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DEVELOPMENT AND ANALYSIS OF SLURRY TRANSPORT SYSTEMS IN LONGWALL MINING OF COAL

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A preliminary systems analysis of coarse-particle coal slurry pipelines as a haulage method for longwall mining systems was performed. Two exhaustive literature surveys on state-of-the-art longwall mining and slurry transport technologies were prepared. Two representative longwall systems were designed, one with conventional haulage and one with slurry haulage. The two systems were then compared on the bases of cost, safety, and reliability. It was concluded that for the case of a two-face longwall mine, a coarse slurry transport system does not have a clear-cut cost advantage over a conventional (conveyor-hoist) system. However, indications were that a cost advantage could occur under different analytical conditions such as (1) a larger scale mine, (2) if intangible benefits were quantified, and (3) if new generation technology rather than currently available technology were applied to the study.

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FOREWORD

This report was prepared by Columbia University, Henry Krumb School of Mines in the city of New York under USBM contract number J0333905. The contract was initiated under the Department of Energy Grant Program. It was administered under the technical direction of USBM Pittsburgh Research Center with Mr. Anthony Miscoe acting as Technical Project Officer. Mr. Michael L. Nowicki was the contract administrator for the Bureau of Mines. This report is a summary of the work recently completed as part of this contract during the period 30 September 1981 to 30 September 1982. This report was submitted by the authors on 30 September 1982.

The authors wish to thank Mr. Anthony Miscoe for his generous help in editing the final version of this report.

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INTRODUCTION AND SUMMARY

This report contains the results of all the work accomplished during the course of the project entitled, "Development and Analysis of Slurry Transport Systems in Longwall Mining of Coal."

The project, conducted by two mining engineering graduating seniors under the supervision of the principal investigator, is divided into two parts. The first part consists of two independent reports--one on longwall mining and the other on state-of-the-art, coarse-coal slurry transport technology. Integral to the development of these reports was a field trip organized in order to familiarize the students with the latest advances in longwall mining and slurry transport technology. On this trip two longwall operations and the Hydraulic Transport Research Facility (HTRF) in Bruceton, PA were visited. It was not possible to secure permission from the Consolidation Coal Company to visit its Loveridge facility, the only operating longwall slurry transport system in the Western world.

The second part of the project is devoted to the analysis of longwall haulage systems. To establish a comparison between conventional haulage and slurry haulage for a two-unit, longwall mine, a computer program developed previously in a U.S.D.O.E. study for the purpose of studying room-and-pillar/slurry-transport combinations was modified to apply to longwall-mining/slurry-transport combinations. Once an optimal longwall slurry haulage system was designed through the use of this modified program, it was compared with a conventional longwall haulage system on the bases of cost, equipment requirements, and certain intangible factors such as health and safety.

It has been found, keeping in mind the current immaturity of slurry haulage equipment and the lack of reliable technical and cost information on slurry transport systems, that for a two-face longwall mine the slurry transport option cannot be clearly identified as having a cost advantage over the conventional haulage option. The actual value of the intangible benefits accruing to the use of a slurry haulage system, however, was not included in the comparative cost analysis. Although these benefits (reduced ventilation requirements, fire hazard, accident rate, and absenteeism due to environmental health problems, for example) appear to be very large, budget and time restrictions prevented a semi-quantitative analysis of their magnitude.

Additionally, it must be noted that the comparative cost conclusion was based on a longwall mine with two faces, only. The study, however, found that as the scale of throughput increases, the haulage cost per ton by slurry transport falls much more rapidly than that by conveyor transport. This implies that the comparative cost conclusion arrived at in this study may be reversed if a similar study were done on a mine of a larger scale. Explicit confirmation of this concept was not possible under the budget and time constraints experienced by the study.

Another significant point about the comparative cost conclusion is that it may be outdated in a matter of months. Coarse-particle slurry technology is developing more rapidly than ever before. Recent investments in slurry research and development facilities are expected to result in a whole new generation of coarse-particle slurry equipment and flow theory in the very near future. Such an occurrence would totally change the circumstances under which this study's comparative cost analysis has been made and perhaps its conclusion, too.

In consideration of these points, it is suggested that a more detailed analysis is needed in the near future to evaluate the potential performance of the slurry longwall haulage system. Such a study, besides being able to use updated information on the imminent slurry technology developments, should examine a wide variety of longwall mine scales, should be able to optimize its proposed slurry systems globally, and should have stronger cost information retrieval ability (the current study was restricted in this respect by its budget and time limitations).

LONGWALL MINING

Historical Development

The earliest mining of coal dates prior to the Roman conquest of Great Britain. Coal had primarily been stripped from outcroppings and the depletion of this source drove man to develop subterranean techniques. Stall mining emerged as the most advantageous form of ore removal, but it was highly restrictive on output and suffered from low rates of recovery. The expanding market for coal soon forced suppliers to develop new means of providing more coal at greater rates. [101]

It was late in the seventeenth century, in Shropshire, England, when mining was introduced to a new concept. The imaginative approach entailed the development of long circular faces centered around a single shaft. Entries radiated from the focal point and were protected by packwalls, constructed with material that was either shot out of the roof and floor or transported in from the surface. The face was undercut while posts and cribs were advanced in step with the progression. The void behind the working area was allowed to close as the overlying strata subsided. Although superior to its predecessors, the long face was burdensome because of the intense labor required. Coal had to be hand drawn from the wall and loaded into carts as the timber was re-set. With the aid of technology, this cyclic activity would gradually transform into the method now referred to as longwall mining. [88]

In 1761, the first coal cutter was introduced to a British working face. The machinery was a composition of chains which drove a series of cutting tools that sliced into the coal. The apparatus was run by either hand or horse on a treadmill but suffered because neither could provide the power necessary for such a task. An alternate source of energy became available in 1853 when compressed air was introduced as a means of driving tools. In 1863, Thomas Harrison patented a disk cutter which soon became popular. By 1900 electricity had entered the picture and a whole family of coal-winning machines using disks, chains, picks and bars were experimented with, but, nevertheless, longwall remained a strenuous and clumsy form of extraction. [64]

It was in 1906 when Richard Sutcliff developed a 20" canvas conveyor belt driven by compressed air to move coal from the face. The introduction of automated haulage helped longwall take its first step towards a continuous configuration.

The mechanization changed both the layout and economics of longwalling. By 1913, the combination of coal cutter and conveyor resulted in record production of the material in Britain. Unfortunately, the method still demanded a great deal of manpower.

The cruel economic atmosphere which existed between 1913 and 1935 forced the mining industry to either trim their costs or exit the industry.

Great efforts were directed towards decreasing the labor intensive practice of longwalling. By 1940, the jib was effectively replacing sizeable portions of the production crew. The jib was designed to be locked perpendicularly at the end of the face and hauled itself along a rope attached to a set prop. The 4' to 6' cut allowed the coal to be drilled and blasted so the next shift could hand-fill and set wood or steel props. The last shift was responsible for dismantling and re-erecting the belt conveyor, as well as advancing the waste-side chocks and constructing strip packs. Caving was rarely practiced. Production was soon to be introduced to even further modernization.

In the late '40s several forms of coal winning and removal were attempted. The most popular model proved to be the Medco-Moore Cutter Loader. This unique piece of equipment encompassed two horizontal jibs, one cutting on the floor level, the other 2' to 3' above this, and a trailing vertical jib which sheared the coal. The ore would fall onto a cross conveyor and be carried to a transfer belt. The operation had one production shift per cycle and experienced a problem with coal build-up behind the unit. The Medco-Moore was improved by the addition of a German innovation called the "chain conveyor" which later proved to be the hinge pin for further advancement. [114]

The next benchmark in longwall history was the introduction of yielable hydraulic supports during the '50s. The new form of support could be rapidly moved and set, in addition to having pre-loaded and yielding characteristics. The prop-free face provided favorable conditions for mechanized mining and the chain conveyor offered to be a mobile haulage system. The Medco-Moore gave way to another generation of face equipment that operated with reduced webs and could utilize the new support. The elements for continuous production were now assembled. The shearer would move in front of the saw tooth pattern of hydraulic props in which the rear support advanced to the face after the pass. Although the pattern was a major breakthrough, the support system still required an army of workers and could not match the speed of the shearer. In addition, mines experienced ground control problems with the newly exposed cantilevered strata. [101]

During the late '50s and early '60s, sections of hydraulic jacks were united with a base plate and covered with a canopy, creating a new era in longwall mining. The supports, through their stability, created a versatile means by which the strata could be controlled in the face area. The use of coal cutters, chain conveyors and self advancing hydraulic supports has enabled longwall mining to gain acceptance throughout the world. Longwalls can be found wherever a coal mining industry exists. Canada, Mexico, Brazil, Chile, Argentina, Venezuela, and Poland are just a few countries which employ such operations. Ninety percent of all the coal produced by the Common Market is done so with longwall mining while the USSR leads the entire world in coal produced through this technique.

U.S. Growth

The acceptance of longwall mining in the United States was marked by a series of surges and withdrawals of application. The long face was first introduced to the domestic market in the late 1800s by an immigrant mining engineer in LaSalle, Illinois. [57] The energy demands of a growing nation and favorable geologic conditions assisted in the acceptance of the new approach and caused it to spread as far north as Washington and as far south as Alabama. Twenty-one long faces were put into operation, reached a depth of 750', and could produce up to 750 tons/day with a 42-man crew. The longwall market was captured in 1912 by the introduction of 200-ton capacity friction-yielding jacks which allowed greater productivity and better caving control. By 1935, the use of Langham, Lorraine and H-beam jacks in longwalls threatened the existence of room and pillar mining. The new support pushed the maximum production up to 850 tons/day and cut the crew size to 25 men with an optimum depth of 500'. Unfortunately, by 1940, unionization, the development of mobile loaders, the refinement of surface mining, and a weak market made thin-seam mining unprofitable, so longwall once again became dormant. [97]

Room-and-pillar and surface mining managed to fulfill the energy demands for a country at war and it was not until the early '50s before longwall mining attracted any interest. Metallurgical coal producers sought to exploit thin seams with greater efficiency through the use of a newly-developed, German plow. In 1951, the U.S. Bureau of Mines arranged a number of trial longwall mines owned by Eastern Fuel and Gas Association, Inc. The friction jack/coal plows were supplied by Mining Progress Inc. Numerous difficulties, arose but valuable experiences were gained. From 1953 to 1960 this combination was attempted in Pennsylvania, West Virginia and Arkansas. Crew sizes were reduced to 15 men, but production fell to a maximum of 700 tons/day due to poor caving, plow-resistant coal, increased down time and insufficient support capacity. Once again the prospects of longwall mining reached a low. [97]

It was at the 1959 American Mining Congress Coal Show in Cleveland when the domestic market was introduced to another technological achievement which resurrected the longwall method. Eastern Gas and Fuel Associates were impressed by the new yielding, self-advancing, 100-ton capacity hydraulic roof supports and quickly installed them in their Keystone Mine in 1960. The new supports and a variable-height, tandem plow loader were applied to a 340' face. The success of the Keystone project, in terms of roof control and productivity, encouraged others to invest in the technology. In 1961 the Kaiser Steel Company installed the supports and the first shearer loader. [72] By 1965 there were ten facilities in West Virginia, Illinois, Pennsylvania and Colorado. The maximum production achieved rose to 980 tons/day with a twelve man crew and a depth of up to 2000'. The characteristics of the massive overlying strata and soft floors in the U.S., however, lead to the demise of this equipment. [96]

The development of high-capacity hydraulic supports was the key to a U.S. commitment. In July of 1966, Island Creek Caol implemented the new equipment in its Beatrice Mine while Eastern Fuel installed theirs in August.

The number of faces grew to 27 in 1970 and 41 in 1972. [62] Depth of mining increased to 2500', while production peaked at 6000 tons/day. As the coal industry gradually embraced the longwall method, there was a steady development of custom supports and winning equipment. The Americanization of the machinery helped raise production to 8000 tons/day at a depth of 3000'. By 1975 there were 58 installations producing 3.1% of all deep-mined coal in the United States. [62]

In October of 1975, the industry was once again spurred by technological achievements through the introduction of shield supports. The market was also exposed to heavy-duty conveyor pans and chains with the perfection of double drum shearers. By 1979 there were 91 longwall faces, among which all the new installations (with the exception of one unique case) were using shields. [80]

Today, production varies with seam thickness and face length. The lowest daily production from a longwall with a 380' face, 40" seam, was 682 tons/day. The greatest daily tonnage from a similar seam, 360' face, 42-48" thick, was 2000. The lowest production from a larger panel 550' face and 86" seam was 1560 tons/day, while the largest production from a 600' face, 66-90" seam was 6000 tons/day. The record output is 17200 tons in a 450' face, 84-130" seam. The overall average daily production for domestic longwalls was reported as 3730 tons/day. [79]

The future demand for coal will influence the expansion of longwall faces. Conservative estimates indicate that by 1985, 192 longwalls will produce 12% of the U.S. deep coal. Meanwhile, many in the coal mining community hold to the conviction that in the same period of time, the number of faces will sky rocket to 247 and account for 25% of the deep coal. [61]

Advantages of Longwall Method

There are several forces pushing the coal producers towards longwall operations. The oil embargo of 1973 and subsequent events have created a national desire for energy independence. Compliance with public sentiment may cause production to rise to 1.6 billion tons by 1985. But, as the market expands, the precautionary measures dictated by the Federal Mine Health and Safety Act of 1969 have unfortunately resulted in a steady slip in productivity. The continuous, mechanical nature of longwall offers a unique solution.

The advantages offered by longwall are often cited as follows [121]:

(1) Higher production rate. It is estimated that by replacing 10% of the continuous miners with longwalls, coal production would rise by 20%.

(2) Greater control over subsidence.

(3) Successful multi-seam mining.

(4) Less down time due to the fact that there are fewer motors, gear trains and mobile equipment in longwalls.

(5) Good performance in areas geologically disturbed or previously-worked.

(6) Safer environment at the face due to the canopy which protects workers and improves the level of supervision.

(7) Better ventilation because air must pass a single producing face and encounter fewer crossings.

(8) Greater recovery. As coal mines reach a depth between 2000-3000' pillar sizes, outbursts, and rock bumps decrease the extraction ratio of room-and-pillar to 30% where longwall operations can easily recover 75%.

Disadvantages of Longwall Method

At the same time there are resisting forces which prevent an accelerated growth. The mining community, by nature, has a tendency to resist change, especially when it entails extensive capital tied up in equipment. This natural pessimism is compounded by the fear of applying new techniques.

The disadvantages of longwalling are seen as:

(1) More pronounced and costly delays;

(2) Higher moving costs;

(3) Higher capital costs, e.g., a medium longwall face costs 7.5 million dollars whereas continuous miners for the same region would cost between 1 to 1.5 million.

(4) Problems of finding qualified personnel;

(5) Problems of ground control in difficult-to-cave strata like sandstone or areas with soft roofs and floors.

(6) Difficulty of operating in seams of variable thickness. [79]

Development

The development principle behind longwall mining is to isolate and extract a block of coal while the peripheral ground is forced to support the redistributed stress field. The panels (blocks) are removed by driving parallel entries (gates) off of a main passage and working a long face that connects the drifts. The entry which is positioned on the unworked side of the panel is referred to as the head gate and must accommodate the movement of men, material, machinery ventilation and the belt conveyor. The entry which usually lies parallel to old workings is known as the tail gate is primarily used for ventilation.

The head and tail gates can be single-entry or composed of several unified passages separated by unmined chain pillars. The number and size of the passages is a function of roof and floor properties, ventilation, percentage of extraction desired, depth, and the geologic conditions. The optimum gate, in terms of development and recovery, would be the single entry, but due to U.S. regulations, this is not as yet feasible. Less than 10% of all domestic longwalls have one or two entries per gate while the majority operate with three or four entries which range in size from twelve to twenty-four feet in width. [19]

The spacing and lengths of the gates are governed by the same variables that control their placement. The separation and length of the head and tail gates determine the dimensions of the panel to be removed. The length varies from 900' to 6000' with the average U.S. mine being 3633'. The face has a range between 200' and 600' with the average being 466'. The direction in which the face is developed defines the two distinct methods of longwall production: "Retreat" and "Advance."

Retreat is initiated by completing the gates prior to extraction and connecting the two in the rear with a bleeder passage providing positive ventilation. The roof support, coal winning equipment, and conveyance are introduced at the back of the panel, and mining progresses towards the main drift. The shearer or plow slices into the face while the canopy is advanced, and fractured coal is carried off by chain and belt conveyors. This snake-like burrowing allows the coal to collapse behind the supports (referred to as the gob). The gob and bordering coal are forced to carry the weight of the overburden.

The retreat method requires the entries to stand and remain safe until production begins. The presence of strong roof strata in the U.S. which yield to relatively easy support through rock-bolting has been one of the primary factors leading to the almost unanimous adoption of Retreat mining. Presently, there are only two domestic mines which do not utilize this form of longwall mining. [99]

Retreat is viewed as advantageous because: (1) geologic conditions are exposed prior to mining; (2) the separation of development and production allows a consistent and faster rate of advance, in addition to higher output and efficiency; (3) there are no walls, pillars or gob packing; (4) roadway maintenance is reduced because disturbed regions are left behind; (5) spontaneous combustion is less likely. [100]

The greatest difficulty encountered by U.S. operators in retreat mining is the inability of gate development to proceed at a sufficiently rapid enough rate to replace face production. Continuous miners have achieved mining rates as high as 210' per shift for this purpose but remain underutilized. [13] Development may involve ten months of work while extraction may only take five months. Recently, the industry has expressed an interest in replacing the multiple openings with a single drift driven by a tunnel borer in order to increase the rate of formation. [103] This approach has not been widely-accepted.

The other variety of longwall mining is the advance process which runs in the opposite direction. Panel entries are driven just prior to face extraction, and thus the whole movement is towards the rear of the block. This operation is practiced in the United Kingdom where it is found to offer the following advantages: (1) at great depths, close to 4000', there are increased ground control problems so that entries cannot be made too far in advance; (2) extraction is immediate: there is no time lag between development and production; and (3) capital investment tied up in panel development is minimized. The system, however, suffers in the long run because total cost of roadway maintenance is higher. The advance longwall is considered as impractical for U.S. application because it violates several federal regulations, e.g., (1) the belt is located on the air intake; (2) there are multiple mining machines feeding on a single split of air; and (3) no intake escapeway is provided. [96]

British coal mines have encountered several set-backs with this technique. Development often proves to be too slow to either match the advance rate of the longwall or provide a large enough drift for the safe passage of men and material.

As a consequence of the advantages and disadvantages of both systems, hybrid configurations have developed. These entail the alternation of the panel removal method and the unification of head and tail gates leading to their reuse. [97]

Once the equipment has been set up, longwall mining becomes an efficient and continuous method. The shearer or plow passes along the face line ripping the material free. As the coal falls to the floor onto the panline, an armored chain conveyor drags it to a stage conveyor. The stage conveyor acts as an intermediate and transfers the coal to the belt located in the head gate. The roof support and face conveyance are advanced either in an immediate snaking motion or in unison as the shearer or plow passes. This formalized and consistent pattern continues until the shift changes, unwarranted delays occur, or the panel is mined-out. In all three cases the stoppage affects the available production time and, therefore, the productivity of the system.

In a recent attempt to identify and quantify the unscheduled delays, the Jet Propulsion Lab. performed an eleven month reliability/availability study on an operating longwall using a shearer. The data gathered covered 365 shifts, each 480 minutes long. Available working time was reduced to 381 minutes after inherent delays were accounted for. These inherent delays could be broken down to 40.5% travel time, 51.16% external conveyor failure and 7.9% other. The external conveyor failure, described as "belt slipped off" or "could not start," caused an average delay of 71 minutes per shift; 53% of these delays were less than 50 minutes and 25% greater than 90. The other category was dominated by "power out" which caused 39% of the lost time, averaging 61 minutes each.

The data compiled on unscheduled delays highlighted several weak links in the system. The shearer caused 44.2% of the lost production time with an average consumption of 113 minutes: 75% of these shearer delays lasted less than 105 minutes, and the average time between occurrences was 83 minutes. The chain conveyor had a tendency to get "hung up or stuck," leading to an average delay of 49 min. for every 404 min. Other face elements caused 5.9% of the delays, diminishing the available time by 35 minutes every 1970 min. Supports proved to be reliable and could only be blamed for 4.5% of the lost time with a stoppage averaging 87 minutes every 6258 minutes. Pumps claimed 2.8% of the time with 85% of these delays lasting for less than 50 min. every 4432 min. The sum-total of all the face delays resulted in a monthly variation in availability which ranged from a low of 56% to a high of 86% averaging 76.5%.

