ tons) + (TM(dist)_{M}(r\ tons))]]
+ [(RE -RP)(w\ tons)] + [(\Sigma_{CA}(a\ tons) + (AT)(dist)_{A}(a\ tons)) + (ES)(c\ tons)]
+ (PT)(c\ tons) + (I)(1.0896)(10^{-4})(c\ tons) + (IT)(dist)_{I}(i\ tons) + (M + L)(r\ tons))
+ (TS)(c\ tons) + (F)(c\ tons)]
\]  

(7.2)

7.3. Definition of Variables

The following list defines the terms in the aforementioned equation and Figure 7.1.

TC = total cost
T = transportation component
LF = landfill tip fee
TL = transport to landfill
(dist)_{LF} = distance to landfill
(w tons) = waste tonnage
TR = transport to recycling plant
(dist)_{R} = distance to recycling plant
TM = transport to mine
(dist)_{M} = distance to mine
(r tons) = tonnage after recycling
RE = recycling expense
RP = recycling profit
\Sigma_{CA} = sum of component additives
A = additive cost
(a tons) = additive tonnage
AT = additive transport
(dist)\_A = distance of additive transport
ES = emplacement system
(c tons) = coal tonnage
PT = pumping costs
I = impermeable layer cost
IT = impermeable layer transport
(dist)\_l = distance of layer transport
(i tons) = layer tonnage
M = environmental monitoring
L = collect and treat leachate
TS = tailgate support elimination
F = face load reduction

7.4. Average Mine Data Spreadsheet

The spreadsheet allows for easy data manipulation, because equations are embedded within the cells. This allows for determining the effects of different stowage techniques and effects of waste tonnages and other pertinent values that can change depending on the mine site. The spreadsheet utilizing the average mine and waste disposal values is shown in Table 7.1. The source and calculations of the values are found in Appendix D. The disposal technique of using a stabilizer, in particular the quick setting - high water cement being developed in China, to eliminate interaction between the waste and the groundwater is used as a baseline, due to its viability. The italics indicate the values that are automatically calculated and the plain text, values that must be input by the mine operator. Depending on the stowage method chosen, some entries can be left blank or given a value of zero. Furthermore, negative values indicate a number subtracted from the
Table 7.1 Spreadsheet utilizing average mine and waste disposal data.

<table>
<thead>
<tr>
<th>Economic Model for Waste Disposal Underground</th>
</tr>
</thead>
<tbody>
<tr>
<td>Landfill Tip Fee ($/ton refuse)</td>
</tr>
<tr>
<td>Transport to Landfill ($/ton*mile)</td>
</tr>
<tr>
<td>Distance to Landfill (miles)</td>
</tr>
<tr>
<td>Waste Tonnage (tons)</td>
</tr>
<tr>
<td>Public Income for Waste Disposal ($)</td>
</tr>
<tr>
<td>Transport to Recycler ($/ton*mile)</td>
</tr>
<tr>
<td>Distance to Recycler (miles)</td>
</tr>
<tr>
<td>Percent Recycled (%)</td>
</tr>
<tr>
<td>Transport to Mine ($/ton*mile)</td>
</tr>
<tr>
<td>Distance to Mine (miles)</td>
</tr>
<tr>
<td>Transportation Costs ($)</td>
</tr>
<tr>
<td>Recycling Expense ($/ton waste)</td>
</tr>
<tr>
<td>Recycling Profit ($/ton waste)</td>
</tr>
<tr>
<td>Recycling Factor ($)</td>
</tr>
<tr>
<td>Name of Additive</td>
</tr>
<tr>
<td>Additive Density (g/cm³)</td>
</tr>
<tr>
<td>Additive Cost ($/ton)</td>
</tr>
<tr>
<td>Additive Cost ($/m³ )</td>
</tr>
<tr>
<td>Additive Transport ($/ton*mile)</td>
</tr>
<tr>
<td>Additive Tonnage (tons)</td>
</tr>
<tr>
<td>Additive Distance (miles)</td>
</tr>
<tr>
<td>Percent of Additive Used (%)</td>
</tr>
<tr>
<td>Cumulative Additive Costs ($)</td>
</tr>
<tr>
<td>Percent of Mine Filled (MFC) (%)</td>
</tr>
<tr>
<td>Depth of Borehole (ft)</td>
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<tr>
<td>Distance from Borehole to Longwall Panel (ft)</td>
</tr>
<tr>
<td>Emplacement Costs ($/ton coal)</td>
</tr>
<tr>
<td>Coal Tonnage (tons)</td>
</tr>
<tr>
<td>Coal Seam Height (m)</td>
</tr>
<tr>
<td>Coal Density (g/cm³)</td>
</tr>
<tr>
<td>Coal Volume (m³)</td>
</tr>
<tr>
<td>Fill Volume (m³)</td>
</tr>
<tr>
<td>Acreage (acres)</td>
</tr>
<tr>
<td>Fill Acreage (acres)</td>
</tr>
<tr>
<td>Impermeable Layer Cost ($/acre) incl. transportation</td>
</tr>
<tr>
<td>Pumping Costs ($/ton clean coal)</td>
</tr>
<tr>
<td>Environmental Monitoring Costs ($/ton refuse)</td>
</tr>
<tr>
<td>Leachate Collection &amp; Treatment ($/ton refuse)</td>
</tr>
<tr>
<td>Safe Stockage Costs ($)</td>
</tr>
<tr>
<td>Length of Longwall (ft)</td>
</tr>
<tr>
<td>Number of moves (moves/yr)</td>
</tr>
<tr>
<td>Support Method Cost ($/ft of advance)</td>
</tr>
<tr>
<td>Tailgate Support Elimination ($) [not incl. transport]</td>
</tr>
<tr>
<td>Face Load Reduction ($)</td>
</tr>
<tr>
<td>Face Load Reduction with MFC ($)</td>
</tr>
<tr>
<td>Subsidence Reduction ($)</td>
</tr>
<tr>
<td>Subsidence Reduction with MFC ($)</td>
</tr>
<tr>
<td>Cumulative Benefits ($)</td>
</tr>
<tr>
<td>Overall Cost/Profit</td>
</tr>
</tbody>
</table>
total cost or that the method is feasible. This sign notation is opposite to what is seen in
the sensitivity analysis graphs. These graphs indicate that as the value becomes more
positive, it also becomes more feasible.