The largest scheduled delay in longwall production occurs when the equipment is moved to a new panel. Longwall moves take from 2.5 to 6 weeks, and the average panel extraction takes six months. This implies that up to 19% of production time is lost during this transition. Mines have sought for ways by which to shorten this period. Thorough and accurate planning appears to be the only solution arrived at so far. Maintenance material must be on hand, transportation equipment must be prepared, and both the old and new face must be ready to make the transition. If the movement is interrupted in any respect, the capital intensive equipment will sit idle. [71]

The Elements of the Longwall

Coal Winning Machines

- Plows

The plow is a set of static cutting tools arranged in a heavy steel frame which is pulled back and forth across the face and wedged into the coal to break it free. The action is somewhat like that of a wood plane. The cutting tool removes from 2-8' of the face but the average web is 3.8'. [96] The typical plow is used in a seam less than 42" thick where its size advantage is greatest. The width of the working area ranges from 10 to 12', and in a thin seam mine, the available area for the machine is reduced. The plow is fine for these low seams but suffers from stability problems when used for thicker applications. [88]

The plow commonly rides below the face conveyor and transverses at a speed between 100 and 300 feet/min. The most common speed is 200 feet/min. The higher rate of 300 feet/min. is used for tough coal but is avoided because it causes excessive maintenance problems. The lower rates of 100 feet/min. are used in soft coal where the face is under cut and abutment pressure is counted on to fracture the remaining coal. [96]

There are several varieties of plows in use which differ by varying their means of stability and their guidance and steering, in addition to the location of the haulage chain. In hook plows, the cutting and loading components are arranged on an articulated base which slides under the conveyor while the haulage chain is run through a covered guide on the back of the gob side of the pan. Another form has a guide plank and haulage chain both positioned on the face side. Both of these plows suffer from steering difficulties in inconsistent faces, soft floors, fine particle build-up under the base, difficulties in hard coal, and problems of face alignment. The Gleithobel plow has demonstrated an ability to hurdle many of these stumbling blocks. This variety is guided by a ramp like track and pulled on the face side while stabilizing arms reach over the conveyance. [19] Presently there are 21 plows in the U.S., 12 of which are the hook type and eight of the Gleithobel variety. [28]

- The Shearer

The shearer is a complicated and capital intensive piece of equipment. Basically, the machine is a self propelled frame on which are mounted motors, hydraulics and one or two rotating drums that have the capability of cutting from 24 to 36" into the face with the average being 30". The drums have ratings between 150 and 500 horsepower and can rotate at speeds varying from 30 to 120 RPM. [62] The trend of the longwall installation is to use 950V with the intent of reducing the cable sizes. The drums are equipped with either a hydraulic cowl which prevents coal from falling into newly-mined areas and can be flipped for the return trip, or hydraulic doors which operate in a similar fashion. The shearer is best suited for seams between 72 and 120" thick because of their versatility and size. [79]

The shearer rides on the edges of the armored face conveyor, being held and guided on the gob-side edge by the tubular arrangement. The riding action of this equipment, which can weigh as much as 25 tons, causes excessive wear on the conveyor chains, resulting in a life 1/5 that of those working with plows. The shearer pulls itself on a 1" diameter wire-size chain through an internal captan-drum arrangement using electric drive that are available up to 600kw. Speed is regulated by a gouvernor which monitors the voltage draw to the drums in order to recognize the presence of hard coal. The progression of a shearer across the longwall is commonly 8 to 9 feet/min., but the machine can achieve speeds up to 44 feet/min. [19]

There are several versions of the shearer. There are single fixed, single ranging, and double ranging drum configurations. The single fixed drum is the simplest, most inflexible and the most unpopular. The single, ranging drum has the cutting device attached to a mobile arm. This allows the shearer to remove from 24 to 81" coal thickness in a single pass or extract a thicker seam in two trips by ranging the height of the arm. The double ranging form is the most flexible and removes the full seam in a single pass by having a lead drum extract the upper 2/3 of the coal and the trailing drum the remainder. There are two types of double drum: those with the motor and hydraulics located in the frames (e.g., Eickoff, Anderson) and thos with lighter frames and the mentioned equipment oriented on the arms (e.g., Sagem). The choice between the single and double drum forms depends on the wall lengths, caving qualities, roof conditions, organization at the face end, and support design. The present trend is towards the installation of the double drum shearer. In 1980, there were 14 single ranging and 87 double drum shearers in the U.S. [73]

Anderson Mavor*	31
Eickhoff*	58
Sagem Sirus*	6
Sagem DTS	5
British Jeffrey	1
Joy Manuf. Co.	5
	107

*Made in USA

The following is a list of advantages and disadvantages of the plow and shearer systems:

Plows	Shearers
Additional blasting and shovelling required in areas of poor geologic condition, i.e., floor undulates, is soft, or hard streaks of coal exist	Unaffected by many geologic barriers, because it can adjust to such things as changes in seam height or coal hardness
Plows experience poor control in faulted regions	Shearer operates well in faulted strata
Support advances in small moves creating equal loading and minimizing exposed roof	Support is positioned under newly-exposed roof very quickly and fewer moves are required.
Produces coarse coal	Tendency to produce fine coal
Dust and methane liberation is minimized	Methane liberation is 5-10 times greater and dust suppression is accomplished with water sprays
Organization and manpower required are less important	Organization and manpower become more important
Purchase price is relatively low	Shearers are capital-intensive
Installation, repair, and maintenance are minimized due to fewer moving parts	Installation, repair and maintenance account for more of the total cost

Lower production rate	Higher production rate
Safer because men are not required to move with the machine	Not as safe because of the presence of more active pieces of equipment

Haulage

The purpose of the armored face conveyor is to create a continuous form of haulage from the working area, to provide a track for the cutter loader, and to serve as both a reference and anchor for the support advance. In order to fulfill this task, it must be horizontally flexible for conveyance advance without interfering with haulage and tramming while having a vertical flexibility to cope with seam undulation. The design of the AFC calls for a tensioned chain system divided with perpendicular flight bars to be dragged in a long pan, pulling fractured coal to the stage loader. The chain drive is electro-mechanical, commonly a 1800 RPM induction motor, 480 HP, hydraulic coupling and bevel spur gear located at either the head gate or both gates. The conveyance can achieve speeds of 3.5 mph (305'/min.) moving up to 18 tons/min. The typical chain moves at a speed of 214'/min with a capacity of 6.2 tons/min. consuming 4 Kw/ton plus .2 Kw/min. for the conveyance. [19]

The flight bars range up to 33" in length and run in line pans which are traditionally 59" long and flexibly connected to give them adaptability and robustness. The manufacturers, all foreign except Long-Airdox, vary their design but offer similar production rates. [73]

There are several chain options. Most longwall mines prefer three-chain conveyors (chain thickness of .708") because breakage of any one strand will not prevent movement. The second most popular configuration is the single-centered chain (1.18") because there is an equalization of load when the face is out of line. There is a strong movement towards the double-centered chain conveyor (1.02' to 1.18") for medium and thick seams because it offers the best of both alternatives. [19]

There are several inherent problems with the AFC. There is excessive wear on flight bars, chains (estimated life of 1.65 million production tons) and pans (2.75 million production tons). This problem is accentuated in shearer operations where excessive loads are inflicted. Blockage and overloads are common problems, and hang-ups occur when the chain loses tension. Dust is created by the scraping action, and fine coal recirculates or collects under the pan. [64]

The chain haulage leads to the stage loader which is responsible for the continual flow of coal from the face to a belt conveyor without interfering with the plow or shearer discharge. The stage loader is constructed similar to the AFC but has shorter and more widely-spaced flight bars that are run at a slightly greater speed between 250-300 ft/min. in order to prevent pile-ups. The connection to the main belt can be made by overlapping a panel belt (36x42") with the stage loader for a distance from 10' to 16', by using a boom of up to 75' between the two, or by having an hydraulically operated belt storage unit near the loading port. The stage and panel belt are advanced simultaneously and moved from 15 to 30' during the maintenance shift. Stage loaders are presently manufactured by Westfalia, Eickhoff, Dowty Meco, and Airwood Irwin. [19]

Roof Supports

The self advancing roof supports provide mobile ground control at the face creating a flexible and safe longwall system. The proper design is based on the underground mining conditions, geologic data, the coal cutting device, and the mine layout with respect to dimensions. The varieties available are the frame, chock and shield supports, accompanied by a family of hybrids.

The frame support is the most out-moded form being used in domestic longwalls. This support is constructed from parallel "frames" each of which has two or three jacks (legs) attached to a common base plate and roof bar. The roof bar cantilevers over the front leg from 45 to 108", supporting from 38 to 92% of the roof area. The frames are joined by one or two shifting cylinders locked to the base. Steel plates or slats hang from the canopy or are strapped to the legs. In order to advance the unit, one frame is lowered and pushed foreward by the connecting cylinder. After this, the leading frame is set and the trailing unit is lowered and pulled to the face where it sets. The walking is unique to the frame. Frames can be equipped with push rods to advance the face conveyance. The yield capacity ranges from 200 to 1050 tons while leg pressures range between 2500 and 7200 psi. The maximum floor pressure encountered varies from 61 to 474 psi. The support height available ranges from 32 to 127". The frame offers the advantage of being able to re-orient the face and accommodate floor undulations more readily. The present U.S. application of frame supports is limited to nine faces. [63]

The chock system encompasses from two to six legs united into a single unit by a rigid base plate. The base has a hydraulic ram which advances the face conveyance and pulls the support forward when the legs are unloaded. With this variety, the exposed roof area is greater so the unit spacing is smaller. The center to center distance ranges from 48" to 53" and allows roof coverage between 70 and 100%. Chocks operate at heights ranging from 28 to 144' with maximum vertical travel from 25 to 94". The legs can yield from 150 to 800 tons with leg pressures from 2150 to 8560 psi. The maximum floor pressure created varies from 22 to 535 psi. There are forms of chocks which have a hinged steel plate gob shield fastened between the roof canopy and the base. Presently there are 48 chock-type faces of the four leg form with capacities from 280 to 720 tons in the U.S. [63]

Both the frame and chock experience similar difficulties. The nature of the design directs the support thrust vertically through the legs, behind the face. The cantilever dependence results in a limitation upon support capability. The nature of these units also induces poor stability on inclined seams and insufficient resistance to lateral migration of strata. Lastly, the chock and frame offer minimal protection from the gob.

The most advanced, and most popular development, is the shield support. This equipment is composed of a base with a gob shield hinged at the rear, extending up and over to support the canopy. The canopy is hinged to the gob shield and extends over the working area. The shield is operated with one to six legs and some have actuators to extend and tilt the canopy. Roof coverage of 100% is easily obtained. Yield capacities range from 115 to 595 tons with pressures between 490 and 7500 psi created in the legs. The maximum floor pressures run from 56 to 515 psi while the roof pressure is between 50 and 185 psi. [73]

The shield supports offer several advantages. The three pointed system offers greater vertical and lateral stability. There are no gaps in the roof support which decrease the quantity of unwanted material entering the working face, thus resulting in safer conditions, lower maintenance and more rapid advance. The shields also have the ability to be advanced under pressure. In addition, the primary roof load is displaced closer to the face, improving caving conditions and aiding control problems with weak capping strata. Finally, the support is easily dismantled, transported and can be reassembled in as little as 40 minutes. [22]

There are 128 longwalls in the U.S.A. and 71 of these utilize shield supports. Of these, 41 longwalls have shields with two legs and yield capacities ranging from 350 to 400 tons; 21 have four legs with 570 to 610 ton yields, and only one has six legs. The manufacturers are:

Dowty Mining Inter.	22
Klockner-Becorit	17
Hemscherdt	12
Westfalia	9
Thyssen	6
Joy Manuf. Co.	4
Gullick Dobson	1

Costs

The costs of installing and operating a longwall unit are dependent upon the individual mining conditions. Each application requires a tailored approach related to seam thickness and panel size.* Due to the tremendous

*Presently, longwall units cost between $15 and $20 thousand per foot of face in the United States.

variation found in present coal mines, it is impossible to supply an exact figure to encompass the general design. [31]

Roof support, the largest portion of the capital outlay, and the face conveyance costs both vary linearly with panel width while complimentary materials costs vary at a non-uniform rate. Seam thickness will dictate the form of coal cutter to be used which, in fact, fixes this part of the investment. Thin seams using plows can mean a 10 to 15% reduction over the shearer with respect to coal winning machines and face conveyance. Seam thickness will also alter support costs in a stepwise fashion. In the following list, the panel length is held constant while the seam height is changed (note the rise in final costs). [96]

Height Range (inches)	Capital Cost $
40 - 60	4,500,000
50 - 80	5,845,000
60-108	6,660,000
90-168	7,668,000

COARSE-COAL SLURRY TRANSPORTATION

The linkage of a slurry pipeline to a longwall miner is one of the next logical steps in the sequence of the development of coarse-particle slurry technology. The large, general problems of characterization of slurry flow and development of equipment to handle the flow are being steadily resolved. It is now necessary to consider the smaller task of studying specific applications of the slurry technology. One such application is the connection of a slurry system to a longwall mining system.

The following intends to review the historical development of coarse-particle slurry transport, describe the current state-of-the-art system and the problems it has to deal with.

History

Coal slurry technology has developed in a manner similar to a growing plant: starting with the same material at the root, branches develop differently if they face different environments. A branch in the sun, for example, may develop in a manner varied from one in the shade. In the case of the growth of slurry transportation, two trends are noticeable: one group of countries (East Bloc and a few Western European nations) has focussed its attentions primarily on coarse-particle slurry technology while the other (West Bloc , particularly the U.S.) has emphasized fine-particle technology. The single factor most responsible for this divergence is hydraulic mining: the first group, having steeply-dipping seams, finds it productive to employ hydraulic mining, while the second group, having relatively level seams, need not resort to this method.

The two branches started from the same root. Before the discovery of hydraulic coal mining in the 1930s, both groups faced the same stock of information upon which to build coal hydrotransport systems: (1) the hydraulic dredge had been discovered in Germany in 1856 [12]; (2) jet elevation, a system where solids are entrained and propelled through a pipe by a jet of fluid, had been discovered during the California Gold Rush in the late 1850s [116]; (3) hydraulic stowage with fine and coarse particles had been in use since the 1880s [124]; and (4) slurry lines had come into common usage to dispose of various wastes in the 1930s [118].

In 1935, however, the paths of the two groups began to diverge with the development of hydraulic coal mining in the U.S.S.R. A large proportion of Soviet coal seams are steeply dipping and thus difficult to exploit. In 1935 a Soviet scientist, Dr. V. S. Muchnik, proposed a method which would effectively mine these areas. Earlier in the century the U.S.S.R. had been using hydraulic monitors to mine peat and remove overburden. As they developed hydro-equipment with higher pressure capabilities, Muchnik suggested that they apply it directly to coal seams, particularly those that were steeply-dipping (the Soviets now claim that hdyromining systems allow them to exploit up to 85% of reserves in difficult mining conditions when the dip is in excess of 10 degrees, compared to 72% extraction by conventional mechanized methods in the best mining conditions). [7, 81]

This suggestion had a momentous impact on slurry transport's history because it created a mining environment where virtually the only acceptable mode of materials handling was hydrotransport. The use of hydraulic monitors produced, all within the extremely tight space limitations of an underground coal mine, a flow of coarse, unsized particles mixed in the runoff stream from the monitor's water jet. The materials handling system best suited to such an environment, of course, was slurry transport, more specifically, <u>coarse-particle</u> slurry transport.

Experimental trials for Muchnik's system were successfully conducted in 1936/37, and the world's first coal hydromine was subsequently constructed --a mine that laid the basic pattern for hydromining and hydrotransport for the next several years.

This pattern involves a hydraulic monitor ejecting a high-velocity jet of water against a coal seam which breaks off chunks which are then carried by the run-off water down either an incised channel in the floor or in a flume to a shaft bottom sump area. At this point, either the material is sized, with -6mm particles being thickened and centrifugally pumped up shaft and +6mm particles being hoisted, or else all the material is pumped after being subjected to crushing to a maximum diameter. Later, with further technological development, if the shaft were a great distance from the mined face, then upon leaving the flumes, the slurry would be crushed, injected into a coarse-particle slurry pipeline, and pumped to shaft-bottom. [86]

After being temporarily side-tracked by World War II, this Soviet hydromining system began to spread; by the early 1950s it was being utilized not only in the U.S.S.R. but also in Poland and Czechoslovakia. These countries, however, began to add their own embellishments. Having to deal with deeper seam than the Soviets, they found that the centrifugal slurry pump system could not deliver a sufficiently high head to get the coal out of their deeper shafts. They began to develop slurry feeders (Poland started research in 1953 and had a prototype by 1957; the Czechs had one by 1958) which would separate the solid particles from contact with the pump, allowing a more efficient, higher-head pump to be used.* [6]

*Although the U.S.S.R. has developed a lockhopper feeder, it has never been commercially used. All current hydromines use either centrifugal pumps in series or airlift pumps. [86]

By the late Fifties, hydromines were being constructed frequently. In the 1956-1960 five-year plan, the Soviet government scheduled 69 hydromining units to be deployed [70], and the Poles built several large hydrocollieries such as the Sjerza, Radoszowa, Andaluzja, and Deviensko mines (the latter having a 1000-foot lift distance). The expertise was exported to new areas: the Poles developed several hydromines in India and China, including the world's largest, the Lu-Cja-to, in 1958 (the Chinese currently have ten hydromines) [66], while the Soviets created the Hydrocoal Institute in 1957 which eventually leased out its hydraulic technology to the Mitsui Mining Co. for use in Japan (which by 1972 had six hydromines) and to Kaiser Resources Ltd. for use in Canada. [129, 44]

Construction of hydromines in the U.S.S.R. continued on into the Sixties but a new, more sophisticated design concept began to surface which advanced hydrotransport technology considerably. The old hydrocolliery design usually utilized hydrotransport only for hoisting or for short horizontal runs. Once the coal reached the surface, it was dewatered, loaded into rail cars, and shipped to its destination. A significant proportion of the total materials handling cost originated from flaws in the rail system: coal was lost over the sides of the cars and transfers (loading and unloading) were costly. The Soviets realized that these problems could be eliminated through the construction of large complexes in which a permanent consumer of coal would be directly connected to a permanent producer by a coarse-particle slurry transport line. This way the coal would be contained in a single, enclosed system from the bottom of the mine to the doorstep of the consumer.

A typical example of such a complex is the Yubileynaya (Jubilee) mine in the Kuznets Basin, constructed in 1966 (see Appendix I). Chunks of coal are flumed to shaft bottom where they are crushed to -70mm and hydrohoisted 200 meters to a surface storage tank. The slurry is then pumped eleven kilometers to a preparation plant located next to the ultimate consumer: the Western Siberia Metallurgical Plant, a steel producer. [81, 86]

Several such complexes were built in the Sixties and are continuing to be built today. Three projects being currently proposed are coarse particle hydrotransport lines from: (1) the Ekibastuz mine to the Bulkhashsk regional power plant; (2) mines in the Kuznets basin to the Perim regional power plant; and (3) mines in Dobropolskiy rayon to the Chigirinskaya regional power plant. Additionally, many rail links between old hydromines and their preparation plants are currently being replaced by hydrotransport pipelines.

This new twist in the use of hydrotransport forced the rapid development of its technology. Previously, with only short distances being covered, fine-tuning of the slurry lines was unnecessary since the line made up only a small portion of the entire mining project. For example: flow conditions in the lines were determined by rough rules of thumb since non-optimal operation did not significantly affect total costs, little attention was paid to the amount of pipewear since relatively little was invested in pipe, and coal particle degradation could be overlooked since only small amounts occurred over the short length of the pipeline.

All this changed, however, with the advent of the need for a long-distance coarse-particle slurry line. Suddenly the pipeline incurred a much larger proportion of the total costs, making the optimization a necessity. Allowing the slurry to flow in an inefficient mode added greatly to power costs, thus forcing research into refining the mathematical models of slurry flow. Large amounts of capital had to be invested in the long pipelines, thus forcing inquiry into means of resisting the pipewear inflicted by the slurry. Coal particle degradation was extensive over the length of pipeline, thus forcing research into a way of predicting the degree of degradation so that the preparation plant could be designed having the proper amount of capacity devoted to fine particles. This research effort has probably placed the U.S.S.R. in the position of world leader in coarse-particle slurry technology.

While the Soviets may show lots of experience and sophistication with coarse-particle slurry lines, this is not so with fine-particle slurry lines. Soviet hydrotransport has always been linked directly to the mine--to an environment where it must handle an uncontrolled feed (the randomly sized chunks of coal being washed off a face by a hydrominer, for example). Recently, however, the U.S.S.R. has shown interest in the hydrotransport of "prepared" slurries, i.e., fine-particle slurry pipelines.

The Soviet scientist, N. E. Ofengenden, explains this interest as being a result of the development of large-scale power engineering and the separation by great distances of energy sources from energy consumers. As of 1980, Soviet fine-particle slurry lines were, at best, still in the planning stages. Some of the proposed projects are:

- 250 km line from the Kuznets Basin to Novosibirsk (planning started in 1979).
- 420 to 450 km line from the Donetz Basin to the Black Sea region (solids conc. = 50%, mean particle diameter = 1.3mm, 4.2 million tpy). Sponsored by UkrNIIGidrougol--a research institute.
- 436 km Dnepr regional power plant line (50% solids concentration, a 1mm mean particle diameter, 4.3 million tpy). Sponsored by Tepoelektroproyekt.
- 2,500 to 4,000 km line from the Kansk-Achinsk Basin to the center of the country--has only been suggested. Product is a brown coal which combusts spontaneously and thus cannot be transported by railcar.