The first sensitivity analysis is conducted to determine which disposal methods are
the most feasible alternatives. Figure 7.2 displays this graphically. It can be seen that
both select backfill and artificial impermeable layers are infeasible techniques. The
Chinese cement and other fills have roughly the same feasibility; however, fly ash and
impermeable floors have a greater feasibility. Therefore, these techniques will be used
before the others, but not before a site with an impermeable roof. Ideally, a site with
favorable geology is preferred because major additive costs are eliminated; however, this
is uncommon, so the next step is to use a natural stabilizer which also decreases costs.

In the other sensitivity analysis, Chinese cement is held as the constant stowage
technique and the other variables are changed one at a time to show their effect on
feasibility. The results indicate linear relationships between the variables and feasibility,
except for the coal seam height which indicates a parabolic relationship, as displayed in
Figure 7.3. It can be seen in the figure that as thickness increases, so does the feasibility.
A larger increase in feasibility is seen in Figure 7.4, which indicates the linear effects of
the recycling factor. A slight decrease in the recycling value used with the average mine
data would make the method infeasible. Transportation costs, additive costs, and solid
waste tonnages have the greatest effect on feasibility. An increase of 50 miles to the mine
Figure 7.2 Feasibility of different stowage techniques.
Figure 7.3  Effect of coal seam height on feasibility.
Figure 7.4 Effect of recycling on feasibility.
decreases feasibility by about 5 million dollars as shown in Figure 7.5. An increase in waste transportation distances to the recycler decreases feasibility more noticeably, because such a large amount of waste is being handled. An additive cost increase of $10/m³ increases the total cost by roughly 8 million dollars as shown in Figure 7.6. Although the exact amount of additives to achieve impermeability and other previously mentioned properties are unknown, a sensitivity analysis shows the effects of different additive percentages. Logically, feasibility increases when fewer additives are used, which is displayed graphically in Figure 7.7. If the additives do not cost anything, then this property does not hold. Another option, other than decreasing the amount of additives, is decreasing the percentage of the mine being filled. This results in a smaller overall change in the feasibility than seen by changing the additive percentage. The assumption made is that as fill volume decreases the same percentage decrease will be seen for each benefit. However, fewer additives are used because the void to be filled is smaller by the given percentage which results in an increase in the feasibility. Thus, as shown in Figure 7.8, there is only a small decrease in overall feasibility.

7.5. Case Study

The purpose of this case study is to assign real values to the model, instead of average values. Values from a mine in the southern Appalachian region and its closest large town are used in the case study. Mine X is an underground coal mine using a
Figure 7.5 Effect of distance to mine on feasibility.
Figure 7.6 Effect of additive cost on feasibility.
Figure 7.7 Effect of percent of additive used on feasibility.
Figure 7.8 Effect of percent of mine filled on feasibility.
longwall material extraction system.

7.5.1. Case Study Spreadsheet

Table 7.2 is the spreadsheet developed with values from the case study mine site. The spreadsheet characteristics described previously are valid for the case study spreadsheet. It should be noted that landfill tip fees are more expensive, but waste tonnage is lower. In addition, transportation costs have decreased because the haulage distances are lower than those used in the case of the average mine.