Other nations whose coarse-particle hydrotransport systems were based on or tied to hydraulic mining are Japan, Great Britain, and West Germany. In 1953, for example, the Japanese government's Coal Mining Research Center and the Tokyo Institute of Technology joined to develop a hydrotransport system. Hydrotransport had been under consideration for some time as this mode of materials handling was seen as being compatible with the dewatering problem present in several Japanese mines. The pipefeeder developed through this program was later combined with hydromining technology obtained by license from the U.S.S.R.'s Hydrocoal Institute by the Mitsui Mining Company and installed in a few mines. [40, 44]

Great Britain was also interested in hydromining. Having initiated research into hydrotransport in the early Fifties, from 1959 to 1960 the National Coal Board experimented with a hydromining section in the Trelewis drift mine. The concept never took hold, however, and no commercial facilities were constructed. The NCB, nevertheless, continues to investigate hydromining. [40]

While with Japan and Great Britain hydrotransport developed before hydromining, in West Germany hydrotransport developed as a result of hydromining efforts. Ruhrkohle AG, with West German government support, has been involved in an extensive R&D program to construct an hydraulic mining system since 1957. From 1958 to 1961 experimental hydromining of steep seams was conducted at the Robert Müser mine (at which coal was flumed to shaft-bottom and hydrohoisted to the surface). From 1962 to 1970, four more small-scale pilot plants were established (including facilities at the Consolidation and Dahlhausen Tiefbau mines). [40, 44]

Harzer claims that the need for pipeline hydrotransport as opposed to flume transport was apparent from the earliest hydromining experiments and, as a result, has since been closely studied in Germany. [40] This is particularly evident in the later pilot projects which utilized increasingly sophisticated slurry technology. In two large-scale experimental installations constructed in 1971, for example, one, the Carl Funke Mine, contained a five-kilometer underground slurry pipeline for -1 mm coal particles and a vertical lift of 700 m, while the other, the Gneisenau Mine, had a set-up in which all coal was crushed to -60 mm, fed into a new two-chamber pipefeeder, and pumped 700 m to the surface. [40, 44]

In 1977 the results from all of these pilot programs were lumped together and incarnated into West Germany's first commercial hydromine, the Hansa Mine (see Appendix 1). The mine was closed, however, in 1980 due to failure primarily in the hydromining sector. In the meantime, interest in hydrotransport has expanded to other applications, such as debris disposal in shaft-boring and drift-driving operations. [40, 105]

Western Development

The West never had any compelling stimulus like the invention of hydromining that the U.S.S.R. experienced which focused its hydrotransport effort underground and thus tied it almost irrevocably to a coarse-particle system. Having no need for hydraulic mining since its coal seams were all relatively flat, development of hydrotransport in the U.S. was directed by an entirely different set of stimuli.

Originally, the West had the same stock of slurry information as the East: (1) German invention of the hydraulic dredge in 1856 and reinvention in the U.S. in 1876; (2) jet elevation during the California Gold Rush in the late 1850s; (3) hydraulic stowage emerging in the 1880s; and (4) slurry waste lines.

The years between these early precedents and the first flurry of engineered slurry pipelines in the 1950s were not entirely vacant. In this time period, the U.S. was steadily amassing a stock of experience that would allow it to make the big technical advancements in later years.

Although little was done in terms of actual pipeline construction from the first patent on coal slurry transport in 1895 well into the 1920s, there were some signs of progress. First, the idea was attracting a lot of enthusiasm: a search in the mining literature of this period reveals a wealth of articles speculating on the potential of slurry transport applications and even proposing ambitious long-distance pipeline projects. In 1921, for example, three different plans for coarse-particle slurry transport lines arose: a 130-mile line from Scranton, PA to the New York City area, a 195-mile line from Scranton, and a 322-mile line from Clearfield PA.

Second, significant research was being done in other fields which would later be useful in slurry pipeline design. Mineral processing research in this period yielded float-sink theories which would later be used in understanding the solid-liquid interaction in slurry flow. Additionally, a few scientists were beginning to look at the slurry transport occurring in dredging and making basic observations on characteristics which would later become of extreme interest to coal pipeline designers: the vertical distribution of sand flowing in a dredge pipe and friction loss in hydraulic dredging, for example.

Third, a few slurry pipelines were actually constructed. In 1900 the phosphate industry began using slurry lines for the first time to dispose of the overburden overlying the phosphate matrix. [43] Additionally, in 1913, an Englishman by the name of Gilbert Bell constructed a 660-yard coarse-particle slurry line that transported 5" pieces of coal in an 8" pipe from the Thames River to a power plant and operated it successfully for eleven years. He then proposed a similar line 130 miles long which was never constructed.

In the late Twenties and in the Thirties, the pipelines which had been talked about for so long were finally being built. Slurry lines transporting phosphate matrix became commonplace in Florida. [43] Waste lines transporting sludge, such as a one-mile anthracite culm line in Pennsylvania constructed in the late Twenties also became common. It is important to realize, however, that the design of these pipelines was based on past experience rather than any well-developed theory. No developed theory existed, according to Bain and Bonnington [3], because the economic scale of these projects did not justify the expensive research needed to optimize their operation. Instead, the slurry lines were designed according to rough rules-of-thumb which allowed wide design margins.

In designing a fine-particle waste line from a preparation plant to a settling pond, for example, the engineer would mimic as closely as possible any previous project that had shown itself to be successful. The pipeline was often laid without any further assurance that it would, indeed, function.

If the slurry was transported when injected into the line, then the project was successful. If the slurry did not make it, the engineer would tinker around with the pump speed and the slurry concentration until, hopefully, some successful combination was found. While no coherent theory came of these efforts, a wide foundation of experience was accumulated that could be put to use in later years.

Hydraulic dredging, also, became commonplace in the U.S. during the Thirties and on into the early Forties as the result of a California gold rush induced primarily by the Depression. This occurrence had a significant effect on the development of solid/liquid flow theory. In contrast to coal sludge waste lines, the hydrotransport system in hydraulic dredging has to be relatively fine-tuned. The flow of dredged material is more complex than the flow of fine, uniformly-sized particles coming from a preparation plant in a waste line, thus making the simple trial-and-error method then utilized in waste line design less likely to succeed.

Additionally, the economic impact of non-optimal flow conditions is negligible in the case when the slurry line is a small component in a large complex like a sludge line in a colliery, but very significant in the case of hydraulic dredging in which hydrotransport is the single most indispensable component in the system. All in all, dredging demanded a deeper understanding of solid/liquid flow than the other contemporary slurry applications. Several dredging operations failed as a result of a lack of this understanding. The threat of failure perhaps induced more research into finding a quantitative characterization of slurry flow. Surveying the literature of the period, one finds a sudden shift from articles giving operational anecdotes and rules of thumb to ones attempting, for the first time, to quantify relationships, most of these articles being based on and applying to sand transport in dredging.

These theoretical foundations, the mass of operating experience collected, and then certain technological innovations (specifically, the emergence of high pressure pipe manufacture [3] set the stage for the big developments of the Fifties: (1) accurate fine-particle slurry head-loss equations; and (2) the first long-distance fine-particle slurry pipelines.

In 1951, the Consolidation Coal Company announced that it had finished constructing a demonstration facility which would be the precursor of the 108-mile pipeline it would construct in 1957. Just as the Soviet slurry system evolved into coarse-particle transport because it had to be compatible with hydraulic mining, the U.S. slurry system became fine-particle because it had to be compatible with long distance transport.

Transport over a long distance required a fine-particle slurry rather than a coarse slurry because: (1) typically, only centrifugal pumps can handle large particles. These pumps, however, have such a low head capacity that too many pumping stations would be required. A high head positive displacement pump would reduce the number of pumping stations, but at a cost of making the slurry of fine particles, since only these would pass through the pump's valves; (2) a fine-particle slurry has lower friction losses than a coarse-particle slurry. It was determined that in the case of long distances, the savings in power would outweigh the extra costs of grinding and dewatering fine particles. As a result of these decisions, Consol's demonstration project was tested with -3/8" particles (although the line was only 17,000 feet long). [29]

In 1952, a breakthrough was made in slurry headloss theory. At a hydrotransport conference held by Great Britain's National Coal Board, Durand and Condolios presented a paper describing a correlation between headloss and several flow and pipeline parameters. Following the correlation came a generation of headloss equations, a few of which had enoug predictive accuracy to design large fine-particle slurry pipelines.

The first of these pipelines were built in 1957 and others have been constructed on a regular basis since then:

1957: American Gilsonite constructs a 72-mile line.
Consolidation Coal completes its 108-mile coal line.
Anaconda builds a 14-mile copper concentrate line.
1964: 57-mile Rugby limestone line.
1967: 53-mile Savage River iron ore concentrate line.
1970: 273-mile Black Mesa coal line.
1971: 17-mile Calveras limestone line.
1972: 17-mile Bougainville copper concentrate line.
1973: 69-mile Ertsberg, Irian Jaya copper concentrate line.
1974: 28-mile Peña Colorado iron concentrate line.
11-mile Pinto Valley copper concentrate line.
1976: 17-mile Las Truchas iron concentrate line.
20-mile Sierra Grande iron concentrate line. [118]

A number of large-scale projects are currently being proposed, but have been held up by eminent domain problems. One example is the 1,036-mile, 25 million tons per year fine-particle coal line from Wyoming to Arkansas advanced by Energy Transportation Systems Inc. (ETSI).

The dramatic progress of fine-particle technology overshadowed another movement occurring simultaneously--that of coarse-particle hydrotransport. Although very few installations were constructed before 1950, interest was high. Long-distance coarse-particle slurry lines were proposed in the early Twenties, as well as hydrohoisting systems for pumping ore out of underground mines. In the mid-1930s a 300-mile pipeline carrying six-inch coal was studied by the Koppers Company. [29] In the Forties hydrohoisting using a heavy-medium slurry was suggested. [133, 134] When the U.S. Bureau of Mines showed its first interest in hydrotransport in 1950, it considered primarily long-distance, coarse-particle pipelines, specifically, slurry lines 10" to 34" in diameter carrying -1.5" to -3/8" coal hundreds of miles. [29]

The first actual physical manifestations of coarse particle hydrotransport, however, came in the form of hydrohoisting. The first record of a hydraulic hoisting operation appears in 1951--a Winsconsin zinc mine installed a lockhopper feeder (used to feed particles into a pipeline under pressure) which could inject 4" particles into a 10" pipeline running from shaftbottom to the surface. [3, 6]

Further development occurred around the same time in Great Britain. A serious mine fire in 1950 involving a conveyor belt probably prompted the National Coal Board to look for alternative methods of materials handling involving less dust and danger of fire. Seizing upon the hydrotransport ideas floating around, they began to sponsor further research and development. In 1952, a conference was held at which articles on solid/liquid flow theory (including the one by Durand exhibiting his new correlation) and the first British experimental lockhopper feeder were presented. Great Britain would continue to develop and test several different feeders at its Markham and Woodend experimental mines during the rest of the Fifties and into the Sixties, but ultimately none of them were successful as a result of size degradation, wear, and leakage. [82]

The French were simultaneously becoming interested in coarse-particle hydrotransport. Durand's famous 1952 headloss correlation was based on data gathered at Sogreah (the research arm of a French engineering company, Neyrpic) facilities. In 1958 two coarse-particle horizontal transport lines were constructed, based on Sogreah studies: (1) a 4.8 km line carrying -50 mm soot and cinder particles from the Kincardine Steam Power Station in Scotland; and (2) a 600-meter granulated slag (2 to 50 mm particles) line. [21]

In 1959, intrigued by the potential of hydrohoisting, the French government's Coal Mining Research Center (CERCHAR) funded the construction of a hydrohoisting pilot project by Sogreah and Neyrpic at the Devillaine Mine near St. Etienne. Wanting to avoid the pump wear and coal degradation involved in sending a coarse slurry through a series of centrifugal pumps, a lockhopper feeder device was developed which hoisted 50 to 60 tph of -80 mm ROM coal up 590' and 215' horizontally. [20, 21] The use of such a system, however, did not spread, and French involvement in coarse-particle slurry development faded away.

The U.S. Bureau of Mines, although expressing interest in slurry pipeline hydrotransport as early as 1950, did not become actively involved in R&D until the Sixties, when it, also, tried to develop a lockhopper feeder and researched into solid/liquid flow theory. Two test facilities were constructed: one in 1964 and the other in 1969. The lockhopper tested in the project was a failure: due to its complexity and the sluggish performance of its relays, timers, and valve operators, it could neither operate reliably nor attain a concentration higher than 17%. [82]

In the mid-1960s a new development emerged which not only provided new justification for hydrotransport technology research, but also expanded what interest there was in hydrotransport from just hydrohoisting to a hydrohoist linked to an underground, horizontal, coarse-particle slurry pipeline, i.e., a hydraulic system running from the face to the surface.

In 1947 the first continuous miner was invented and its production capacity and rate of advance rapidly outstripped the mode of materials handling common at that time: the chain conveyor. Even when the chain conveyor was replaced by the shuttle car, which had a much higher capacity, it was becoming evident by the mid-Sixties that this substitution could not resolve the capacity problem--as a result of better continuous miners and the mining of thinner seams (which forces the use of smaller shuttle cars) an underground bottleneck was occurring in which coal was being mined faster than it could be removed, thus leaving the continuous miner inefficiently idle. [15, 76]

Studies have indicated that "nearly 25% of the continous miner cycle time is consumed by delays in the haulage system, usually waiting for shuttle cars." [74] Mining companies began to look for alternate methods of haulage which would allow full utilization of the continous miner's capacity. Instead of choosing larger "batch" systems, they preferred continuous systems. In 1965, the Consolidation Coal Company began to investigate coarse-particle hydrotransport as a possible solution to the capacity problem. In 1966 it obtained a patent for its concept of continuous underground hydraulic haulage, and in 1969 it established a long-term research and development program to come up with a commercial system it could apply in its mines. [27] This decision to go ahead with hydrotransport was reinforced by the sudden emphasis placed on safety after 1968, the hydraulic system having several features which suggested it as potentially much safer than conventional haulage methods.

The first work was done in 1970 at the Continental Pipeline test loop (the same one used to run tests for the Ohio fine-particle pipeline) at Continental Oil's Research and Development Center in Ponca City, Oklahoma. Two carloads of -2.5" coal were brought in and injected into a 650-feet loop of 8" line to determine headloss, coal particle degradation, handling characteristics of large-diameter rubber hose, and optimal mixing sump design. Further work was then done at a temporary facility at Consol's Humphrey #7 mine where coal was more accessible for the test runs. Here, slurry pumps, mixing sumps, and their control devices were tested under a wide variety of flow conditions. [26]

In 1973, all of the various components based on work at Ponca City and Humphrey #7 were brought together and tested as an integral system at the Robinson Run Mine (see Appendix I). In this set-up, a mobile crusher-feeder accepted coal from a continuous miner and injected it into a flexible 10" rubber hose which allowed the crusher feeder to follow the miner's movements. The hose then connected to a steel pipeline with a variable-speed booster pump which transported the -4" coal 2,950-feet horizontally and 115-feet vertically to a preparation plant. [26]

Further development work followed. Consol ran 330,000 tons of -5" raw coal through a variety of pipe sizes and grades at its Loveridge Mine to get an idea of what sort of pipe wear problem would exist. A specific combination of pipe and corrosion inhibitor was chosen by 1976. [27] Meanwhile, in Ponca City, a multiple-feed sump was designed and tested which would accept flows from several sources and emit a single, smooth flow of slurry with adjusted concentration. The integrated system was tested once again at the Robinson Run Mine, and then a full-scale commercial installation at Loveridge was constructed in 1978 (see Appendix I). The first coal from this hydrotransport system (which at one point accepted coal from two continuous miners and one longwall face) was pumped in 1980 and experimentation is continuing there currently. [27, 76]

Another organization interested in coming up with a continuous transportation solution to the materials handling bottleneck was the U.S. Bureau of Mines. In 1975 it started a long-term research effort to look into a variety of proposed continuous haulage systems, one of which was coarse-particle hydrotransport (in October of 1977 this work was transferred to the newly-established Department of Energy). [74] Areas of contracted research have been the following:

1. A feasibility study of a continuous coarse-particle hydrotransport system roughly similar to that conceived by Consol called Feasibility of Hydraulic Transportation in Underground Coal Mining, by Link, Allen, and Faddick of the Colorado School of Mines Research Institute in 1975. [67]

2. A study, Control and Automation of Hydraulic Transport Concepts for Coal Haulage in Underground Mines, analyzing the compatibility of coarse-slurry transport with varying degrees of automation. [111]

3. An investigation into the economically optimal layouts of a slurry system in a multiface mine--Evaluation of Alternate Hydraulic Transport Concepts for Coal Haulage in Underground Mines. [112]

4. The theoretical and operational study of coarse-coal hydrotransport. To expedite this, the U.S.D.O.E. constructed the world's largest coarse-particle test facility of its kind: the Hydraulic Transport Research Facility at the Pittsburgh Mining Technology Center (see Appendix I). The essential purpose of the project is to: "generate design and operating data for the transport of run-of-mine coal in underground mines. [35] Also aiming towards this objective is the Separate Effects Test Stand run by Battelle Pacific Northwest Laboratories. This unit is a rotating pipe ring which is used to obtain data on pressure drop, coal particle size degradation, and wear. [83]

5. The conception and development of a sensor able to measure accurately, simultaneously, and continuously the coal, refuse, and water concentrations of a flowing slurry. The sensor, actually a combination of neutron beam, conductivity, and gamma-ray sensors, has been fabricated and installed in the U.S.D.O.E.'s hydraulic Transport Research Facility for evaluation. Additional sensors are being constructed for independent evaluation in Canada (Canada Centre for Mineral and Energy Technology) and West Germany (Steinkohlenbergbauverein). [83]

6. The examination of plugging and its cures. Batelle Pacific Northwest Laboratories is under contract to look into plugging phenomena, refine the equation describing the plugging mechanism, and quantify some of the parameters in this equation. Future work will involve research into

the cures of plugging, such as a "roto-rooter" for coal and a jet flushing system. [83, 113]

7. The accumulation of a size degradation data bank. The D.O.E. participates in an international information exchange between the energy agencies of various nations. In the area of degradation research, it acts as the lead agency and collects degradation data from the other agencies involved. This data is then reported on every six months back to the exchange's participants. [83]

8. The evaluation of the enlightened slurry concept. The Colorado School of Mines Research Institute has been contracted to "evaluate the concept of creating froths with certain chemicals to enhance the transport characteristics of coarse coal by reducing water and energy requirements and particle degradation. . . . It consists essentially of adding air and common chemicals to coal, creating a froth and reducing the amount of water and power required for transport." [83, 113]

9. The conception and development of a compact, dry-feed coal injector. Several concepts have been investigated of which two have been chosen for testing:

(a) Foster Miller's Helical Inducer Injector--a full-scale prototype has been fabricated and tested. This unit successfully handled up to 11.2 tpm of 3-inch anthracite at a 56% slurry concentration by weight, yielding a 12 to 18 fps slurry velocity. Further testing continues. [83] A booster pump based on the helical inducer is also being developed.

(b) Ingersoll-Rand Research's Jet Injector--this unit (Figure 1) is based on the same concept as the jet elevators used in the California Gold Rush in the 1850s. A full-scale prototype unit has been designed, fabricated, and successfully tested (as of May, 1982). [83]

10. The research and development of a compact, coarse-particle booster pump. Two routes have been followed thus far: (1) a pump based on Foster Miller's helical inducer concept mentioned above; and (2) a booster system being studied under contract by the University of Minnesota. This latter system involves the removal of water from the slurry line, its pressurization, and reinjection into the line as a high pressure jet. [83]

The Current Coarse-Particle Slurry System

The designer of a coarse-particle slurry system is faced with two major challenges: (1) what is the optimal operating state and what specific parameter values would lead to it; and (2) in the face of many sources of variability, how can the optimal operating state be maintained, or at least approximated?

The realization of an optimal system is now essential as coarse-particle slurry lines are moving into critical materials handling

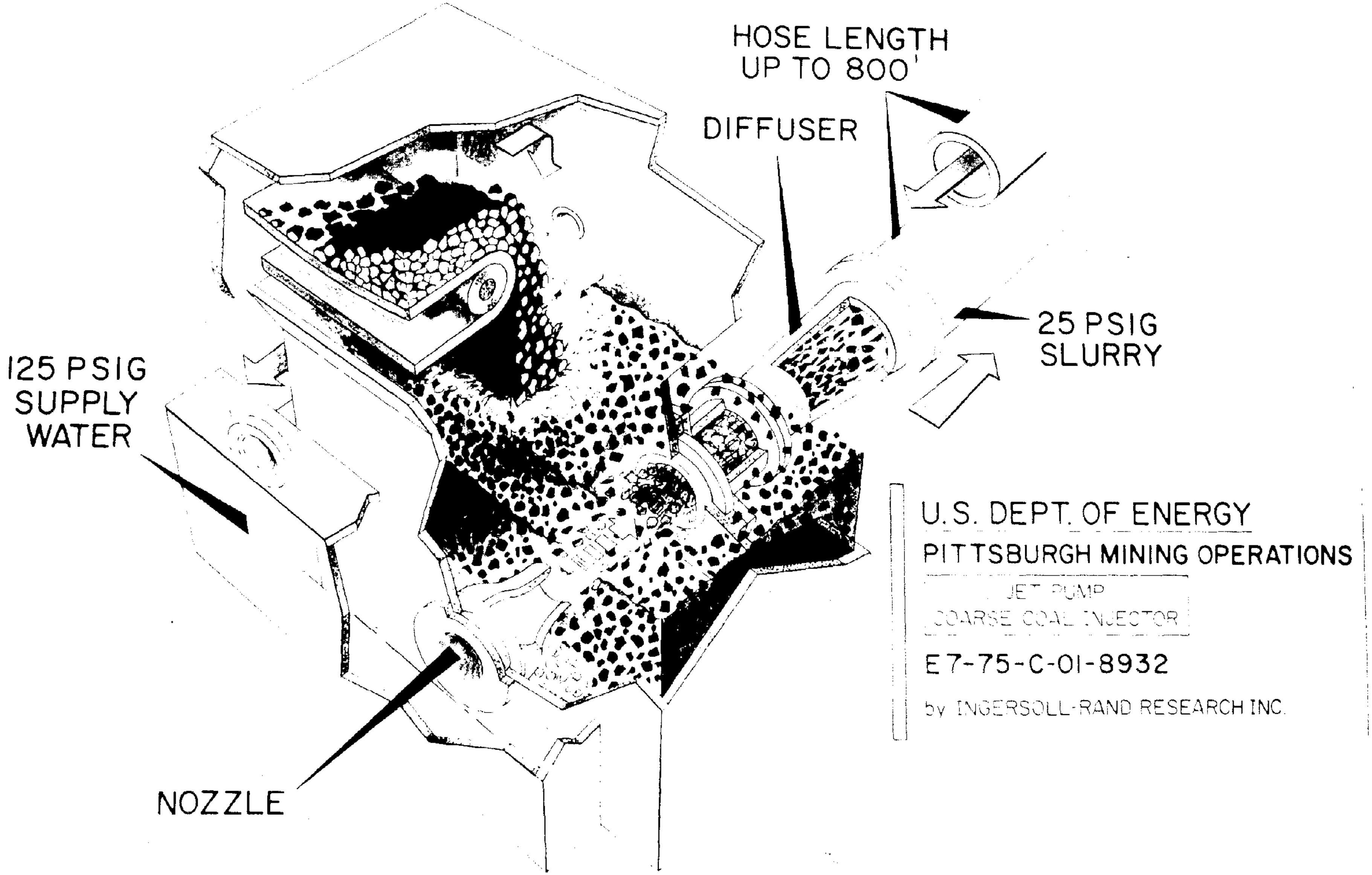


Figure 1. Cut-Away View Showing the Jet Pump Injector Concept (83)

applications because of their safety and high-productivity advantages. No longer is it sufficient to base slurry line design on trial-and-error tinkering: as coarse-coal lines become major components of coal-mining systems, it is necessary to refine their design to attain absolute cost minimization.