7.5.2. Summary of Case Study Results

As can be seen from Table 7.2, the proposed waste disposal method is very uneconomical for this particular mine and town combination. Even with recycling the project is unprofitable. Large amounts of additives are required, because there is not enough refuse to fill the entire void created from coal mining. Therefore, excessive amounts of additives are required to achieve the benefit of subsidence control, with the assumption of complete filling of the voids. An option to alleviate the amount of additives required is to combine coal refuse, which creates many environmental problems when stored on the surface, with the household waste. If this is done, then the proposed stowage method will be viable. Another problem at this mine site is the rate of closure. The roof caves as soon as the shields are moved. This can be overcome by extending the
Table 7.2 Spreadsheet using case study specifics.

<table>
<thead>
<tr>
<th>Economic Model for Waste Disposal Underground - Case Study</th>
</tr>
</thead>
<tbody>
<tr>
<td>Landfill Tip Fee ($/ton refuse)</td>
</tr>
<tr>
<td>Transport to Landfill ($/ton*mile)</td>
</tr>
<tr>
<td>Distance to Landfill (miles)</td>
</tr>
<tr>
<td>Waste Tonnage (tons)</td>
</tr>
<tr>
<td>Public Income for Waste Disposal ($)</td>
</tr>
<tr>
<td>Transport to Recycler ($/ton*mile)</td>
</tr>
<tr>
<td>Distance to Recycler (miles)</td>
</tr>
<tr>
<td>Percent Recycled (%)</td>
</tr>
<tr>
<td>Transport to Mine ($/ton*mile)</td>
</tr>
<tr>
<td>Distance to Mine (miles)</td>
</tr>
<tr>
<td>Transportation Costs ($)</td>
</tr>
<tr>
<td>Recycling Expense ($/ton waste)</td>
</tr>
<tr>
<td>Recycling Profit ($/ton waste)</td>
</tr>
<tr>
<td>Recycling Factor ($)</td>
</tr>
<tr>
<td>Name of Additive</td>
</tr>
<tr>
<td>Additive Density (g/cm(^3))</td>
</tr>
<tr>
<td>Additive Cost ($/ton)</td>
</tr>
<tr>
<td>Additive Cost ($/m(^3))</td>
</tr>
<tr>
<td>Additive Transport ($/ton*mile)</td>
</tr>
<tr>
<td>Additive Tonnage (tons)</td>
</tr>
<tr>
<td>Additive Distance (miles)</td>
</tr>
<tr>
<td>Percent of Additive Used (%)</td>
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<td>Cumulative Additive Costs ($)</td>
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<td>Emplacement Costs ($/ton coal)</td>
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<td>Coal Tonnage (tons)</td>
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</tr>
<tr>
<td>Coal Density (g/cm(^3))</td>
</tr>
<tr>
<td>Coal Volume (m(^3))</td>
</tr>
<tr>
<td>Fill Volume (m(^3))</td>
</tr>
<tr>
<td>Acreage (acres)</td>
</tr>
<tr>
<td>Fill Acreage (acres)</td>
</tr>
<tr>
<td>Impermeable Layer Cost ($/acre) [incl. transportation]</td>
</tr>
<tr>
<td>Pumping Costs ($/ton clean coal)</td>
</tr>
<tr>
<td>Environmental Monitoring Costs ($/ton refuse)</td>
</tr>
<tr>
<td>Leachate Collection &amp; Treatment ($/ton refuse)</td>
</tr>
<tr>
<td>Safe Storage Costs ($)</td>
</tr>
<tr>
<td>Length of Longwall (ft)</td>
</tr>
<tr>
<td>Number of moves (moves/yr)</td>
</tr>
<tr>
<td>Support Method Cost ($/ft of advance)</td>
</tr>
<tr>
<td>Twillage Support Elimination ($) [not incl. transport]</td>
</tr>
<tr>
<td>Face Load Reduction ($)</td>
</tr>
<tr>
<td>Face Load Reduction with MFC ($)</td>
</tr>
<tr>
<td>Subsidence Reduction ($)</td>
</tr>
<tr>
<td>Subsidence Reduction with MFC ($)</td>
</tr>
<tr>
<td>Cumulative Benefits ($)</td>
</tr>
<tr>
<td>Overall Cost/Profit</td>
</tr>
</tbody>
</table>

79
back canopies of the shields out further. Moreover, backfill provides ground support, so this will help keep the roof up longer to allow hydraulic emplacement.

7.6. Summary of Economic Model Results

Table 7.3 is a summary of the average mine and case study data sets, where negative values indicate feasibility. When waste tonnage is large, the largest cost comes from transportation. But in the instance of the case study where the waste stream is small, the largest cost is from additives. Viability of the stowage method is increased by using coal refuse as an additional additive at any underground coal mine. This solves the material deficit problem, which creates excessive additive use and eliminates the costs associated with surface stowage of the refuse. Without using coal refuse, waste stowage at the case study site is highly infeasible, unlike the average mine data which proves feasible if recycling is successful.