Exactly what makes up an optimal system, however, has not yet been defined. Coarse-particle technology has not reached the maturity of fine-particle transport in which most of the basic research has been carried out, making long-distance slurry lines able to be designed out of books alone (the 69-mile Ertsberg copper concentrate line, for example). Coarse-particle technology is in a rather primitive stage of development, and this requires that substantial pilot-project research be executed for a successful, optimal line to be constructed.

Coal-particle degradation and wear studies, for example, are often necessary. The degree of disintegration of a coal particle as it is transported through series of centrifugal pumps, sumps, and lengths of line is important not only because smaller sizes may hurt the marketability of the coal, but also because some idea of the magnitude of fine particle content to expect when designing the capacity of the fine-particle dewatering section of the preparation plant is necessary (Consol, for example, ran a degradation test at its Ponca City loop prior to designing its Robinson Run and Loveridge hydrotransport facilities).

The magnitude of the impact of erosion and corrosion on pump and pipe wear is also essential for design and economic reasons. It is necessary to know how quickly, for example, the pipe is being worn away in order to design a sufficient wear allowance into the thickness of the pipe. Additionally, pipe and pumps make up the bulk of the capital cost in a slurry system, making it necessary to know precisely what their expected lifetimes are in order to do economic feasibility studies. Consol, for example, in designing its Loveridge facility did a series of tests on a variety of pipes which allowed it to choose the best pipe type, select the best chemical erosion inhibitor, and determine how fast the pipeline would be expected to wear away. [27]

The two most crucial factors which have to be predicted before a coarse-coal line can be constructed, however, are friction loss and critical velocity. The critical velocity is that velocity at which particles in the slurry begin to settle on the pipe bottom, and it is necessary to know its magnitude in order to prevent the plugging of the pipe. Headloss is necessary not only in order to determine the pump capacity needed to transport the slurry through the line, but also to ascertain how much power is consumed in doing so (power being the single largest component of operating costs), thus basically being the deciding factor in the feasibility of any given project.

This is when the degree of difference in the development of coarse-particle and fine-particle transport becomes particularly noticeable: fine particle flow theory is relatively well-defined while coarse-particle theory has barely developed beyond the stage reached in the old dredging days. There are couple of explanations for this. First, no reliable scaling law exists for coarse-particle flow models. This means that even elementary research cannot be done on a small scale in laboratories--experimentation has to be done in very expensive, full-scale facilities.

These facilities are well beyond the means of academics in universities, so little advance comes from that sector. [35] The ones that are more able to afford such huge installations, private companies, find that the required investment is so high that it would make little sense to give the acquired knowledge away for free, thus, they keep most experimental results proprietary.

Recently, however, a third sector has emerged which has shown itself willing to finance research and make the results public information. This new participant in coarse-particle flow investigation is government. In the last few years, for example, the U.S. Department of Energy has constructed the Hydraulic Transport Research Facility (see Appendix I), the West German government's mining research entity, Steinkohlenbergbauverein, has built a similar facility, the British government has helped fund another (run by the British Hydromechanics Research Association), and the Canadian government (CANMET) funds part of the operations of the Saskatchewan Research Council's pipeline research facility, which was built by the Saskatchewan provincial government. Substantial results from these facilities, however, have not yet been forthcoming as their construction has been so recent and delays are being experienced.

Besides the problem of the research having to be done at a full scale, the second explanation for the relatively primitive state of coarse-particle flow theory is that it is much more complex than that of fine-particle. The fine particles in a homogenous slurry exhibit little inertial effect and thus approximate rather closely the behavior of water. This statement cannot be made about coarse particles, however; their behavior in a slurry is more intricate. One group of researchers, Lazarus and Neilson, have stated that in order to characterize solid/liquid flow, up to 14 independent variables are needed. [70]

The approaches which have been taken so far in trying to come up with critical velocity headloss formulae can be divided into three groups: (1) the empirical, (2) the semi-theoretical, and (3) the theoretical. The body of purely empirical headloss equations is centered around a single correlation between certain flow parameters and headloss which was first presented by Durand and Condolios before the seminal 1952 hydrotransport conference held by the British National Coal Board. The general form of this equation is:

$$\Phi = K\psi^n$$

where $\Phi = \frac{i_m - i_w}{i_w c_v}$ and $\psi = \frac{v^2\sqrt{c_D}}{gD(s-1)}$;

i_m = slurry headloss in ft. of water/feet of pipe;
i_w = clear water headloss in ft. of water/feet of pipe;
c_v = delivered volumetric concentration (decimal fraction);
v = mixture throughput velocity (feet/second);
c_D = drag coefficient (dimensionless);
g = gravitational acceleration (feet/second^2);
D = inner pipe diameter (feet);
s = solids specific gravity (dimensionless);
K and n are constants.

The original Durand and Condolios equation held K to be either 81 or 180 and n to be -1.5. The subsequent variations of this headloss equation usually differ not in the basic form but rather in the values of K and n. The critical velocity formula is expressed as:

$$V_{critical} = F_L[2gD(s-1)]^{\frac{1}{2}} \text{ (ft/sec)}$$

where F_L is a constant depending on solids concentration and particle size.

The semi-theoretical approach utilizes conceptualization in conjunction with empirical relations. Typical of this genre is Wasp's equation, which, roughly put, makes a distinction between the fraction of the slurry flowing homogenously (no vertical concentration gradient) and the fraction flowing heterogenously (vertical concentration gradient). The headloss incurred by the homogenous fraction is calculated using single-phase Newtonian methods while that incurred by the heterogenous fraction is calculated using the Durand-Condolios correlation mentioned above.

The purely theoretical approach is exemplified in spirit by Wilson's sliding bed model. Basically, Wilson divides flow into two components: suspended load (solids suspended in fluid by fluid's turbulence) and contact load (solids that are in contact with the pipe). He then examines the sources of resistance to movement exerted on each component and from this analysis derives a headloss equation.

The major problem associated with contemporary equations is that they are based primarily on particles which are significantly smaller than those transported under actual mining conditions. The designers of current coarse-particle hydrotransport facilities, as a result, did not get their critical velocity and headloss figures from one of the above-mentioned equations, but rather from full-scale pilot projects which were contructed specifically for this purpose.

Although the recent arrival of experimental facilities constructed specifically to test the flow of coarse particles such as those coming off of a mined face may refine the current equations, it is doubtful that the need for pilot projects will ever be totally eliminated. So many variables are involved that although reasonably accurate formulae may be developed, it is dubious that any theory could ever take them all sufficiently into account. Indeed, the U.S.D.O.E.'s Hydraulic Transport Research Facility states that its interest is to reduce the amount of testing necessary rather than eliminate testing altogether:

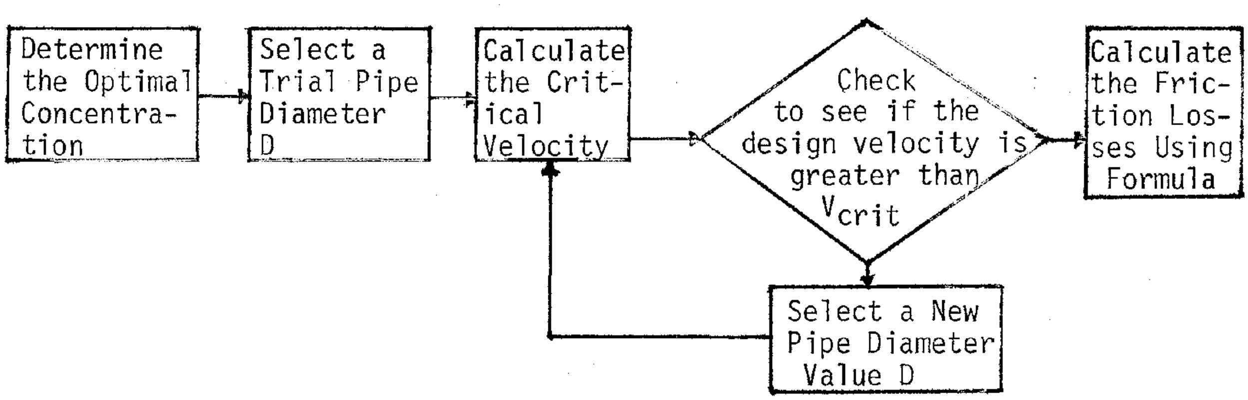
> Since coals differ significantly in their physical properties it is unlikely that one study, however comprehensive, could remove the need for all further testing. Nevertheless, the results of such a test program, available in the public domain, will drastically reduce the amount of testing required for the design of a transport system in a given mine.[35]

The Soviets seem to have come to the same conclusion. In the English translation of <u>Technology of Hydromining and Hydrotransport of Coal</u>, Ofengenden, commenting on coarse-particle slurry experimentation, states:

> Each slurry has its own characteristic properties; therefore, universal equations suitable for all types of slurries cannot be given. The long range hydrotransport department of UkrNIIGidrougal Institute has undertaken a research program encompassing most possible conditions of application of hydrotransport in the coal industry. . . . The method suggested is based on experimental materials produced on full-scale installations, i.e., full-sized pipes and natural coal. . . . Since the method is based on experimental data encompassing the entire range of industrial diameters, consistencies and speeds of movement of slurries and the coal most probably to be used in hydrotransport, it is quite reliable.[86]

No one formula can take into account every single one of the factors known to affect slurry flows. As a result, some pilot testing will be necessary.

Even supposing reliable headloss and critical velocity formulae were available, the design of coarse-coal lines would still be far from optimal. The problem lies in determining the "best" values for slurry flow parameters and pipeline characteristics. Again, fine-particle pipeline designers have a much easier job. This is the procedure they follow:



The first step is to choose the optimal slurry composition and concentration (which will then determine the rest of the unknown parameters). Using a consistometer, the consistency (related to viscosity) of several slurries of varying solids concentration is measured in order to construct a consistency vs. concentration curve. This entire procedure is then repeated for a number of different size distributions (certain size fractions having been added or removed). Using this data, the slurry with the lowest consistency (and the lowest friction loss) over the widest range of concentrations will have the optimal composition and concentration. [126]

Having been given the amount of solids needed to be transported and having determined the optimal concentration, the throughput Q can be calculated. An initial trial pipe diameter is randomly selected and the resulting actual slurry velocity is calculated (V = Q/A). Plugging in the chosen parameter values into the critical velocity equation, the magnitude of the minimum velocity needed to prevent solids settling is determined. If the actual slurry velocity is less than this critical velocity, then a different pipe diameter must be chosen. This procedure is continued until a pipe diameter is found which results in a slurry velocity just above the critical velocity (a slurry velocity far above the critical velocity would result in excessive friction losses and thus increased power costs). Once this optimal pipe diameter is ascertained, all of the parameter values are plugged into the headloss equation and the friction loss per length of pipe is obtained. The fine-particle slurry line is then designed on this optimal basis. [125]

The calculation of the optimal parameter values of a coarse-particle slurry line would follow the same pattern except for one problem: it is not possible to find an explicit optimal value for the particle size and concentration as it was with the fine-particle slurry. Instead, the values of the various parameters are based primarily on experience. The rationale behind choosing specific values for the unknown parameters is given below:

(a) Concentration:

As solids are fed into a liquid, the amount of energy used to transport this slurry increases since the slurry density is increasing. The specific power costs (power/unit weight transported per unit distance), however, decrease. This is exhibited in Figure 2. For a certain interval,

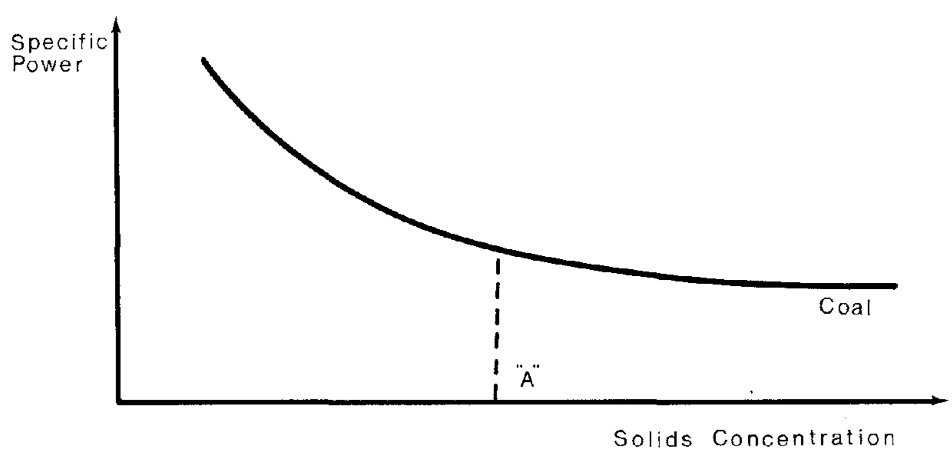


Figure 2. Specific Power and Solids Concentration [3]

as solids concentration increases, specific power decreases dramatically, until a turning point is reached after which only slight decreases in specific power follow increments in concentration. [3] A minimum concentration of "A" is therefore defined, and a rationale is provided for increasing the solids concentration as much as possible in order to capture further decreases in specific power costs, even though they are becoming increasingly small.

A counterbalancing factor, however, exists which defines an upper limit. The absolute maximum volumetric concentration, of course, would be that existing in the situation of maximum packing density (in the case of uniformly-sized particles this would be 74%). [3] A practical maximum volumetric concentration, however, surfaces well before this stage is ever reached. Particles move with the carrying fluid if they have a certain minimum amount of jostling space; when this minimum space limit is violated, rigid structures develop which lead to plugging of the pipeline. The practical maximum concentration, therefore, is that which just allows sufficient space.

Exactly what value this upper limit has is not clear, but Bain and Bonnington suggest that since the incremental advantage of exceeding the "A" concentration level is increasingly small, it is not worth the risk of pipe blockage to go far beyond it. A maximum volumetric concentration level of 40% and an operating level of 30% is advised. [3]

An article by D. L. McCain, however, based on Consolidation Coal Company experimentation, suggests that a volumetric concentration from 40% to 50% may be preferable. In their tests:

> An interesting phenomenon has been observed in pumping coarse coal slurries of concentrations up to 60 volume percent. Pressure losses at a given velocity in horizontal lines are essentially linear with respect to concentration up to about 35% concentration. . . . Losses drop off over the 35 to 45% range and then climb again at a rate generally lower than the climb for low concentrations.[76] [Figure 3]

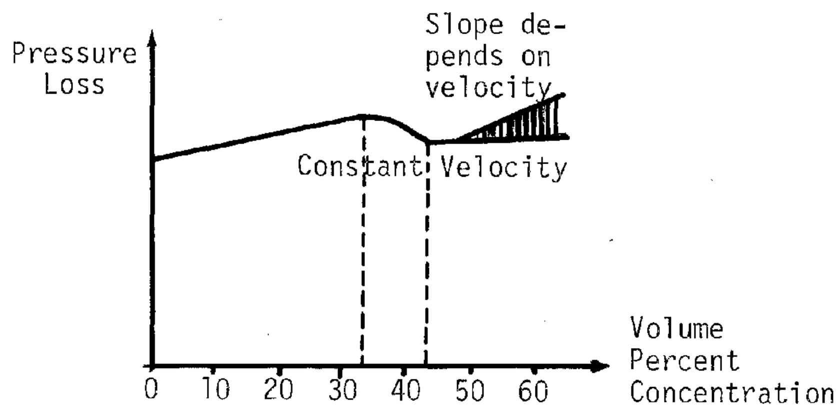


Figure 3. Concentration Effect on Pressure Loss at Constant Velocity.[75]

In a compromise between the desire to have low specific power costs and the wish to avoid plugging, most current coarse-particle installations have taken on a value of 30% for volumetric concentration (see Appendix I).

Other issues having an impact on the concentration decision may be concentration's effect on the wear rate (increased C_V means more particles hitting the pipe surface but also increased interference with one another, lowering the contact velocity) and its effect on pump performance (increased C_V decreases the head supplied, increases the power consumed, and decreases the pump's efficiency). [3]

(b) Particle Size:

A conflict between two goals underlies all slurry particle-size decisions whether a fine-particle or a coarse-particle facility is being designed: on the one hand, the designer wishes to reduce the friction losses by making the particle small (since less velocity is needed to support a small particle and thus less friction loss), but on the other hand, he wishes to avoid high preparation and dewatering costs by making the particle large. In the case of a slurry line in an underground mine, the transport distances are so small that the high friction costs resulting from transporting a coarse particle are outweighed by the savings from not having to extensively crush and dewater the coal; thus, a coarse-particle size is chosen.

This choice is reinforced by the fact that the marketing of the coal demands a particle as large as possible. One major source of criticism for the coarse-particle hydrotransport system has been that sending the coal through series of pumps and pipes will degrade the size of the coal to such an extent that it will be less marketable. This statement has varying degrees of validity depending on the ultimate use of the coal. In the case of virtually all electric utilities it is false since all coal is pulverized before use. In the case of coking plants and industrial stokers, however, it is to some degree true since although these facilities do not need very large sizes (virtually all stokers use -2" particles and coking plants usually pulverize their coal to 90% -1/8") they need specific particle size distributions. In order that the preparation plants have as much flexibility as possible in creating particular size distributions for both cleaning and marketing reasons, the size of the coal being run through the slurry system should be as large as possible, subject to certain limitations.

The major constraint on the size of the particles run through a pipeline is called the "1/3" or "bridging" rule. Established through experience, this rule says that the top size shall be one third the diameter of the pipe, otherwise serious risks of bridging (plugging) will arise. This requirement can be fulfilled by one of two methods: either by increasing the diameter of the pipe so that it is at least three times the size of the largest particle mined, or else by crushing the coal to a size which would be consistent with an already-determined, optimal pipe size.

This first alternative is defective in that it is hard to reconcile with the need to establish optimal flow. As indicated earlier in the design flow chart, varying the pipe diameter is a way of establishing an optimal slurry velocity just above the critical velocity. If the "bridging" rule is applied by increasing the pipe diameter instead of reducing the particle size, then this process of flow optimization through pipe diameter control is disrupted and non-optimal flow may result.

The second alternative (reducing the particle size instead of increasing the pipe diameter), however, allows both rules, flow optimization and bridging, to be respected. This is why all current coarse-particle slurry systems have some sort of size control unit (crusher) at the injection site.

The most frequently-used method for determining the extent of particle size reduction is to make use of information on the size distribution of the coal coming from the miner. For example, if the bulk of the coal is primarily of one size but with a small amount of larger sizes, then it would be a good idea to crush the larger sizes (Figure 4) while leaving the greater mass of the material alone.

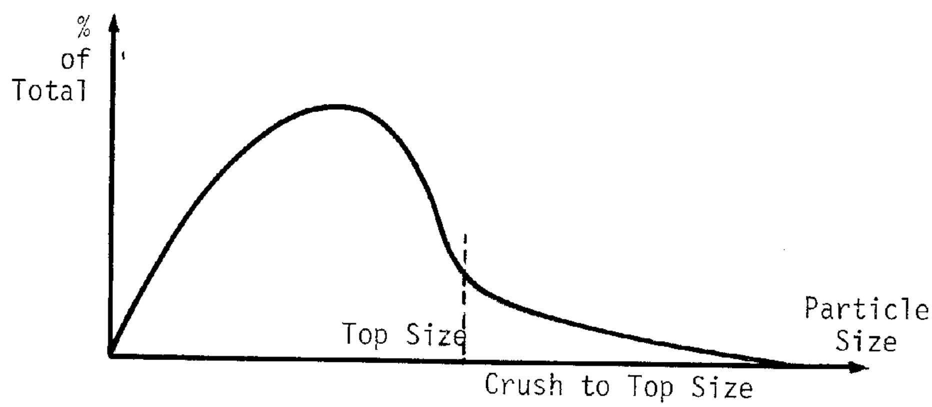


Figure 4. A Typical Particle Size Distribution

Two physical limitations must be kept in mind, however, one dealing with the size range of material to be crushed and the other dealing with the amount of material to be crushed. A wide range of particle sizes, e.g., from 24" to ½", should not be crushed unless absolutely necessary because doing so with a single machine is very inefficient. The more efficient use of two crushers, however, is impossible due to space limitations at the face.