However, if the time value of money is taken into consideration, this disposal technique will prove to be infeasible. The internal rate of return is small, so the method is not very economical. Mining companies will not likely take the risk posed by such large amounts of money changing hands for only a very small percent profit. There is a chance that the smallest factor of error could make what seems profitable, a severe burden for the mining companies. So even though the model shows the system is feasible, an in-depth economic analysis would probably show otherwise.
Table 7.3  Spreadsheet summary of both average and case study data.

<table>
<thead>
<tr>
<th></th>
<th>Average Mine Data</th>
<th>Case Study Data</th>
</tr>
</thead>
<tbody>
<tr>
<td>Waste Tonnage</td>
<td>4,408,000</td>
<td>29,100</td>
</tr>
<tr>
<td>Disposal Income</td>
<td>-480,472,000</td>
<td>-1,200,375</td>
</tr>
<tr>
<td>Transportation</td>
<td>508,573,000</td>
<td>270,630</td>
</tr>
<tr>
<td>Additives</td>
<td>18,099,280</td>
<td>63,366,310</td>
</tr>
<tr>
<td>Safe Stowage</td>
<td>3,271,572</td>
<td>7,029,881</td>
</tr>
<tr>
<td>Overall Cost/Profit</td>
<td>-1,620,987</td>
<td>66,544,309</td>
</tr>
</tbody>
</table>
Chapter 8

Conclusions And Recommendations

8.1. Conclusions

Mining companies could gain considerable economic benefit if the municipal solid waste stowage method can be implemented. Reducing face loading and eliminating secondary tailgate support by providing support with the backfill-like material will decrease mining expenses. Furthermore, dollars spent on subsidence control and damages will decline, while controlling subsidence will help the environment. Waste storage underground will provide more land for public and private use and improve aesthetics, which can also be considered environmentally beneficial. This stowage method provides an increase in "effective integrated waste management planning by.....industry," which was the purpose of three national goals for municipal solid waste management which were promulgated by EPA [U.S. EPA, 1991]. The waste disposal model, as presented, has site specific economics. If the mine is located near an industrialized area, the model will be feasible, but if the mine is far from an industrialized city, the economics do not look as promising. If large waste tonnages are involved, as with the first analysis using average mine and waste disposal data, then transportation costs drive the feasibility, whereas if waste tonnages are small, as with the case study, the additive costs determine the feasibility. This is with the assumption that the mine is completely filled to achieve
subsidence benefits. Site specifics, such as geology, are factors for success as well. However, if the time value of money is addressed, then the method will be infeasible, unless the value of the environment is taken into consideration, too. Mining companies will not likely take the risk of this proposed disposal method, unless the economics improve and the profit margin is larger than a few percent.

The disposal model can be applied to other underground mining industries in addition to coal. It may even be easier or better to use other mineral types to dispose waste, such as the salt caverns previously discussed or the impermeable hard rock mines. Although the coal mining industry is the one that needs the financial help to survive, other types of operations could reap the benefits as well.

8.2. Recommendations for Future Research

This research disregarded the fact that environmental regulations exist that may not allow waste underground. Before any changes to current laws were made, the feasibility of the method had to be addressed. If the method is feasible, then the laws must be looked at closer. However, legal issues are beyond the scope of this thesis.

Further investigation is necessary into the recycling step and the quick setting - high water cement. Additionally, the economics associated with aesthetics and environmental benefits should be determined. Finally, a pilot installation should be tried before full-scale implementation takes place.
In summary, a business analysis should be conducted, because only a technical analysis was completed in this research. Economic indicators including the internal rate of return, depreciation, and salvage values should be used to determine the economic feasibility of this disposal method.
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APPENDIX A

Conversion of Coal to Electricity
This appendix shows the conversion between coal and electricity. The average energy output from coal in the Appalachian region is 12,500 Btu/lb. (Luttrell, 1995)

Furthermore, the average coal production from the Southern Appalachian region is about 30 million tons per year from 20 longwall installations. (Weisdack & Wolf, 1995) Using these values, the electric output from the aforementioned region is calculated below.

\[
\text{Electric Output} = (\text{annual coal tonnage})(\text{energy conversion factor})
\]

\[
\text{Electric Output} = (30,000,000 \text{ tons}) \left( \frac{12,500 \text{ Btu}}{1 \text{ lb of coal}} \right) \left( \frac{2,000 \text{ lbs}}{1 \text{ ton}} \right) \left( \frac{2.928 \times 10^{-4} \text{ kW - hr}}{1 \text{ Btu}} \right)
\]

\[
\text{Electric Output} = 219.6 \times 10^9 \text{ kW - hr}
\]

An electric company in the Southern Appalachian region gave 10,400 kW-hr as an average energy consumption value per household per year. (Dellinger, 1995) This means that over 21.1 million homes could be powered annually by the coal produced in the Southern Appalachian region. The supporting calculation is shown below.

\[
\text{Number of Homes} = \frac{\text{electric output from coal}}{\text{average energy consumption per household}}
\]

\[
\text{Number of Homes} = (219.6 \times 10^9 \text{ kW - hr}) \left( \frac{1 \text{ home}}{10,400 \text{ kW - hr}} \right)
\]

\[
\text{Number of Homes} = 21,115,385
\]
APPENDIX B

Appendices I and II to Part 258 of the Resource Conservation and Recovery Act (RCRA)
Appendix I — Constituents for Detection Monitoring

Inorganic Constituents:

- Antimony
- Arsenic
- Barium
- Beryllium
- Cadmium
- Chromium
- Cobalt
- Copper
- Lead
- Nickel
- Selenium
- Silver
- Thallium
- Vanadium
- Zinc

Organic Constituents:

- Acetone
- Acrylonitrile
- Benzene
- Bromochloromethane
- Bromodichloromethane
- Bromoform; Tribromomethane
- Carbon disulfide
- Carbon tetrachloride
- Chlorobenzene
- Chloroethane; Ethyl chloride
- Chloroform; Trichloromethane
- Dibromochloromethane; Chlorodibromomethane
- 1,2-Dibromo-3-chloropropane; DBCP
- 1,2-Dibromoethane; Ethylene dibromide; EDB
o-Dichlorobenzene; 1,2-Dichlorobenzene
p-Dichlorobenzene; 1,4-Dichlorobenzene
trans-1,4-Dichloro-2-butene
1,1-Dichloroethane; Ethylidene chloride
1,2-Dichloroethane; Ethylene dichloride
1,1-Dichloroethylene; 1,1-Dichloroethene; Vinylidene chloride
cis-1,2-Dichloroethylene; cis-1,2-Dichloroethene
trans-1,2-Dichloroethylene; trans-1,2-Dichloroethene
1,2-Dichloropropane; Propylene dichloride
cis-1,3-Dichloropropene
trans-1,3-Dichloropropene
Ethylbenzene
2-Hexanone; Methyl butyl ketone
Methyl bromide; Bromomethane
Methyl chloride; Chloromethane
Methylene bromide; Dibromomethane
Methylene chloride; Dichloromethane
Methyl ethyl ketone; MEK; 2-Butanone
Methyl iodide; Iodomethane
4-Methyl-2-pentanone; Methyl isobutyl ketone
Styrene
1,1,1,2-Tetrachloroethane
1,1,2,2-Tetrachloroethane
Tetrachloroethylene; Tetrachloroethene; Perchloroethylene
Toluene
1,1,1-Trichloroethane; Methylchloroform
1,1,2-Trichloroethane
Trichloroethylene; Trichloroethene
Trichlorofluoromethane; CFC-11
1,2,3-Trichloropropane
Vinyl acetate
Vinyl chloride
Xylenes
Appendix II -- List of Hazardous Inorganic and Organic Constituents