The second constraint is that since space is so limited, only crushers below a certain size can be utilized. This means that the top particle size should not be reduced to the extent that the capacity of the largest possible crusher is outstripped. Consol had this in mind when it chose its top size as 4"--only 15% of the raw coal coming from the continuous miner was larger than 4". [26] Link, in his study for the U.S.B.M., followed the same logic; he selected a top size of 2" because the amount of material larger than this which would have to be crushed made up only twenty percent of the total. [67]

Other factors may also be involved in selecting the value for the particle-size parameter. Abrasion, for example, increases with slurry particle size, so the top size may be reduced if pump and pipe wear is a serious problem. [3] Additionally, if a hydrohoist is used in conjuction with the horizontal coarse-particle line, it may be desirable to limit the top size so that an extreme slurry flow velocity (and thus high friction loss) is not needed to counteract the largest of particles' rapid settling velocity. [3] The Soviets' top particle size is determined by what will pass through the equipment they plan on using. [89] This constraint is valid in that it is possible to make the channels for slurry flow in pumps only so large before efficiency losses become severe.

(c) Pipe Diameter

The optimal pipe diameter is a result of concentration and particle size choices. As described earlier, a number of pipe diameters are considered until one is found which finally results in a slurry velocity just above the critical one. This diameter would then be chosen as optimal for the system. In some situations, however, the pipe diameter value is determined exogenously. Link, for example, in his coarse-coal hydrotransport feasibility study for the U.S.B.M., did not follow the slurry velocity optimization process, but rather found the largest pump he could fit into a 5.5' high entry and chose the pipe size (10") which was compatible with this pump. [67]

Pipe diameter selection for a hydrohoist is an entirely different story from that for horizontal coarse-particle hydrotransport. Vertical flow theory is based simply on the settling velocity of solid particles in the fluid, while horizontal theory is based on the much more complex notion of particles being suspended by the turbulence of a high-velocity flow. While the diameter of a horizontal line is chosen to induce this high velocity, the velocity in a vertical line need not be so great, allowing the pipe diameter to be very large and the friction losses small. The diameter can be so large and yet still induce the minimum flow velocity that eventually the added capital cost of the next larger size of pipe becomes more significant than the benefit received by the resulting lower friction losses caused by a lower slurry velocity. In vertical transport, therefore, the pipe diameter is limited by a capital cost constraint while in horizontal transport it is limited by an operating cost constraint. [3]

(d) Slurry Velocity

To ascertain the operating velocity of the slurry line, the critical velocity (that speed at which solid particles begin to settle to the pipe bottom) must be determined either by formula (unreliable) or by pilot project. The closer to operating at this critical velocity the slurry comes, the lower the friction losses; but, this comes at the cost of increased risk of plugging as a result of some accidental velocity drop. Depending on how much of a risk the operator is willing to take, the slurry is pumped at a rate separated from the critical velocity by some buffer zone. Other issues in choosing the slurry velocity are pump operating costs, coal degradation, and pipe and pump wear.

It was mentioned previously that the designer of a coarse-particle slurry system is faced with two major challenges, the first of which (the selection of an optimal operating state and its parameter values) has been covered above. While this optimal steady state may have been established in theory, in practice it is virtually never obtainable: in the context of underground coal mining so many sources of variability exist that no steady state, let alone an optimal one, may be maintained. The second part of the design problem, therefore, is to create a system which will control the everpresent variation and thus approximate the optimal operating state as closely as possible.

This approximation is realized in practice by fulfilling three "operation objectives": (1) the prevention of plugging, (2) the minimization of specific power costs, and (3) the avoidance of certain operational catastrophes such as sump overflow, cavitation, or water hammer. The following portion of this report will delineate the basic layout of an underground coarse-coal slurry system, point out for each of these "operation objectives" areas in the system where sources of variability threaten their fulfillment, and describe how design embellishments have eliminated these threats, thereby allowing an approximation to the optimal steady state.

The modern underground coarse-coal slurry system can be broken up into three major components: (1) face haulage, (2) sub-main haulage, and (3) main haulage. The essential functions of the face haulage unit are that it must accept raw coal from a mining system, prepare a slurry, and send this slurry to the next haulage unit. To do this, coal is fed into a feeder-breaker which mixes the coal with water, crushes it, and pumps the resulting slurry through a pipeline to the sub-main, aided in some cases by a booster pump en route. The feeder-breaker may be mobile or immobile depending on the mining method utilized. With mechanized room-and-pillar mining, for example, the feeder-breaker is put on treads and pulls a flexible rubber hose, which enables it to follow the movements of the continuous miner for up to 1000 feet (see Robinson Run Mine in Appendix I). With longwalling, on the other hand, the only system currently in use in the U.S. has an immobile feeder-breaker which accepts coal from a conveyor belt linked to the longwall (see Loveridge Mine in Appendix I).

The essential function of sub-main haulage is that it must accept slurry from a number of the face haulage units mentioned above and transport it to the main line. This is made possible through the use of sumps which accept the flows from the face units and the immediately upstream section of the sub-main and feed the mixture into pumps which send it further downstream in the sub-main line until, eventually, the main haulage system is reached.

The main line's function is to take the coarse-coal slurry from the sub-main and transport it through the use of booster pumps over relatively long distances to a shaft-bottom sump. The slurry from the sump is then injected into a hydrohoist system, usually either a series of centrifugal pumps or a pipefeeder, and transported either directly to a preparation plant or to a surface storage tank from which it is pumped to the preparation plant by centrifugal pumps.

The slurry system just described is merely the skeletal structure of the modern one: to its form have to be added several modifications which allow the system to deal with the several sources of variability which discourage the fulfillment of the three "operation ojectives" and thus the approximation of the optimal steady state. These sources, how they threaten the system, and how the system counteracts these threats are described below according to the objective affected.

Operation Objective #1: Prevention of Plugging

Plugging (the blockage of pipeline throughput by a mass of stationary material) basically occurs for one of two reasons: (1) violation of the previously mentioned 1/3 rule which may result in an impassible bridge across the pipe width; or (2) violation of the critical speed rule which may result in solids settling to pipe bottom to form an immobile mass. The source of the "1/3 rule" problem lies in the feeder-breaker: these units virtually always utilize roller crushers to reduce the coal fed from the mining system to a desired top size. The design of roller crushers, however, allows the occasional injection of oversize prismatic shapes into the system which causes bridging risks.

Additionally, the feeder-breaker cannot screen out the abundance of tramp material, often oversize, which makes its way into the system--Consol's Loveridge slurry installation has found that the entrance of debris has been the major source of plugging in its experience. In one specific case, for example, a plug at a pump station was caused by the injection of four large pieces of tramp iron, one of which was 18"x10"x6". [77]

The number of plugs caused by violation of the 1/3 rule, however, is insignificant in comparison to that threatened by the lack of maintenance of the critical velocity. The sources of variation causing this insufficiency of speed (and thus insufficiency of turbulence needed to keep the slurry's solid component from settling) are numerous and can be classified into two broad groups: (1) those that occur only once in a great while and over which one either has no control or has no wish to control due to overbearing economic reasons, and (2) those that are continuously present and thus have to be continuously guarded against through design modifications in the slurry system.

In the first category two particular instances stand out: plugs induced by power failure and those induced by leaks in the slurry line. Power failure plugs have been experienced by Consol's Loveridge mine, for example, during summer electric storms. The lightning knocks out the mine's power supply and thus its pumping system. Without pumps, the slurry velocity rapidly falls below the critical level and plugs are formed as the solids settle out. [77]

Little can be done to avoid the effect of these unpredictable storms unless an auxiliary power source is developed to replace the knocked-out system; this, however, would probably not be economically justifiable considering the infrequency of the power failure. Instead, Consol either shuts the slurry system down during electric storms and utilizes the time for maintenance or else keeps the concentration low (which would allow the plug to be removed more easily if a power outage did indeed occur). [77]

The occurrence of plugging through leakage in the line is reported by the American Gilsonite fine-particle line. In this case, the loss of fluid through a leak reduces the throughput in the line; increases the flow cross-section, and causes a severe pressure loss, resulting in a loss of slurry velocity and the settlement of solids.

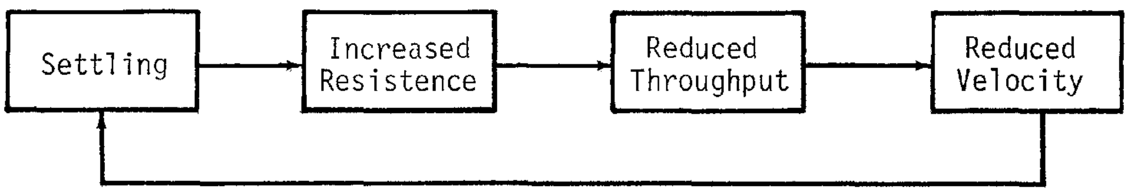
The first category of critical velocity-related plugs can be characterized by its sporadic occurrence and by the fact that deposition results from throughput loss caused by some factor outside of the normally-operating system (power failure and pipe leakage, for example). Plugs in the second category, on the other hand, have sources which constantly, rather than occasionally, threaten the viability of the system. This type of pluggage is brought about by one or both of two phenomena. The first is similar to that causing the category-one plug (reduction in throughput results in the fall of actual slurry velocity below critical velocity), but the throughput reduction comes not as a result of some exogenous condition, but rather from increased flow resistance generated by endogenous sources of variability (changes in flow parameters).

The second phenomenon also has its root in those interior variability sources--while the sources cause settling of solids through a reduction in the actual slurry velocity, they may simultaneously reinforce this deposition by increasing the critical velocity necessary to suspend the solids, thus widening even further the dangerous gap between the actual and critical velocities.

The variable "sources" causing this second, more economically significant category of plugging are the random variations in the values of the very flow parameters which determine the critical velocity and friction loss of the flow.

The mechanism by which these parameter value variations lead to pluggage in a coarse-coal line is the following: due to some change in the characteristics of the slurry being fed into the line (an increase in specific gravity, for example), resistance to flow is increased which, if a fixed speed centrifugal pump is being used, results in reduced throughput and an actual velocity which may decline below critical velocity, causing the settlement of particles.

Once this initial deposition occurs, the system moves into a cycle of further settling which ends ultimately in pluggage.



The initial deposition of solid particles induced by the increased specific gravity of the slurry increases the frictional resistance to flow by reducing the cross-sectional area open to flow. This increased resistance results in a reduced throughput since centrifugal-type pumps are used (which is generally the case, see Figure 5) causing a reduction in the slurry velocity which leads to further settling, which leads to further resistance, etc.

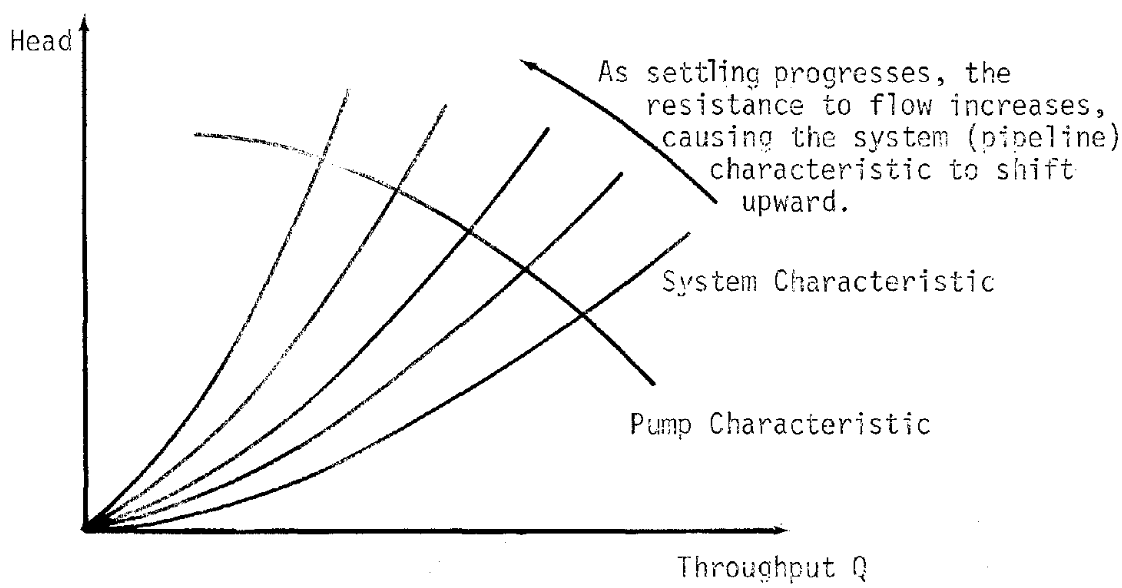


Figure 5. A Typical Pump Characteristic Curve

As described earlier, with the second category of pluggage there may be another force inducing the deposition of solids. The change in the characteristics of the slurry being fed into the line may make necessary a higher critical velocity whose magnitude must be reached to keep particles from settling out. A system running on a fixed-speed centrifugal pump has no way of detecting this need for a changed velocity, and even if it could, it would not be able to react by changing its pump speed. As a result, deposition would occur which would then initiate the previously-mentioned cycle of further settlement.

The phenomenon which has the potential to break this cycle is the fact that as settling occurs, the cross-sectional area available for flow decreases which induces, according to $V = Q/A$, an increase in velocity through the constricted area. This velocity may resuspend the settled particles if it satisfies their critical velocity requirement, but if the variation in the parameter value is so extreme that the critical velocity is not immediately reached, then resuspension will not occur. This is due to the fact that while the velocity is being increased by a reduced cross-sectional area, this very same reduced cross-sectional area increases the frictional resistance to flow (particularly since the deposited bed is rough), reducing the velocity not only in the restricted area, but throughout the length of the pipe. Additionally, the deposition of particles may reduce the pipe area open to flow to such an extent that suddenly the 1/3 rule may come into play and pluggage may occur not through further settling, but rather through bridging.

The crucial issue, however, is the identification of the significant parameters whose variation has such an impact on critical velocity and frictional headloss as to overwhelm the natural self-correcting mechanism described in the paragraph above.* These parameters are revealed by inspection of Durand's headloss and critical velocity equations.

Headloss:

$$\frac{i_m - i_w}{i_w c_v} = 81 \left[\frac{v^2\sqrt{c_D}}{gD(s-1)}\right]^{-1.5}$$

Critical Velocity: $V_{crit} = F_L[2gD(S-1)]^{\frac{1}{2}}$

where c_D = coefficient of drag = $4gd(s-1)/(3w^2)$;
d = particle size;
P_s = solids density; and where
w = settling velocity = $k\sqrt{(P_s-1)d}$;
k = constant depending on material.

*A major objective of the U.S.D.O.E.'s Hydraulic Transport Research Facility is to determine the extent of variation allowable before deposition will begin. [35]

Since g and D are fixed, k is a constant depending on the material, V is an independent issue, and F_L is a constant whose value depends on concentration and particle size, the critical velocity and frictional headloss are to a large degree dependent on three variables: particle size, concentration, and specific gravity. Below, each of these parameters will be examined to see why its variability in value can initiate plugging and where this variability comes from. The design solution to the problem posed by the variation in these parameters ($V_{act} < V_{crit}$) will then be described.

1. Concentration

In the case of the concentration flow parameter, both the headloss and critical velocity mechanisms of pluggage are at work, although the V_{crit} mechanism works in a manner opposite from that expected: increased concentration apparently does not increase the magnitude of the critical velocity, but rather decreases it due to a sort of hindered settling phenomenon which reduces the particles' settling velocity.

The friction headloss mechanism, however, works in the opposite direction. A higher concentration increases the specific gravity of the slurry which increases the resistance to flow. This increased resistance results in a lower throughput and thus a lower slurry velocity. If the slurry concentration swings high enough, then the velocity will be reduced to sub-critical and deposition will occur initiating the further settling cycle previously described.

A major source of this variation from the optimal concentration level lies in the slurry mixing unit. This unit supposedly mixes water and solids to form a slurry having the design concentration, but error can be large and concentration sensors warning of such errors are imprecise and have delayed reactions, all of which results in a highly variable concentration level. An additional source of concentration variability involves the particle-size distribution and is described below.

2. Particle-Size Distribution

In the case of the particle size parameter, the critical velocity mechanism of pluggage is not applicable. The only particle size which is important as far as critical velocity is concerned is the top size since this top size would be the first to settle as actual slurry velocity decreases. This top size, however, does not take on the random, variable values which would disrupt the system by changing the critical velocity; rather, it is determined by the designer and maintained through the use of a crusher in the feeder-breaker.

Other than this crushing to a top size, however, there is no control over the size distribution of the coal fed into the system--coal is injected into the coarse-particle slurry line in essentially the same condition it is in when ejected form the mining system. The resulting variability in the size distribution can disrupt the system through the headloss mechanism (initiates plugging through increased resistance to flow).

Bain and Bonnington say this disruption "arises from the nature of solids-fed systems, in which solids flow to the pump intake usually is under the action of gravity. Any fortuitous change in, for example, the solids size distribution may influence the internal shear-resistance within the hopper, varying the feed concentration." [3] As seen in the concentration discussion above, increasing the feed concentration leads to increased resistance to flow, reduced throughput, and thus reduced actual slurry velocity-possibly below the critical level. Consol's Loveridge installation is experiencing this type of problem in its Multiple Feed Sump. Concentration control is made difficult because: "there is some segregation occurring in the sump, as the coal and rock are discharged and settle. The large rocks tend to sink to the bottom of the sump under the input gates while the finer coal sizes tend to wash to areas away from the input gates." [77]

3. Specific Gravity

In the case of the specific gravity parameter, both plugging mechanisms are active: as the specific gravity of the solids increases, the critical velocity needed to prevent deposition shifts from its original level to a higher one. At the same time, however, the actual velocity of the slurry is being reduced because of the headloss plugging mechanism: the increased solids specific gravity increases the resistance to flow, causing a reduced throughput and a reduced actual slurry velocity. Both of these mechanisms reinforce one another in initiating solids settling.

Control over the specific gravity of the solids particles in the slurry is rather difficult. Coal in itself is a rather heterogenous substance. Coal Preparation (1979 ed.) states that the specific gravity of coal taken from different layers in a single seam can vary as much as .30 of a specific gravity point.

The variation in the slurry specific gravity, however, comes not only from that of the coal itself but also from the varying amount of rock in the raw coal. This rock is entrained into the system as the mining machine sumps into the coal seam and the surrounding strata, and its quantity is very difficult to predict. Consol reports that plugging has been experienced as a result of high concentrations of slate entering into the system. [77]

4. Saltation

Bain and Bonnington suggest that another source of pluggage initiation in the same vein as those mentioned above is the very nature of the flow regime utilized in coarse-coal slurry lines: "The inherent nature of flow of large particles by saltation must be expected to cause a temporary increase in the depth of the solids layer, with a corresponding increase in resistance." [3]

As seen before, this increase in resistance may lead to reduced throughput and thus reduced actual slurry velocity, possibly inducing solids deposition.

Examination of these four sources of variability (concentration, size distribution, composition, and saltation) which set off plugging processes reveals a common characteristic which is the key to the solution of the problem they present: all four create gaps between the actual slurry velocity and the critical velocity either by inducing increased resistance to flow resulting in a lowered actual velocity or by increasing the necessary critical speed above its previous level.

The solution to the problem, therefore, is simply to close the gap between the actual and critical velocities by gaining control of the actual slurry velocity and matching it to the critical velocity. To control this velocity it is necessary, in terms of characteristic curves, to control either the system characteristic curve or the pump characteristic curve, or both.

In this respect, a slurry line utilizing fixed-speed centrifugal pumps is obviously deficient because it can manipulate neither of the two curves. Consider, for example, the sudden entry of highly-concentrated slurry into the system (see Figure 6). This influx would increase the

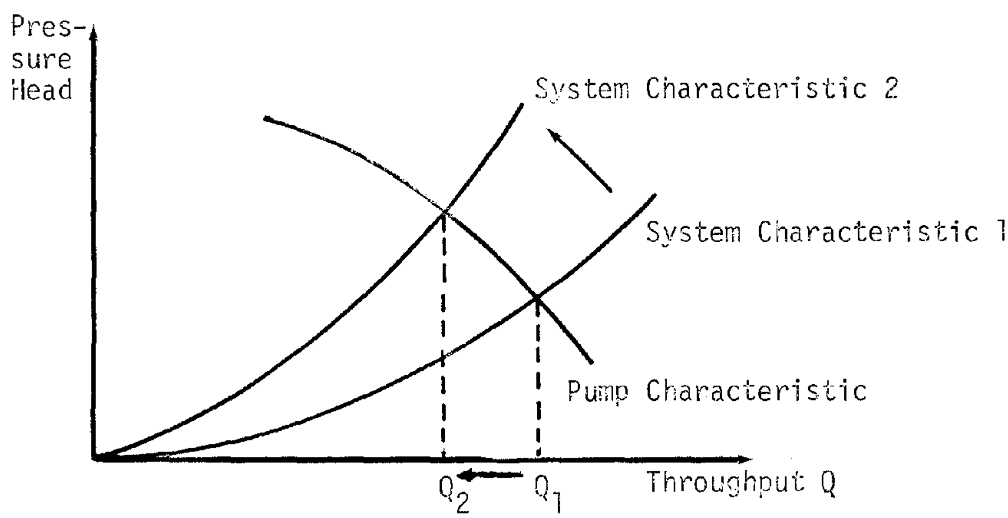


Figure 6. A Typical Fixed-Speed Pump Characteristic Curve.

resistance to flow and therefore shift the system characteristic upwards. Having no ability to reverse this shift or counteract it with a shift in the pump curve, the throughput Q and the slurry velocity fall, perhaps to a sub-critical plugging level.