Acenaphthene
Acenaphthylene
Acetone
Acetonitrile; Methyl cyanide
Acetophenone
2-Acetylaminoflourene; 2-AAF
Acrolein
Acrylonitrile
Aldrin
Aliyl chloride
4-Aminobiphenyl
Anthracene
Antimony
Arsenic
Barium
Benzene
Benzo[a]anthracene; Benzantracene
Benzo[b]fluoranthene
Benzo[k]fluoranthene
Benzo[ghi]perylene
Benzo[a]pyrene
Benzy1 alcohol
Beryllium
alpha-BHC
beta-BHC
delta-BHC
gamma-BHC; Lindane
Bis(2-chloroethoxy)methane
Bis(2-chloroethyl) ether; Dichloroethyl ether
Bis(2-chloro-1-methylethyl) ether; 2,2-Dichlorodiisopropyl ether; DCIP
Bis(2-ethylhexyl) phthalate
Bromochloromethane; Chlorobromomethane
Bromodichloromethane; Dibromochloromethane
Bromoform; Tribromomethane
4-Bromophenyl phenyl ether
Butyl benzyl phthalate; Benzyl butyl phthalate
Cadmium
Carbon disulfide
Carbon tetrachloride
Chlordane
p-Chloroaniline
Chlorobenzene
Chlorobenzilate
p-Chloro-m-cresol; 4-Chloro-3-methylphenol
Chloroethane; Ethyl chloride
2-Chloroethyl ethyl ether
Chloroform; Trichloromethane
2-Chloronaphthalene
2-Chlorophenol
4-Chlorophenyl phenyl ether
Chloroprene
Chromium
Chrysene
Cobalt
Copper
m-Cresol; 3-Methylphenol
o-Cresol; 2-Methylphenol
p-Cresol; 4-Methylphenol
Cyanide
2,4-D; 2,4-Dichlorophenoxyacetic acid
4,4-DDD
4,4-DDE
4,4-DDT
Diallate
Dibenz[a,h]anthracene
Dibromochloromethane; Chlorodibromomethane
1,2-Dibromo-3-chloropropane; DBCP
1,2-Dibromoethane; Ethylene dibromide; EDB
Di-n-butyl phthalate
o-Dichlorobenzene; 1,2-Dichlorobenzene
m-Dichlorobenzene; 1,3-Dichlorobenzene
p-Dichlorobenzene; 1,4-Dichlorobenzene
3,3-Dichlorobenzidine
trans-1,4-Dichloro-2-butene
Dichlorodifluoromethane; CFC 12
1,1-Dichloroethane; Ethyldene chloride
1,2-Dichloroethane; Ethylene dichloride
1,1-Dichloroethylene; 1,1-Dichloroethene; Vinylidene chloride
cis-1,2-Dichloroethylene; cis-1,2-Dichloroethene
trans-1,2-Dichloroethylene; trans-1,2-Dichloroethene
2,4-Dichlorophenol
2,6-Dichlorophenol
1,2-Dichloropropane; Propylene dichloride
1,3-Dichloropropane; Trimethylene dichloride
2,2-Dichloropropane; Isopropylidene chloride
1,1-Dichloropropene
cis-1,3-Dichloropropene
trans-1,3-Dichloropropene
Dieldrin
Diethyl phthalate
0,0-Diethyl 0-2-pyrazinyl phosphorothionate; Thionazin
Dimethoate
p-(Dimethylamino)azobenzene
7,12-Dimethylbenz[a]anthracene
3,3-Dimethylbenzidine
2,4-Dimethylphenol; m-Xylenol
Dimethyl phthalate
m-Dinitrobenzene
4,6-Dinitro-o-cresol; 4,6-Dinitro-2-methylphenol
2,4-Dinitrophenol
2,4-Dinitrotoluene
2,6-Dinitrotoluene
Disoseb; DNBP; 2-sec-Butyl-4,6-dinitrophenol
Di-n-octyl phthalate
Diphenylamine
Disulfoton
Endosulfan I
Endosulfan II
Endosulfan sulfate
Endrin
Endrin aldehyde
Ethylbenzene
Ethyl methacrylate
Ethyl methanesulfonate
Famphur
Flouroanthene
Flourene
Heptachlor
Heptachlor epoxide
Hexachlorobenzene
Hexachlorobutadiene
Hexachlorocyclopentadiene
Hexachloroethane
Hexachloropropene
2-Hexanone; Methyl butyl ketone
Indenol(1,2,3-cd)pyrene
Isobutyl alcohol
Isodrin
Isophorone
Isosafrole
Kepone
Lead
Mercury
Methacrylonitrile
Methapyrilene
Methoxychlor
Methyl bromide; Bromomethane
Methyl chloride; Chloromethane
3-Methylcholanthrene
Methyl ethyl ketone; MEK; 2-Butanone
Methyl iodide; Iodomethane
Methyl methacrylate
Methyl methanesulfonate
2-Methylnaphthalene
Methyl parathion; Parathion methyl
4-Methyl-2-pentanone; Methyl isobutyl ketone
Methylene bromide; Dibromomethane
Methylene chloride; Dichloromethane
Naphthalene
1,4-Naphthoquinone
1-Naphthylamine
2-Naphthylamine
Nickel
o-Nitroaniline; 2-Nitroaniline
m-Nitroaniline; 3-Nitroanile
p-Nitroaniline; 4-Nitroaniline
Nitrobenzene
0-Nitrophenol; 2-Nitrophenol
p-Nitrophenol; 4-Nitrophenol
N-Nitrosodi-n-butylamine
N-Nitrosodiethylamine
N-Nitrosodimethylamine
N-Nitrosodiphenylamine
N-Nitrosodipropylamine; N-Nitroso-N-dipropylamine; Di-n-propyl nitrosamine
N-Nitrosomethyl methyamine
N-Nitrosopiperidine
N-Nitrosopyrrolidine
5-Nitro-o-toluidine
Parathion
Pentachlorobenzene
Pentachloronitrobenzene
Pantechlorophenol
Phenacetin
Phenan-threne
Phenol
p-Phenylenediamine
Phorate
Polychlorinated biphenyls; PCBs; Aroclors
Pronamide
Propionitrile; Ethyl cyanide
Pyrene
Safrole
Selenium
Silver
Silvex; 2,4,5-TP
Styrene
Sulfide
2,4,5-T; 2,4,5-Trichlorophenoxyacetic acid
1,2,4,5-Tetrachlorobenzene
1,1,1,2-Tetrachloroethane
1,1,2,2-Tetrachloroethane
Tetrachloroethylene; Tetrachloroethene; Perchloroethylene
2,3,4,6-Tetrachlorophenol
Thalium
Tin
Toluene
o-Toluidine
Toxaphene
1,2,4-Trichlorobenzene
1,1,1-Trichloroethane; Methylchloroform
1,1,2-Trichloroethane
Trichloroethylene; Trichloroethene
Trichlorofluoromethane; CFC-11
2,4,5-Trichlorophenol
2,4,6-Trichlorophenol
1,2,3-Trichloropropane
0,0,0-Triethyl phosphorothioate
sym-Trinitrobenzene
Vanadium
Vinyl acetate
Vinyl chloride; Chloroethene
Xylene
Zinc
APPENDIX C

Quick Setting - High Water Cement Characteristics
The non-poisonous cement being developed in China consists of two components, ‘A’ and ‘B’, which are each individually mixed with water and then combined in a 1:1 ratio. ‘A’ is primarily calcite and is prepared by pulverizing twice and burning once. It has a pH between 9 and 10, which is a weak base. ‘B’ has not yet been patented and is pulverized only once. It is more basic in character with a pH ranging from 11 to 12 [Henghu, 1993]. The raw material used to produce this cement is plentiful in resources and low in cost. If tailings are used instead of water, fewer % solids by volume are needed per component. The compounds should be left unmixed until they reach the point of emplacement, because the material begins solidifying within minutes. The material will not harden for 24 hours if left unmixed [Henghu, 1993]. An hour after the two component streams have been combined, the weight of a person can be supported by the mix [Jones, 1995], which has been measured between 0.5 and 1.0 MPa [Henghu, 1993].

The following information provides the characteristics, general composition, setting time, strength, and applications of this quick setting - high water cement [Henghu, 1993].

**CHARACTERISTICS:**

- High water content - <75% to 90% by volume
- Water resistant once hardened
- Low corrosion factor
- Fast setting - attain solid form within 15-30 minutes
- Application in ocean environment with resistance to NaCl and H₂SO₄
- Winter application to ± 0° C
GENERAL COMPOSITION:

\((3\text{CaO})(\text{Al}_2\text{O}_3)(3\text{CaSO}_4)(32\text{HO}_2)\)
Up to 90% H_2O by volume
Source material: Limestone, Bauxite, and Gypsum

SETTING TIME AND STRENGTH:

<table>
<thead>
<tr>
<th>Time</th>
<th>Percentage of Ultimate Strength</th>
</tr>
</thead>
<tbody>
<tr>
<td>2 Hours</td>
<td>30% of ultimate strength (1.5 MPa)</td>
</tr>
<tr>
<td>6 Hours</td>
<td>60% of ultimate strength</td>
</tr>
<tr>
<td>24 Hours</td>
<td>80% of ultimate strength (3.5 MPa)</td>
</tr>
<tr>
<td>7 Days</td>
<td>±100% of ultimate strength (5.5 MPa)</td>
</tr>
</tbody>
</table>

APPLICATIONS:

High Water Cement - >85% H_2O

Coal Mines

1) Pillars
2) Roof/Rib Support
3) Grouting
4) Fire Suppression
5) Bulkheads and Stoppings

Metal mines

1) Similar applications to coal mines
2) Tailing pond stabilization
3) Hydraulic backfill applications (5% cement + 95% tailings material)
Low Water Cement - <75% H₂O

1) As conventional cement
2) Insulation
3) Shotcrete
4) Soil consolidation
5) Slope stability
6) Retaining wall and structures

In addition, this product does not produce humidity and due to its flowing performance will automatically form a level with satisfactory roof connection. Because the mixture hardens quickly, underground dewatering and drainage are unnecessary [Henghu, 1993]. The cost of this product is assumed to be $130/ton, because it may cost twice as much as Portland cement due to transportation from overseas [Jones, 1995]. Cost analysis studies are still being conducted.
APPENDIX D

Derivations for Spreadsheet
This appendix explains how some of the values in the spreadsheet were derived. The values related to waste disposal include the landfill tip fee which was the average value for Virginia [Steuteville and Goldstein, 1993], the transport to landfill which was an Appalachian town trucking cost and is higher than an average value, the distance to landfill which related to the same town, and the waste tonnage which was the production rate for Virginia that went to landfills [Steuteville and Goldstein, 1993]. This last value can be changed in the following three ways: divide by 8 for the number of coal regions, divide by 10 to 15 for the number of large cities, or divide by 100 for the number of counties in Virginia to give a better estimate of the amount of waste received by the mine. The trucking cost was determined by taking the amount the town is charged by the trucking company and dividing it by the average distance to the landfill.

The values related to transportation include the transport to the recycler which used trucking rates, which as stated before have been estimated relatively high, to a recycling plant near the town, and transport to the mine which utilized transport by train. Transport by train was at a reduced rate, because the assumption was that there would be empty haul-back to the mine to begin with, so any additional charges would benefit the rail companies. Thus, the rate given was for fly ash haul-back. Fly ash haul-back to mines by train would also benefit coal mines, because contracts between the coal mine and power plant are beginning to require mines to dispose of fly ash [Bales, 1995]. A previously mentioned study evaluated the transport of municipal solid waste by train to
an open pit coal mine with the same train cars that took the coal to market [Carrier, 1978]. The transport was found to be feasible and issues such as car cleanliness could easily be resolved.