Slurry systems exist, however, that do have this counteracting ability. The first such system, proposed by Link in his feasibility study for the U.S.B.M. [67], controls the characteristic curves (and thus determines the actual slurry velocity) through the use of variable-speed centrifugal pumps and slurry dilution: the pumps shift the pump curve by varying the rotational speed of their impellers, while dilution shifts the system curve by varying the flow characteristics of the slurry (a slug of high-concentration slurry, for example, would have its dangerously high specific gravity reduced through mixture with additional water).

This method of dealing with critical-velocity-related plugs is put into practice through the use of sumps and variable-speed centrifugal pumps in conjunction with sophisticated electronic sensors and controls. Combinations of these units are located at the face (the feeder-breaker), at all slurry line intersections, and at all booster pump stations. The sensors (velocity, concentration, pressure) are monitored by a computer which, if it perceives a plug forming, sends counteracting instructions to the appropriate sump-pump unit. Either or both of two solutions are taken: (1) dilution occurs in the sump (which receives all upstream slurry flow) through the injection of extra water from a pure water system parallelling the slurry line, or (2) the variable-speed drive is accelerated, enabling the centrifugal pump to exert more pressure against the slurry in the line and thus maintain a super-critical velocity.

Two major problems exist with this type of solution, however. First, variable-speed drives are bulky and expensive. Second, dilution has deleterious effects on the specific power costs of the slurry. In order that the solids be transported as cheaply as possible, concentration must be kept as close to the original injection value as possible. Any subsequent addition of water, such as that used in dilution, is irreversible in its effects: it increases the specific power costs of transport and it uses up pipe capacity further downstream which could have transported high concentration slurry from other sources.

Another system which resolves the actual-versus-critical-velocity gap problem and yet is not subject to the deficiencies of the system above is in the process of being developed by Foster Miller Associates under U.S. Department of Energy contract. This concept, the Helical Inducer, also makes use of the centrifugal phenomena except in an unorthodox manner: simultaneous centrifugal and axial flow are induced by a single-blade, helical rotor (see Figure 7). In the injector based on this idea, a rotary screw feeds dry coal into the grasp of the helical rotor which flings the particles and water (fed into the system at a constant rate at the rotor's leading edge) back into a vortex made up of a cylindrical annulus encircling an interior core of air. The centrifugal force acting on this annulus forces the slurry into the pipeline.

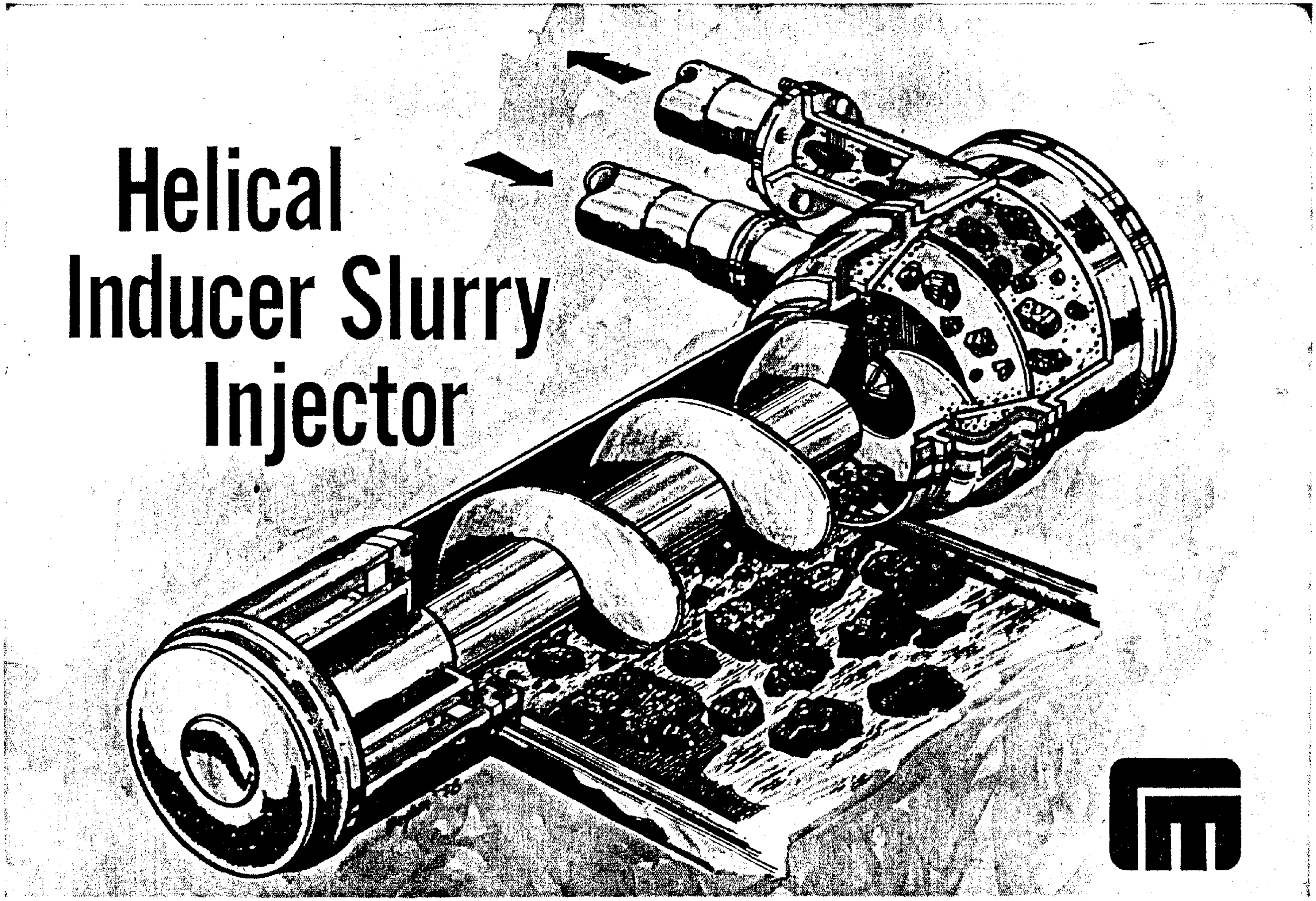


Figure 7

Mobile Slurry Injector

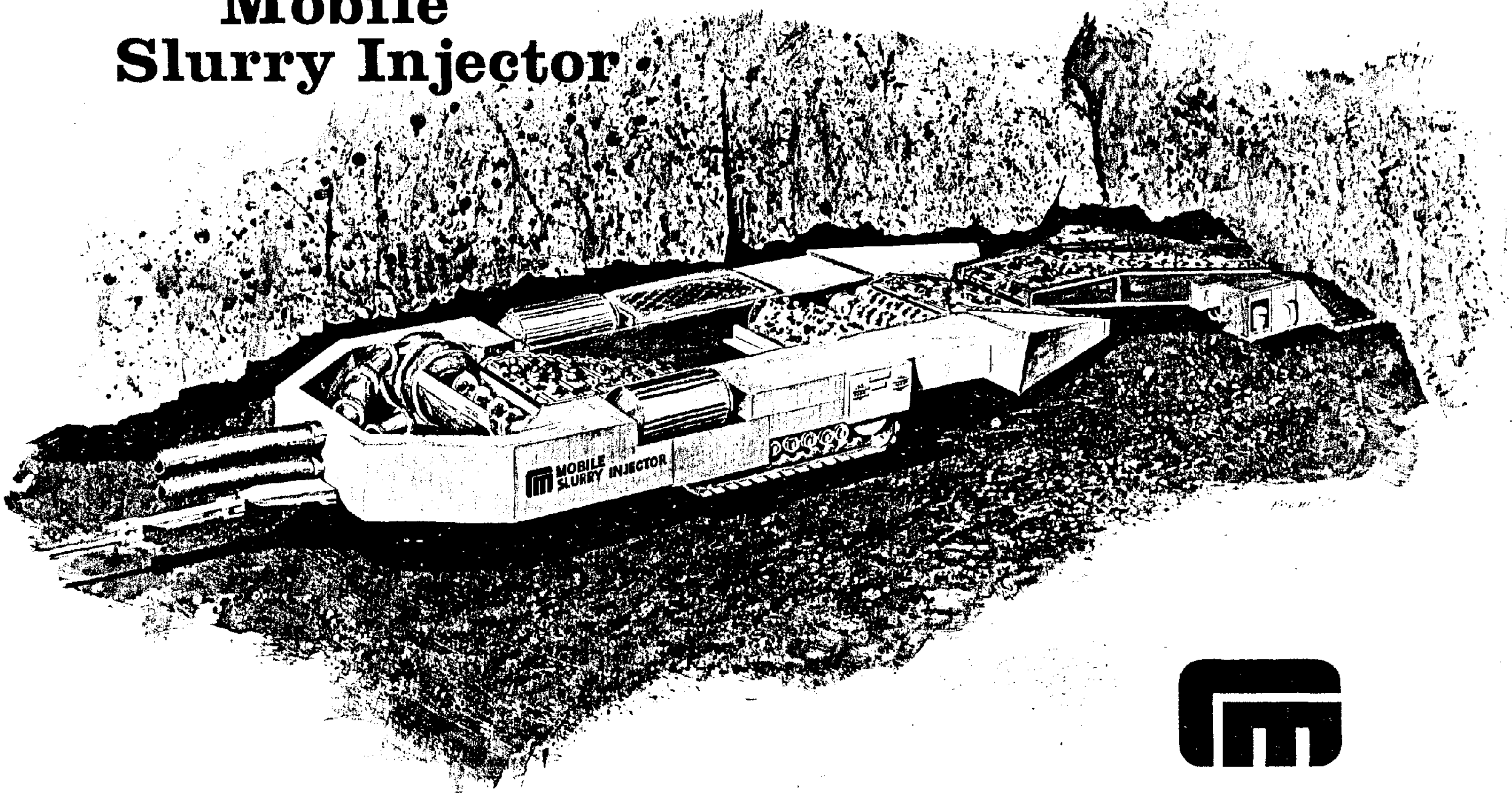


Figure 8

Just as the variable-speed centrifugal pump controlled resistance variations by varying the impeller speed, the stabilizing mechanism in the helical inducer is the above-mentioned annulus. According to Burnett:

> As a higher discharge pressure is required, due to increased flow or solids concentration, the inner radius of the air vortex shrinks. As discharge pressure requirements decrease the air core expands. For a constant speed the helical inducer pump's discharge pressure will primarily vary with the product of the slurry's density and the difference between the square of the revolving liquid ring's outer radius (radius of cylindrical housing) and the square of the inside radius to the liquid ring's free surface. The maximum discharge pressure occurs when the air core of the vortex has contracted to the diameter of the atmospheric solids inlet. Thus, the helical inducer has two operational limits--one due to inlet flooding caused by high back pressure and one due to outlet aeration at low back pressure. [11]

This self-regulating nature of the helical inducer is what puts it at great advantage over the other system. With Link's system the chain of action between a problem and its resolution is lengthy: settlement occurs, sensors pick up a change in flow parameters and send signals to the central computer, the computer makes a reaction decision and sends instructions to the sump-pump unit, which then acts upon them to resolve the settling. With the helical inducer system the chain is much shorter: the problem induces its own solution directly. Settling caused by a high-concentration slug, for example, will immediately cause a thickening of the inducer's slurry annulus. A thickened annulus increases the discharge pressure, which clears away the high-concentration settling by increasing throughput. The helical inducer system, therefore, virtually eliminates the need for an extensive array of sensors, a complex, expensive, potentially-unreliable computer, and an expensive, bulky variable-speed drive unit (the helical inducer rotor turns at a constant speed).

The second great advantage which the helical inducer has over the other system is that it does not dilute the slurry, and since it does not do this, it eliminates the need for sumps at the face and at booster pump stations (sumps to join flows at intersections are still necessary as well as the electronic control system which goes with them). With the injector unit, for example, no mixing sump is needed since the coal is fed in dry by a rotary screw, and water is added only at a constant rate while only the solids' rate of entrance is allowed to vary. [10, 11, 13]

The previous set of critical-velocity-related plug producers have held in common the random, accidental nature of their occurrence. The final type of situation where critical velocity rules may be violated and a plug formed is one where the operator consciously shuts the system down. Coal mining is something of an intermittent process. A continuous miner, for example, not only faces occasional breakdowns, but it also has to stop production periodically to tram to a new mining location.

In such cases the pumps in slurry line cannot be merely turned off because settling would occur immediately and the line would then be difficult to clear for later use. The design solution to this problem has been to flush the coal out of any line about to be shut down for any reason before such termination occurs. Once the coal is flushed out, the pumps are shut off, and no coal is present to plug the line. This solution necessitates a clear water system leading to each sump to supply the needed flushing fluid.

Operation Objective #2: Minimization of Specific Power Costs

Previously it was stated that to approximate the optimal slurry steady state, the operator must fulfill three "operation objectives": (1) prevention of plugging; (2) minimization of specific power costs (power cost of transporting a unit weight of material a certain unit distance); and (3) avoidance of certain operational catastrophes. Having mentioned plugging above, attention must now be focussed on power costs.

The stop-go nature of coal mining mentioned above has a big impact on specific power cost minimization, particularly in multiple face mines. Sub-main and main slurry haulage systems in such mines face highly variable throughput quantities as various combinations of mining units come on and go off line. Typically, a pipeline is designed with a single throughput in mind; therefore, the design challenge in this case is to create a pipeline system which will handle a wide range of throughputs in a manner which will minimize specific power cost.

In other words, the essential problem is that if the pipeline is designed to transport the peak output, say the output of ten faces in a ten-face mine, then when only two faces of the ten are operating, as will frequently be the case, the design is no longer optimal--it is necessary to add massive amounts of water to the line to maintain a sufficient throughput to keep super-critical velocity. This means heavy dilution and large increases in the specific power costs since a large amount of water is being transported for non-productive reasons.

The best solution to date has been proposed by Szekely and Kurz. [111] In this system several parallel pipes are utilized rather than a single large one. Each time a change in output occurs as a result of the entry or exit of a face from production, a computer determines the best possible combination of pipes to use which will minimize dilution (thus minimizing the specific power costs) and yet still maintain a super-critical velocity. Pipes which are shut down as a result of this readjustment process are automatically flushed out to prevent settling.

At shaft-bottom the specific power costs are reduced even further by transforming some of the slurry characteristics before injection into the hydrohoist unit. This is performed through the use of a slurry storage tank. First, any slurry dilution which has occurred previously in the slurry system can be eliminated. The particles in the slurry fed into the storage sump are allowed to settle and are then reconstituted by a dredging unit into a slurry of a higher concentration. This concentration must be as high as technically possible because high concentrations radically lower the settling velocity of the particles being pumped vertically: the greater the solids concentration, the slower the settling velocity (since it is hindered by the presence of other particles) which means a smaller actual vertical slurry velocity and lower pipe friction losses. [20, 21]

At the Hansa Hydromine (see Appendix I), for example, slurry is received at the shaft-bottom sump at concentrations varying from 17% to 50% solids. The slurry subsequently injected from the sump to the hydrolichoist however, has a steady concentration of 50% by volume. [38]

A second function performed by this shaft-bottom sump which lowers specific power costs is to serve as a sort of slurry surge tank. Pumping slurry directly from the main line into the hydrohoist can be rather costly in terms of friction loss and plugging risk. The hydrohoist is designed for one specific throughput, but if the <u>Control and Automation</u> system of parallel pipes is used, then a variety of throughputs must be hydrohoisted. This leads to plugging problems in the case where the throughput supplied is lower than the design throughput and friction loss problems when the throughput supplied is greater than that designed. If a surge tank is placed at shaft bottom, however, these surges can be evened out over time and a steady flow can be injected into a hydrohoist system which has been designed specifically for this throughput.

Operation Objective #3: Avoidance of Operational Catastrophes

Several serious disruptions (cavitation, water hammer, sump overflow) of the normal operation of a coarse-coal line have their root in a single common source: pressure transients. These transients generally are a result of inhomogenous slurries in the pipeline and varying rates of flow in the legs of the slurry system.

What is meant by an inhomogenous slurry is that which exists when two slurries of differing characteristics (concentration and specific gravity, for example) are sequentially injected into the pipeline. According to Burnett et al.:

> Cavitation and water hammer can occur when the system contains some sections with high concentrations of solids and others with little or no solids. These sections of fluid tend to move at different velocities due to inertial effects causing separation of the column (slack flow in horizontal pipes) and potential pump failure.[11]

Such an occurrence would be commonplace in the day-to-day operation of an underground coarse-particle slurry line. Consider, for example, the previously mentioned "on-off" nature of coal mining systems and the resulting need for flushing the lines with clear water to prevent pluggage. In such an environment, situations where pipelines are partly filled with slurry and partly filled with water (either being injected or ejected) would be in common occurrence; and, thus, cavitation and water hammer would be a constant threat.

The second source of pressure transients is the varying rates of flow between the components of the slurry system. When slurry is being transported long distances, several centrifugal pumps, either in series or located at intervals along the route, are generally necessary. The rotational speeds of these pumps must be adjusted perfectly so that all will pump the same throughput, otherwise havoc will ensue: one pump, for example, may move more slurry than the one upstream is providing, resulting in destructive cavitation.

Even if all the pump speeds could be set perfectly, these settings would only be applicable if all units were receiving the exact same slurry. If the slurry characteristics were to change so that the upstream pumps received a different slurry from the downstream ones, then different throughputs would be pumped and system failure would occur. Burnett writes:

> If the incoming solids cannot be maintained at a constant rate, which is usually the case, the centrifugal pumps will be forced to handle varying flow rates, slurry densities, and consequently, varying system pressure requirements. Such conditions can present a monumental control problem for a system using conventional centrifugal pumps. Hydraulic transport lines several thousand feet long with centrifugal pumps appropriately located along the line are subject to severe water hammer and pump cavitation if the input conditions (solids concentration, solids density, and solids or fluid flow rate) change. [11]

Link's previously-mentioned system of sump-variable-speed-pump units utilized to counteract pluggage development through dilution and variable pump pressure is one solution to the pressure transient problem. The sumps act not only to dilute the slurry when necessary, but also to vent it to the atmosphere so air bubbles entrained in the slurry and rapid pressure transients can be released. Additionally, using the sump-pump unit along with its related electronic sensors and control mechanisms, any deviation between the upstream rate of flow and the downstream rate of flow can be resolved.

This matching of consecutive pump throughputs is achieved through the small degree of throughput flexibility offered by the sump through its limited surge capacity and the reactive ability of the variable-speed pump to prevent any surge from going too far. For example, the sump has slurry level sensors in it, and if the level gets too high (the upstream pump is injecting more than the downstream one is removing), then a signal is sent to the associated variable-speed pump to increase its impeller speed and thus reduce the slurry level. If the level gets too low, on the other hand, as a result of the downstream pump ejecting more slurry than the upstream one supplies, a signal is sent to the associated pump to reduce its speed. If this action still does not reverse the low level problem, then water is added to the sump so that the minimum downstream throughput will be maintained (thus maintaining super-critical velocity). [111]

The use of sumps to control the pressure transient danger, however, adds serious dangers of its own. The sumps each have a very small capacity --they can be filled or emptied in a matter of a few seconds. If the level reaction mechanism were to break down, disastrous consequences could occur. In the less calamitous case, if the sensors fail to detect or if the pump fails to react to a falling slurry level, air will eventually be fed into the pump and cavitation damage will result. In the worst case, the lack of reaction to increasing slurry levels will result in sump overflow and the associated flooding of the drift, damage to electric equipment, and electrocution risk. [111]

Excepting the situation in which sumps would be used at slurry line intersections, Foster Miller's helical inducer would eliminate these dangers. What Link's system achieved through sumps, pumps, and electronic control, the helical inducer can attain through its slurry annulus. Rather than escaping through a sump, for example, air bubbles in the line escape into the air core of the inducer's annulus, which is vented to the atmosphere. Air bubles will not be injected into the next leg of the slurry line as long as the lower operational limit is not exceeded (the thickness of the slurry annulus must have a certain minimum level).

Furthermore, any pressure or flow transients caused by inhomogenous slurries or unmatched pump throughputs are absorbed by the annulus through changes in its thickness. A sudden increase in upstream input, for example, would increase the size of the annulus. This, in turn, would increase the delivered pump pressure according to the previously-mentioned pressure function, and increase the throughput through the downstream portion of the pipe, thus equalizing the two upstream and downstream flows. Burnett reports that transient flows of at least 35% change in rate can be handled without cavitation. [11]

ANALYSIS OF LONGWALL/SLURRY COMBINATIONS

The initial attractiveness of a longwall/slurry combination is a result of the fact that in many respects the two systems seem to be perfect complements in comparison to other mining-method/haulage-system couplings. Longwalls, for example, produce a particle size particularly well-suited for hydrotransport. According to Coal Preparation: "Current industry consensus recognizes that the longwall continuous mining system produces somewhat less of the coarser sizes than that produced from either conventional or continuous mining." This smaller initial size is important for two reasons: first, particle preparation costs near the face will be smaller in comparison to that of other mining methods since less crushing is necessary to reach the optimal particle size for slurry transport; and second, the slurry sytem does not have to bear the full responsibility for the extra dewatering costs incurred by reducing the coal particle from a potentially large size, such as that produced by a continuous miner, to one optimal for slurry transport, since much of this size diminution is an unavoidable result of the longwall mining method.

Additionally, longwall mining is more compatible with a slurry system in the sense that its output is less intermittent than that of continuous mining since the shearer does not have to be trammed to a new face every few minutes. Because production is more continuous, there will be fewer cycles of line flushing, shut-off, and start-up, and, thus, longer pump and valve lives, more efficient pumping (since more time is spent in the high-efficiency, full-operating-speed zone), less inadvertent slurry dilution, and longer pipe life (pipe wears rapidly when subjected to cycles of sequential exposure to air and slurry--the air oxidizes the pipe surface, which reduces its wear resistance to the abrasive slurry which follows. [86]

Slurry haulage and longwall mining also have very compatible mine lay-out philosophies: both find that sequential development of consecutive panels along a long main or sub-main is ideal. In the case of longwalls, this pattern is important because longwall capital costs are immense and non-productive transition times (when the face equipment is being transferred from an old panel to a new one) can be minimized by moving to immediately adjacent panels. In the case of slurry transport, the pattern is important because it means that the sizable investment which goes into the construction of sumps and pumping stations in the main results in an infrastructure which can be used for virtually the entire life of the mine instead of being discarded after five months upon panel exhaustion.

Finally, a longwall system's large scale is an asset when considering slurry haulage application. Generally, a longwall unit produces three times the output of a continuous miner. A longwall mine, therefore, would have one third the number of production panels feeding into a slurry system as compared to a mine for the same output utilizing continuous miners.

Consequently the cost and the complexity of the haulage system would also be significantly reduced, as the number of productive panels decrease.

Clearly, then, the fact that slurry haulage and longwall mining are in certain respects extremely compatible cannot be denied. The crucial issue, however, is whether they are compatible to such an extent that the slurry haulage system can be considered as a valid alternative to the conventional lonwall haulage method: the conveyor belt.

An analogous question was faced in the Seventies when interest in underground mining applications of coarse-coal slurry technology generated a series of papers dealing with the technical and economic feasibility of coarse-particle slurry haulage as applied to room-and-pillar mining. [67, 111, 112] These papers justified the use of a slurry haulage system through attacking the weaknesses of the conventional room-and-pillar haulage method: the shuttle car. First, the shuttle cars damaged productivity: the continuous miner would often sit idle as shuttle cars were embroiled in an underground traffic jam. Second, the shuttle cars damaged safety: the presence of fast-moving vehicles created a dangerous mining environment. The papers then argued that slurry haulage, on the other hand, would eliminate delays by removing the miners' output in a continuous stream and would create a safer environment by replacing a multiple of fast-moving vehicles with a single one which would merely follow the sluggish movements of the continuous miner.

Although the current study is a step in the same direction as these earlier papers (feasibility of a practical application of coarse-slurry technology to a mining operation), it is also a step beyond them. In justifying the application of the slurry haulage system to a longwall unit, the arguments of continuous haulage and elimination of fast-moving vehicles are not applicable as this level of sophistication has already been established in the longwall system through the use of conveyor belts. It is necessary, therefore, to go a step further than has been done previously: slurry haulage must be compared not to an inefficient, unsafe batch haulage system, but rather to a system which is already continuous and relatively safe. This study intends to make such a comparison: slurry and conveyor haulage systems will be designed for an assumed longwall mining scenario, and the two alternatives will then be evaluated and compared on the basis of cost, reliability, and such intangibles as health and safety.

The assumed mining scenario is a case in which two longwall units mine 500' by by 5000' panels in a 5.5-foot-thick coal seam 1000 feet below the surface. Coal from the face is transported 5000 feet through the panel entry, 2000 feet through the sub-main, and 2 miles through the main haulageway to shaft bottom (see Figure 9). Continuous miners, which are typically used to develop the longwall panels, are not included in the model as this would unnecessarily complicate the problem.

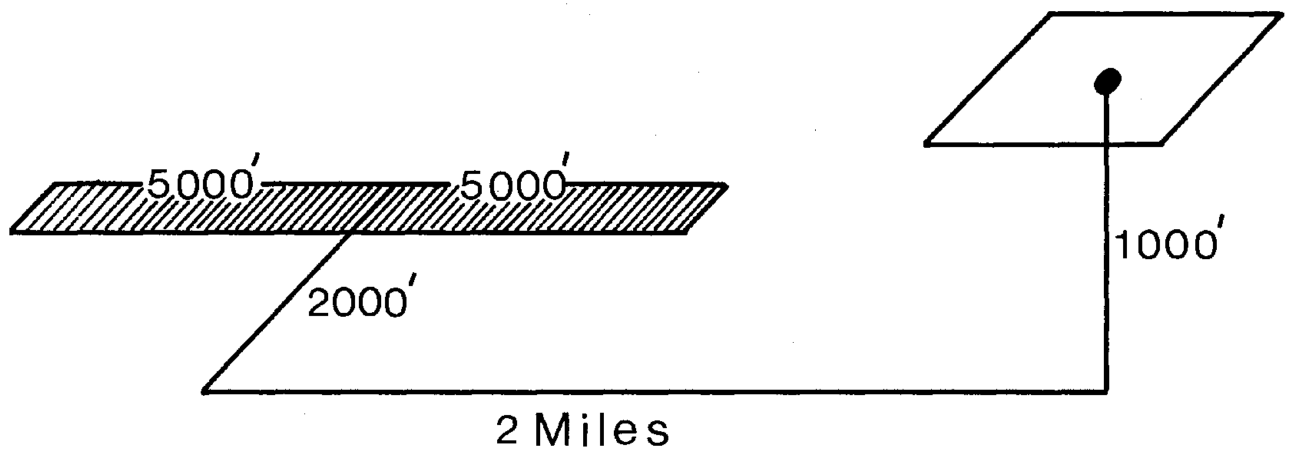


Figure 9. Mine Lay-Out.

Slurry System Design

A fundamental unit exists in slurry technology from which all slurry transport systems are constructed. This unit, called a "leg," usually consists of a sump, a pump, and the subsequent length of slurry pipe. Any slurry transport system is a conglomeration of these legs, and an essential design problem is how to arrange them in an optimal manner. For example, in the mining scenario assumed in this study basically four possible arrangements exist. [Figure 10]

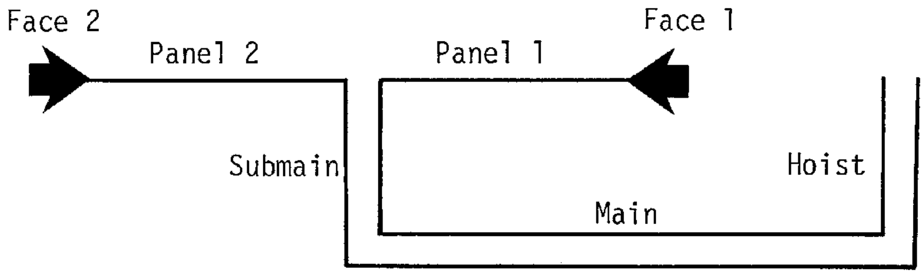
The actual design of these layouts and the determination of which one of them is economically optimal (and thus which one should be compared to the analagous conveyor system) is extremely complex. The need to do these extensive calculations by hand, however, was circumvented by the discovery and utilization of an earlier study done for U.S.D.O.E., Evaluation of Alternate Hydraulic Transport Concepts For Coal Haulage in Underground Mines, Transflux Intn'l. [112]. This study simulates the output characteristics of a multiple-face, room-and-pillar mine utilizing continuous miners, designs, through the use of computer programs and a cost data bank, all of the possible slurry system lay-outs compatible with these output characteristics, and finally determines which of these lay-outs is optimal in terms of cost minimization.

All that was necessary to do in order to convert this continuous miner/slurry haulage design program into one which would apply in a longwall mining context was to change the simulation of the continuous miner production so that it would generate the characteristics of longwall output instead. The flexibility of the program allowed this to be done by merely changing the inputs to the program rather than changing the program itself. For example, it was assumed that the longwall units were double-drum shearers (3.5 foot drum diameter) with a web of 30", which mined 500' along the face of the panel, took one mine to reverse direction, and then mined another 500' on the return trip.

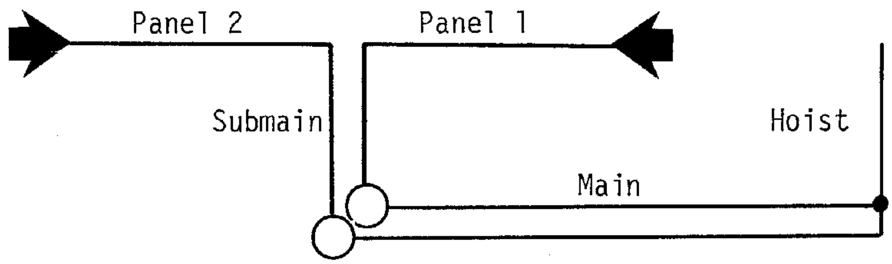
To mimic the resulting coal production characteristics, the input to the program was changed so that the program would evaluate a continuous miner which ignored roof bolting procedures and mined freely for 500 feet before taking one minute to tram to the next face and repeat the same action. Longwall shearer cutting characteristics were simulated by inputting into the program certain appropriate "sump-in" and "shear" times for the continuous miner so that the ultimate quantity of coal produced per unit time would be the same as that assumed to be produced by the longwall miner. These changes ultimately had the effect of creating a continuous miner whose coal output characteristics have the same as a longwall miner. The program could then utilize these characteristics to select an economically-optimal combination of pump, sumps, and pipes in accordance with the dimensions of the mine and dynamics of the slurry flow.

There are, however, certain limitations involved in the use of the Transflux program. First, the extent of this study's cost analysis is

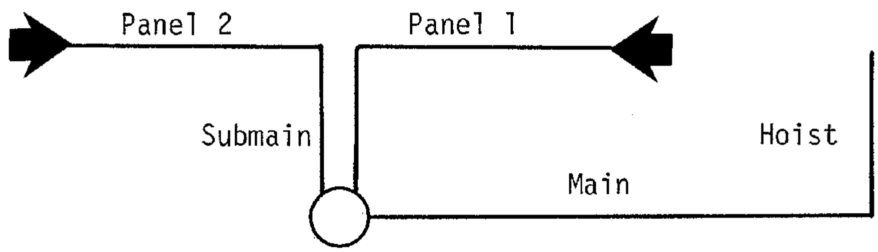
Type I: Totally Independent Slurry Systems from Face to Surface.



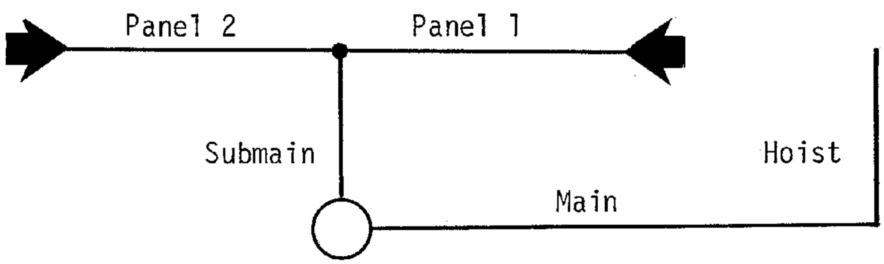
Type II: Independent Systems to Shaft Bottom, Combined Thereafter.



Type III: Independent Systems to Main, Combined Thereafter.



Type IV: Independent Systems to the Submain, Combined Thereafter.



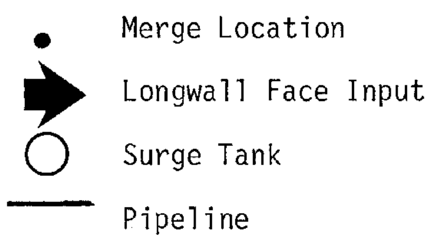


Figure 10. Schematic Representation of Four Alternatives of Two-Face Hydraulic Haulage for Longwall Mining.

limited by the program structure from the face to the shaft collar instead of all the way up to and involving the preparation plant. This second option would be more certain of capturing more completely the costs incurred along the entire haulage route. As a result of the shortened analysis, the costs of slurry system's surface water-clarification unit and the added fines capacity needed in the preparation plant as a result of en route particle size degradation are not included in the comparative cost calculations.

Counterbalancing this, however, is the fact that the Transflux program may overestimate the slurry power costs. First, the program applies a single power cost rate of 4¢/kw/hr to all power consumption despite the fact that utilities generally charge different rates depending on the quantity or power consumed. Second, slurry friction losses may be overstated. The Transflux program considers the pipeline slurry friction loss "to be 30% greater than that incurred by clear water flowing at the same velocity. Headlosses for intermediate slurry concentration between $C_W = 0.45$ and $C_W = 0$ are assumed to be proportioned with the concentration." [112] The generality of this assumption is understandable considering how little is known about coarse-particle slurry flow. The problem, however, is that the same friction loss equation is applied to both the horizontal and vertical portions of the slurry system despite the fact that friction loss in vertical hydrotransport is known to be much lower than that encountered in horizontal slurry transport [20]. This has two effects: (1) the power costs for hydrohoisting may be exagerated, and (2) the design of the vertical slurry leg may be adversely affected, i.e., a decision based on this faulty friction loss figure may result in too many pumps being utilized at shaft bottom, different models of pumps being chosen than would ordinarily be used, and dilution occuring when it actually might not be justified.

A third reservation with the Transflux program is that in some respects it lacks a certain awareness of physical reality. For example, in many cases the severe space constraint present in underground mines is not taken into consideration. No attention is paid to whether the sumps, pumps, and motors specified in the program will actually fit into the space available. In cases where the equipment specified obviously cannot be contained in the 5 foot seam area, no extra costs are allowed for excavation of a volume large enough to contain the equipment.

Regardless of these shortcomings (which are rather insignificant in the face of the program's overall reliability), the converted program was utilized to analyze the four possible slurry system arrangements mentioned earlier for four different longwall shearer speeds (and thus four different levels of longwall output). It was decided that the ultimate design and cost analysis for both the slurry and conveyor systems would be based on the peak output from the face, i.e., when the shearer runs at the highest speed--15 feet per minute. If no delays were to occur, this shearer speed would result in a longwall output of 3,797 tons per shift per longwall unit. Peak output as a basis for design was chosen because then it would be certain that both haulage systems would be technically feasible in the sense that they could handle any output, whatsoever, delivered by the longwall unit.

Based on this peak output choice, the modified computer program designed the compatible slurry system alternatives (see Appendix IV for example of program results) and calculated their resulting costs per ton of coal transported. Type I, type II and type III have costs of .596 per ton, .592 per ton, and .558 per ton, respectively, while type IV was found to be the cost-minimizing alternative at $.552 per ton. This alternative, which is the one which shall be used in the comparison to the conveyor belt analogue, is represented in Figure 11 along with accessory systems not designed by the converted Transflux program, namely, the water supply system, the shaft, the service hoist, and the emergency shaft-bottom sump.

Conveyor System Design

The design process for the conveyor belt system as applied to the assumed dual-longwall mining scenario is not nearly as complex as that for the slurry system, particularly since basically only one layout is feasible for the conveyor arrangement instead of the multitude of alternatives available in the slurry system. The conveyor belt-hoist haulage arrangement to be used in the comparative analysis is pictured in Figure 12, its design mimicking actual longwall and other underground haulage systems described in mining literature.

Systems Comparison

Having completed the designs for the two haulage systems under consideration, it is now possible to begin their evaluation and comparison based on cost and reliability criteria. Assessment through the application of each of the criteria alone, however, is not as fruitful as applying them simultaneously. For example, consider the two alternative cases in which one system is cheap but very unreliable, while the other is expensive but very reliable. If the cost criteria alone is applied, then the first system is chosen as optimal. If, on the other hand, the reliability criteria alone is applied, then the second system is chosen as optimal. To circumvent such contradictory results, it is therefore necessary to apply both criteria simultaneously.

This objective is obtained in this study by comparing the two haulage systems on the basis of their cost-per-ton-transported figures (total costs incurred during a specific time period divided by the tons output produced in this period). Since a change in total costs changes this index through changing the numerator, and since a change in reliability changes the index through changing the denominator (a fall in reliability, for example, will result in fewer tons produced per unit time), both cost and reliability criteria are simultaneously at work in the cost per ton index, which makes it a reasonable indicator of the worthiness of the two haulage methods.

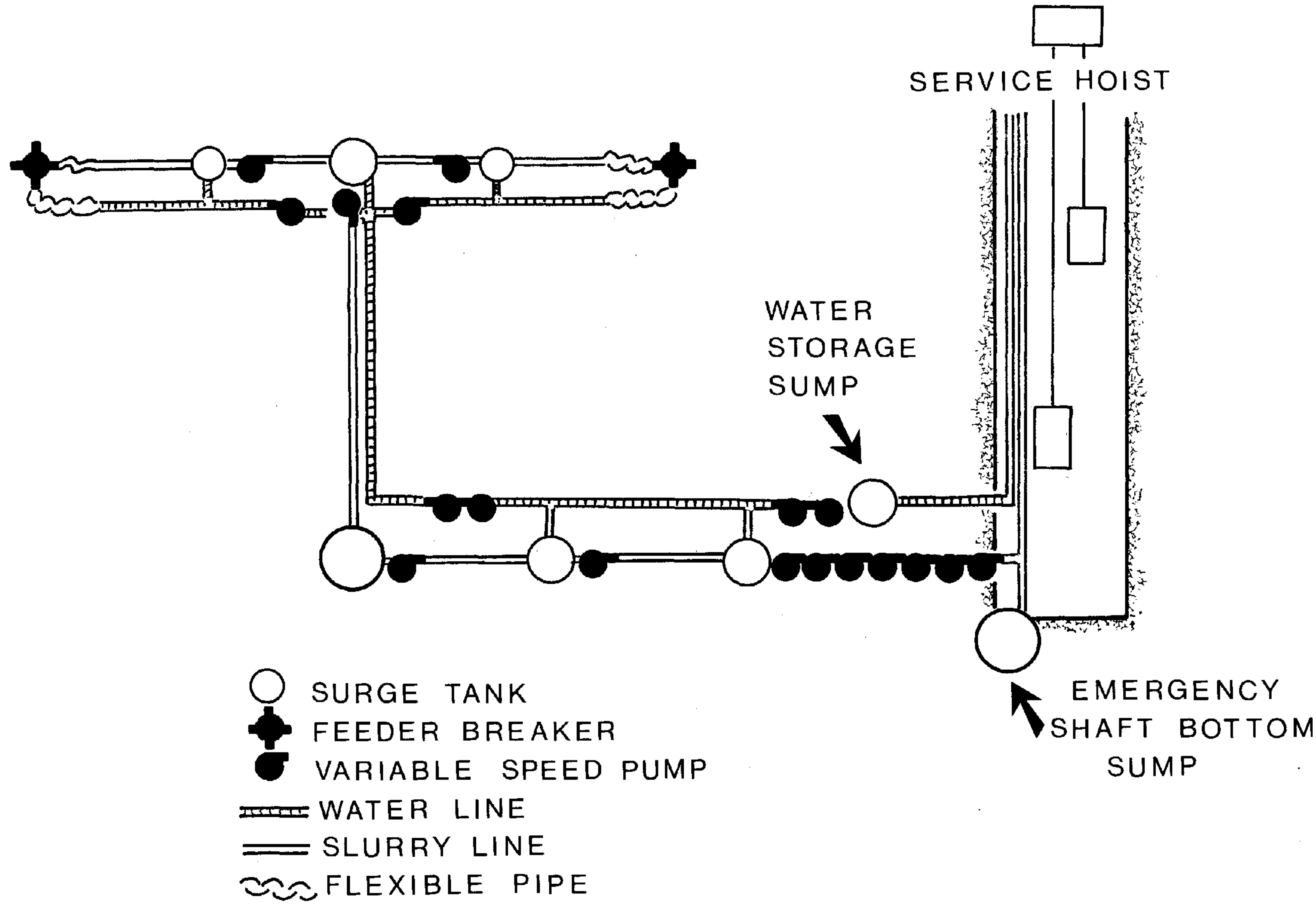


Figure 11. Longwall-Slurry Haulage System

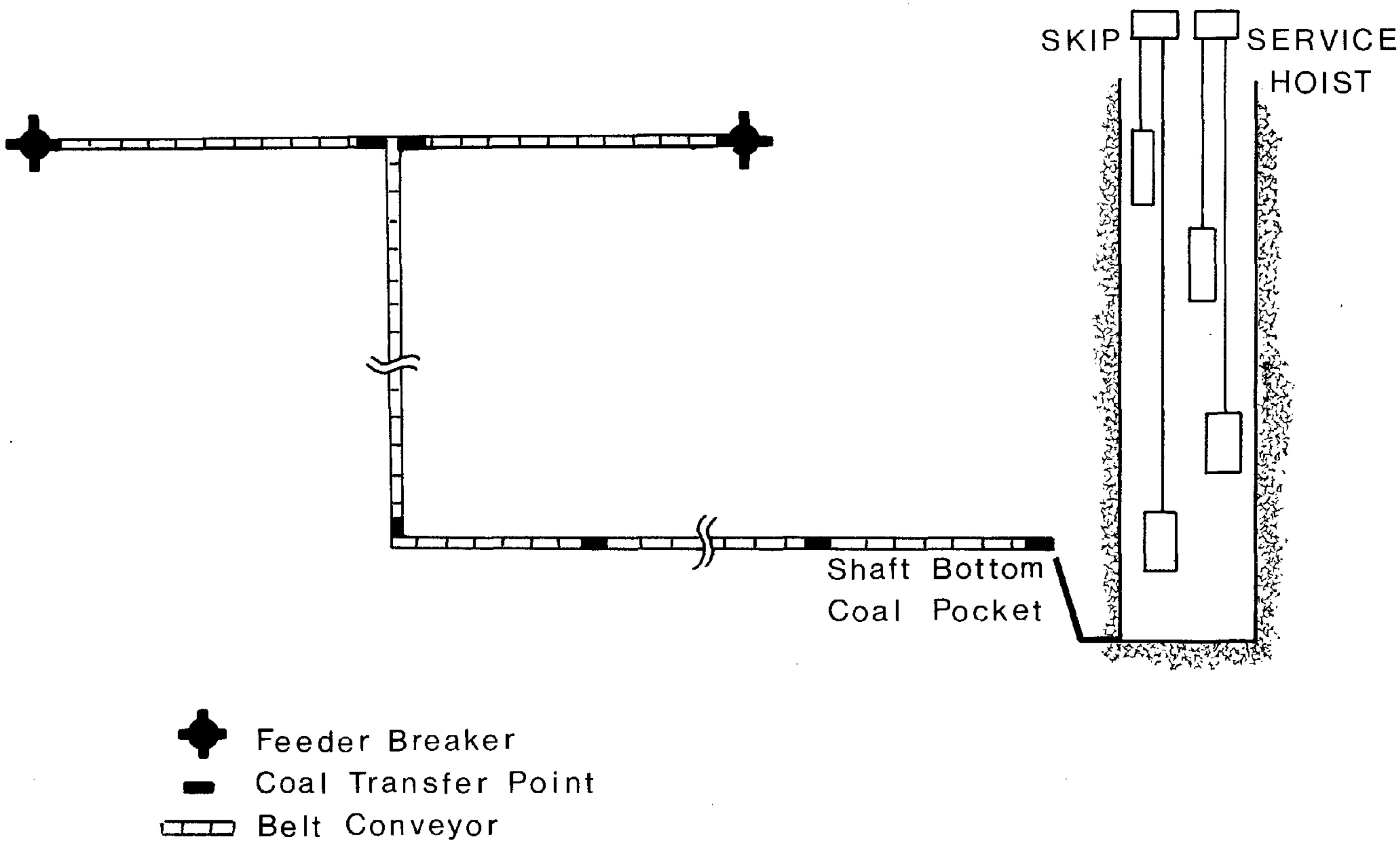


Figure 12. Schematic Representation of Belt Conveyor Haulage System for Two-Face Longwall Mining

To calculate the cost per ton values for the two systems, it is necessary to have both cost and delay data. Since the objective of this study is to compare the relative costs of the two haulage systems rather than their actual total costs, a relative cost method was followed in analyzing the systems. In other words, since the objective is to look for relative cost differences between the systems, it is unnecessary to take into consideration items which both systems have in common. For example, the slurry haulage system would in reality need a man and material transport system independent of the coal haulage system. Since the same system is needed by the conveyor haulage arrangement, however, no difference in relative costs between the two systems would be created by including it in the cost formulation, so it is left out.