The values related to recycling were found from a case study performed by the U.S. Bureau of Mines in 1976 at a pilot scale recycling plant in which the values were converted to 1994 dollars. Recycling was 25%, because this was the percentage of metal, glass, and plastics in the waste stream [Stanczyk and DeCesare, 1985]. This percentage does not include paper, cardboard, or other organics because they lack secondary markets. Thus, it is possible to increase this highly conservative value when these secondary markets are doing well.

The values associated with the additives include density, cost, and transport of the material. Typically the additive cost accounts for transportation costs. In the case of the high water - quick setting cement the density was easily calculated with the assumption that the dry mix had the same density as portland cement. The cement mix tonnage was converted to compound tonnage and then the cost was calculated for the compound. If other additives are used, they will all be summed to give a cumulative additive cost [Henghu, 1993; Grogan, 1993; U.S. EPA, 1989; Kumar and Hudson, 1992].

The values that help in determining the costs of emplacement and costs associated with natural and artificial impermeable layers include depth of borehole, distance to
longwall panel from shaft, coal tonnage, coal seam height, coal density, impermeable layer costs, pumping costs, environmental monitoring costs, leachate collection, and leachate treatment costs. The depth and distance were used to calculate emplacement costs. The coal tonnage given is the average production rate for a longwall mine in the U.S. and the seam height is also the U.S. average for thin seams [Combs, 1994 and Combs, 1995]. The impermeable layer cost was determined by taking the lowest cost liner material used for landfills, which was soil-bentonite, and converting it into dollars per acre [Cheremisinoff, et al., 1979]. The pumping costs had to be converted from 1974 to 1994 dollars [NAS, 1975], and the environmental monitoring, leachate collection, and leachate treatment costs were from landfill cost estimates [Conn and Novak, 1990].

The final values which provide information on the cumulative benefits include the length and number of moves per year of the longwall, support method cost, face load reduction, and subsidence reduction. The first two values lead to a simple calculation of the rate of advance per year. When this is multiplied by the third value, the benefits received from tailgate support elimination are calculated. The first two values are thin seam longwall averages for the U.S. and the third value is for cable bolts. Other support methods include magnum donuts, $75 per foot of advance, and timber cribbing, $80 per foot of advance [McDonnell, et al., 1995]. So if a mine has been using one of these techniques for ground control more money can be saved. Tailgate support elimination and the following reductions are valid if the mine is completely filled. If a different percentage
is filled then it is assumed that the benefits reduce by the same percentage. Face load reduction was estimated at two million dollars, because an entire longwall system costs $10-$12 million [Ross, 1995]. It was assumed that with added support from the backfill, shields would not need to be as strong. Thus, a lower strength steel could be used which would allow for at least a two million dollar reduction. One issue not accounted for is the decrease in face confining pressure, which will allow for easier cutting of the coal. Contrary to the idea that stowing would reduce production, it should actually help improve it by requiring less time to cut the coal. This is assuming that a bunker system is in place for the waste stowage system and that there is no stoppage in the pipeline. One way to assure free flow is to have an alternative line available for use. The final value, subsidence reduction, uses a figure of $20,000 which is the amount to replace a cracked foundation for a house [Harwood, 1995] This figure is used to give a general idea of how much money can be saved by reducing subsidence. In addition, if subsidence interferes with water like a creek the subsidence costs begin to increase dramatically.
VITA

Janet A. Grimes was born on August 21, 1971 in San Jose, CA. In 1989 she graduated from Alhambra High School in Martinez, CA. She was accepted in the mineral engineering program at the University of California at Berkeley for the Fall 1989 term. Ms. Grimes graduated there in December of 1993 with a B.S. degree in Mineral Engineering. In January of 1994, she entered Virginia Polytechnic Institute and State University in pursuit of a M.S. degree in Mining and Minerals Engineering, which she received in December 1995.
ECONOMIC AND SYSTEM FEASIBILITY STUDY
OF MUNICIPAL WASTE STOWAGE IN UNDERGROUND COAL MINES

by

Janet A. Grimes

Chairman: Dr. Christopher Haycocks

Department of Mining and Minerals Engineering

(ABSTRACT)

Public concern about surface disposal of municipal waste offers the mining industry potentially enormous economic and environmental opportunities. If underground space created by mining can successfully be utilized for safe waste stowage during the mining process, there will be immediate and substantial benefits to all sectors of the underground mining industry. To investigate an integrated system of mining and waste stowage, an economic and feasibility model was developed. Major issues include waste transportation, emplacement area, waste characteristics after emplacement, and alterations to current mining operations. In this preliminary investigation, economic feasibility is the basis for comparison between alternative systems in this research. Past and existing underground waste disposal systems are used to evaluate the model.