On the other hand, when one of the haulage systems has a source of cost which is not used in the other system or is used on a different scale, then this cost must be included in the relative cost analysis. For example: both haulage systems use shafts, but the compact nature of slurry haulage allows a smaller diameter shaft than is used in the conventional system; thus, shaft costs are included in the cost analysis because they were a source of relative cost difference.

In certain cases assumptions were made on the equivalency of components in the two systems. Labor costs, for example, were not included in the cost analysis because it was assumed that both transport systems used the same amount of labor. Additionally, the stage-loader (conventional system) and feeder-breaker (slurry system) were not involved in the relative costing because it was assumed that both were equivalent in terms of cost.

Due to budget and time limitations, cost information had to be obtained from secondary sources (e.g., journal articles, handbooks, etc.) instead of the equipment suppliers themselves, and, thus, a certain amount of precision was lost, particularly when this information applied to equipment of a different capacity than that used in the longwall model haulage systems, making the use of adjustment approximations such as the 6/10's rule necessary (see Appendix IV). In the evaluation of the relative cost of the slurry system, for example, cost data was taken from the studies on room-and-pillar-mining/slurry-haulage applications done in the Seventies [67, 111, 112], U.S.B.M. information circulars on coal mine costs, mining handbooks, and journal articles uncovered in the literature survey.*

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*It is important to note that the modified Transflux program does not supply all of the costs involved in a slurry haulage system. The program was originally written to choose the cost-minimizing alternative among several slurry system options. To do this, it used a relative costing method just as this study does. The difference, however, is that while this study compares a slurry haulage system to a conventional haulage system, the program compared only slurry haulage systems, where, of course, fewer sources of relative cost difference exist. This study, Costs from these sources were adjusted, if necessary, to fit the capacity needed in the proposed haulage models and were then updated to 1981 using indexes published by the Department of Commerce in its Survey of Current Business.

In the matter of operating costs, two assumptions were made. First, since no data is available on the maintenance costs of the slurry system components and since the determination of the specific values of such costs for the conventional system is well beyond the time and budget capacity of this study, annual maintenance costs were assumed to be ten percent of the total capital costs of the piece of equipment under question, except in the case of low maintenance shafts and transformers with which a rate of 1% was applied. Second, power cost calculations were based on the assumption that the equipment consumes power at its full rated horsepower for the duration of its entire operating time. In the handling of capital costs, a present-value analysis was not deemed necessary as a result of using a cost-per-ton index. Instead, capital costs per ton were calculated by dividing each piece of capital equipment's cost by the tonnage it handled over its lifetime (Appendix IV).

As mentioned previously, the study's comparison analysis is based on a cost-reliability mixture as incarnated in a cost-per-ton index. Cost data for both the slurry and conveyor systems are available, as can be seen in Appendix IV. Reliability data for a conventional longwall conveyor haulage system is available in a previous U.S.D.O.E. study put together by the Jet Propulsion Laboratory under the title Longwall Mine Availability and Delay Analysis. Reliability data for an analagous longwall slurry haulage system, however, is non-existent, primarily because only four such systems exist in the world, three of which are in the U.S.S.R. [86] Since no slurry reliability data is available, the study calculates slurry transport cost per ton based on perfect haulage operation (no transport failures) and conveyor transport cost per ton based on operation characterized by the Jet Propulsion report, and then will compare the two figures, determining how reliable a slurry system would have to be to compare to the conveyor system in terms of cost per ton of coal transported (see Appendix IV).

From the cost-per-ton figures (Table 1) it is possible to see that even if the slurry system were perfectly reliable, it would still be more expensive than the conveyor system operating with the failure characteristics described in the Jet Propulsion report. The sources of the cost difference are self-evident: although the capital costs per ton are three fourths that of the conventional system, the slurry system's power costs per ton are three times that of the conventional haulage system.

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therefore, while using the modified program's cost information, had to supplement it with cost information on slurry system components that the program ignored since all alternatives held them in common, namely: the shaft, service hoist, emergency shaft-bottom sump, and the water, power and control-and-automation systems.

Table 1. Comparison of Slurry and Conventional Haulage System Costs

	Slurry System			Conventional System			
Cost Area	Capital and Maintenance $/Ton	Power $/Ton	Total $/Ton	Capital $/Ton	Maintenance $/Ton	Power $/Ton	Total $/Ton
Panel Haulage	0.042	0.027	0.119	0.050	0.044	0.009	0.103
Submain Haulage	0.018	0.012	0.030	0.019	0.014	0.004	0.037
Main Haulage	0.091	0.058	0.149	0.093	0.080	0.017	0.190
Hoisting							
A. Hydrohoist	0.064	0.248	0.312	-	-	-	-
B. Service Hoist	0.036	0.037	0.073	-	-	-	-
Total Hoisting	0.100	0.285	0.385	0.052	0.104	0.086	0.242
Shaft	0.072	-	0.072	0.087	0.017	-	0.104
Power Supply	0.026	-	0.026	0.019	0.003	-	0.022
Shaft Bottom Sump	0.0005	-	0.0005	-	-	-	-
Loading Pocket	-	-	-	0.005	0.010	-	0.015
Water Supply System	0.010	-	0.010	-	-	-	-
Control and Automation	0.029	-	0.029	-	-	-	-
TOTAL	0.439	0.382	0.821	0.325	0.272	0.116	0.713

The single largest source responsible for this power cost discrepancy is the hydrohoist. This unit's power consumption alone is responsible for 65 percent of the entire slurry system's power costs per ton and 30 percent of its total cost per ton. The power usage may be slightly exagerated, as explained previously, as a result of applying the same friction loss equation to the vertical line as the horizontal one, but, nevertheless, any slight improvement in the hydraulic hoisting would have a large impact on the total costs per ton and thus on the feasibility of the entire slurry haulage system.

Compared to power costs, the capital costs per ton of the two systems are not as disparate, largely because of component differences which counterbalance one another in terms of cost. More specifically, while the conveyor alternative neither needs the extensive water supply and control-and-automation systems or such a large power distribution grid, all of which are indispensible to the slurry system, the slurry alternative neither needs the expensive ore pocket at shaft bottom nor a large-diameter shaft for hoisting.

The cost per ton results also reveal an interesting trend. Looking at the main, submain, and panel components of the horizontal portion of the haulage systems, one finds a certain consistency: in all the components, the power cost per ton of the slurry system is always about three times that of the conveyor system, and, except in the panel, the capital and maintenance costs are half that of the conveyor's. In the panel, however, the capital and maintenance costs are approximately the same in the slurry alternative as in the conveyor alternative. Since this discrepancy occurs right at the point where the capacity of the line changes from 3797 t/shift (assuming no failure) to 7594 t/shift, the explanation lies in economy of scale--haulage gets cheaper when larger throughputs are transported, and, more important, slurry haulage gets cheaper faster than conveyor haulage does.

This can be seen more easily by dividing each leg's cost-per-ton figure by the length of the leg so a common unit, cost per ton per foot, can be used to readily compare costs between legs (Table 2).

If there were no economies of scale, then each leg of a system would have the same total cost per ton-foot value. Upon looking at the cost figures above, however, this is obviously not the case. The total costs per ton-foot of the main and submain are roughly the same, but this is how it should be since both these legs have the same capacity (e.g., in the slurry system 17.25" ID pipe is used both in the submain and the main). A large jump in total costs per ton-foot is noticed, however, between the panels and the submain. This is due to the fact that economies of scale come into play when the system changes from transporting 3797 t/shift to 7594 t/shift. What is most significant, however, is that as the coal travels from a low-capacity system to one of high-capacity, the total costs per ton-foot in the slurry system drop 37% while in the conveyor system the drop is only 10%.

Table 2. Comparison of Slurry and Conventional Haulage Systems on the Basis of Cost Per Ton-Unit Distance (10^{-5} \$/Ton-Foot)

	Slurry System			Conventional System		
	Panel	Sub-main	Main	Panel	Sub-main	Main
Capital and Maintenance	1.84	0.90	0.86	1.88	1.65	1.64
Power	0.54	0.60	0.55	0.18	0.20	0.16
TOTAL	2.38	1.50	1.41	2.06	1.85	1.80

The source of this economy of scale seems to lie in capital and maintenance savings rather than power savings: one sees that between the panel and the submain capital and maintenance costs drop 51% and 12% in the slurry and conveyor systems, respectively, while power costs show no significant change at all.

The ultimate impact of the fact that slurry and conveyor haulage systems have different economies of scale is that the choice of haulage systems may change with the capacity of the mine. In this study a longwall mine with only two faces was analyzed and a conveyor haulage system was found to be more economically justifiable. In a mine with more longwall faces, however--one which takes full advantage of the slurry system's better economies of scale--it may be found that a slurry haulage system is more economically justifiable. This study, therefore, asserts that in a two-face longwall mine a slurry system does not have a clear-cut cost advantage over a conventional haulage system, but under other circumstances (i.e., larger scale mine) such an advantage may become evident.

Comparison of Intangibles

Up to this point, only cost has been considered as a method of system comparison. This has been to the slurry alternative's disadvantage as some of its greatest contributions are in areas which cannot be easily subjected to cost analysis. Cost is clearly only one of several criteria by which decisions are made, and therefore, to truly judge the comparative potential of the two haulage systems, it is necessary to evaluate them on a wider scale--one which includes intangible factors.

Frequently the mistake has been made that because a factor is intangible (difficult to assign a cost or value to), its impact is insignificant. This is a fallacy; intangible factors are just as important as easily-quantified ones but are merely more difficult to analyze. For example, the impact of health and safety on the ultimate success of a mining venture is difficult to quantify, but this does not mean that it is insignificant: a recent study, for example, surveyed twenty underground coal mines and managed to assess their health and safety costs at $3.75/ton of coal.* Clearly, intangible factors ultimately influence mining costs greatly and therefore deserve as careful an analysis as tangible factors.

Just as the slurry and conventional haulage systems alternately had features which were sources of cost advantage and disadvantage, the same is true in terms of intangible factors. Although these features could not be evaluated semi-quantitatively due to the study's budget and time constraints, a few qualitative differences can be pointed out.

Perhaps the most significant of these intangible differences between the two haulage systems lies in the area of health and safety. First, conveyor systems are a serious source of mine accidents because their use, as with shuttle cars, subjects mine workers to a fast-moving, exposed haulage method. A slurry system, on the other hand, isolates the fast-moving element from the workers by containing it within a pipe. The only comparable hazard introduced by the slurry system is the remote risk of pipe burst or sump overflow.

Second, conveyor haulage systems create an unhealthy mine environment. Due to the quantity of coal dust generated during crushing at the face and at transfers, health and safety regulations require that entries used for conveyor haulage not be used for ventilation: the ventilation in these entries is to be kept strictly neutral. With a slurry system, however, such a costly ventilation practice would be unnecessary as the coal is totally confined within a stream of fluid travelling within a pipeline, thus never being in contact with the mine environment in a manner which would generate a health problem.

Finally, conveyors are a serious fire hazard. More than fifteen percent of all mine fires between 1950 and 1977 were caused by conveyor belts, usually either through electrical failure or as a result of friction heat (e.g., frozen idler or a belt jammed in the drive). In fact, belt conveyors and undercutters are considered to be the two most dangerous pieces of equipment in underground coal mining in terms of fire hazard (U.S.B.M. IC 8830). With slurry haulage, on the other hand, this danger of mine fire generation is virtually eliminated. With the exception of the possibility of electrical failure, the slurry system poses no threat of fire hazard at all and even would reduce the threat in other areas of the mine by supplying at all times a ready, high-volume source of water.

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*Robert T. Reeder and Tom Johansen, "Cost of Safety and Health Regulations at Selected Western Coal Mines," American Mining Congress Coal Convention, session paper, set n. 2, Safety and Health, St. Lois, Missouri, May 20-23, 1979, p. 13.

A second intangible area in which the difference between the two systems which must be considered is that of reliability. Currently it appearsas if conventional haulage is one of the weaker links in the longwall mining system--using Jet Propulsion data, roughly fifteen percent of the time the face equipment is able to operate, it cannot, due to problems in the external conveyor system. Slurry haulage, on the other hand, appears to have potential for higher reliability levels.

In a serial system, for example, reliability can be improved in three ways: (1) through an increase in the reliability of each individual component; (2) through a reduction in the number of serially-connected components; and (3) through the construction of a parallel system, i.e., redundancy. The slurry system appears to be an improvement over the conventional system in all but the first of these ways (in which it is not possible to compare the two haulage alternatives due to the current lack of information on the individual reliability of the slurry system's components).

Slurry haulage, for example, has fewer serially-connected components. A conveyor system, after all, is literally one long string of moving parts, and a failure in any one of these parts is enough to bring the entire system down. If a single idler freezes, for example, the friction may be so great as to start a fire and thus stop haulage (15% of all mine fires have their source in conveyor belts, of which 70% are caused by frictional heating). A slurry system, on the other hand, has fewer serially connected components, particularly in a system utilizing helical inducers, and thus is potentially more reliable.

The other previously-mentioned way to improve reliability is to make a parallel system. This can be done in one of two ways: (1) by actually constructing an entire duplicate system next to the operating one, or (2) by having duplicate components readily available to replace any malfunctioning units. This first alternative is not available to a conveyor haulage system even if cost analysis allows it--conveyor systems are so bulky that in an undergound environment they are subject to a space constraint. The slurry system, however, does not face such a constraint--it is so compact that duplication or triplication of the system would not impede access in the haulageway.

Component duplication, as opposed to total duplication, is a more realistic method of heightening reliability, but here, as well, the slurry system has the advantage. Slurry systems, for example, have their components which are subject to high failure centralized into a few specific locations (pump stations) while the rest of the system is virtually impervious to failure (excepting the case of pluggage). This centralization makes it easier for the system to be monitored for failure and have duplicate parts readily available to replace defective ones. A conveyor system, on the other hand, has its failure-prone components distributed throughout the entire lay-out so discovery and resolution of failure is more difficult.

Additionally, with the slurry haulage alternative it is possible to build into the system and thus move closer to continuous, 24-hour-a-day, seven-day-a-week haulage. Pumps, for example can be installed in parallel so that when one breaks down or is in need of maintenance, the other can substitute while repairs are being made. Such built-in duplication in the conveyor haulage alternative, however, is impossible: two belt drives, for example, cannot be installed in parallel and immediately substituted for one another in the event of failure. Instead, the belt must be stopped, the defective drive removed, and the new drive inserted in its place.

Clearly, then, due to the facts that total duplication of a conveyor system is made difficult by underground space constraints and that component duplication virtually always requires that the belt be stopped while changes are being made, the conveyor haulage alternative cannot achieve high levels of availability except by making each of its many serially-connected components extremely reliable, which is a difficult task. The slurry haulage system, on the other hand, has all three methods of reliability improvement open to it and thus can almost certainly achieve whatever level of reliability is desired.

A final intangible advantage which slurry systems have over conveyor systems is space flexibility. As mentioned previously, underground conveyors face a rather severe space constraint; the systems are relatively large and bulky, and future prospects of compaction are slim. Slurry systems, on the other hand, are compact, and this compactness gives them a flexibility which conventional haulage systems will never have. For example: slurry's space efficiency allows great flexibility in haulage capacity, and therefore mine management will no longer have such strong constraints on the scale at which they mine. In an operation with slurry haulage a mine manager could decide to triple total mine production and this would have no effect on haulage costs other than the expenditures involved in laying a new pipe system next to the original one. Such a decision for a mine with conventional haulage, however, would be out of the question: it would be literally impossible unless the mine was redesigned and massive amounts of capital were sunk into sinking a new shaft or expanding the old one and driving new drifts for extra haulage parallel to the original ones.

CONCLUSIONS AND SUGGESTIONS FOR FUTURE WORK

This study concludes that for the case of a two-face longwall mine, a coarse-coal slurry transport system does not have a clear-cut cost advantage over an analogous conventional (conveyor-hoist) system. Such an advantage, however, would possibly materialize if a larger-scale mine were analyzed or if intangible benefits, which make up some of the slurry system's major contributions, were quantitatively analyzed and included in the cost comparison. These efforts, however, could not be carried out due to the study's budget and time contraints.

Additionally, it is important to realize that coarse-coal technology is developing faster than it ever has before, and due to the recent surge in research and development investment, a new generation of slurry technology is on the horizon. While this study analyzed the potential of slurry haulage as applied to a longwall system based on currently available technology, and concluded that no clear cost advantage over the conventional haulage system exists, this conclusion may be reversed when the same mining scenario is analyzed using the new-generation technology.

For example, if a hydrohoist and helical inducers were used in this study instead of centrifugal pumps, then the cost/ton outcome would have been much different. According to a recent article in Mining Journal, with a hydrohoist such as the one used at Ruhrkohle's Hansa Hdyromine: ". . . energy consumption, compared with a multi-stage centrifugal pumping system, is reportedly reduced by up to 40%. This, with other cost advantages, is said to provide overall savings in costs/ton handled of up to 30%." [120] Additionally, by using helical inducers, the costs of variable-speed drives and the control and automation system can be virtually eliminated from consideration due to the fixed speed, self-regulating nature of the inducers. In the system analyzed by this study, these improvements would have resulted in a slurry transport cost reduction of roughly 13 cents per ton, placing the slurry alternative in direct competition with the conventional system in terms of cost alone.

Clearly, then, the analysis of the potential of coarse-coal applications to longwall mining systems cannot be definitively established in a study of this scale. Future study is necessary, and it is suggested that:

1. Such a study should do a more comprehensive analysis of intangible costs and benefits. Past slurry system feasibility studies have primarily focused on cost analysis alone, ignoring some of the slurry alternative's greatest contributions--those in the intangible areas.

2. Such a study should include and take advantage of the latest information on the new, oncoming generation of coarse-particle slurry technology.

3. Such a study should have stronger cost-information retrieval capability. Reliable, accurate information on coarse-coal slurry systems is difficult to obtain.

4. Such a study should approximate more closely the actual longwall/slurry mining system which would be necessarily used in practice. For example: this study took only longwall units into account, although in practice a few continuous miners always work in conjunction with longwalls. This simplification was made in the study to reduce the complexity of an optimization problem which would otherwise be beyond the range set by the study's time and budget constraints. Such an analysis of the total system, both longwall and continuous miners, should be made.

5. Such a study should base its feasibility analysis on the total haulage system from the face to the preparation plant rather than any segment. If this is not done, all of the associated costs may not be included.

6. Such a study should analyze a wide variety of mine sizes in order to learn about the comparative economies of scale in slurry and conventional haulage systems and thus in what situations one alternative should be used to the exclusion of the other.

7. Such a study should globally optimize its slurry model design. The slurry system considered in this study was designed under the assumption that if each leg were designed to be optimal, the sum of the legs would be the optimal design. This assumption, however, is not totally true in that a sacrifice in one leg may result in a more-than-compensating return in a leg further down the system. Therefore, to get the optimal design for the cost comparison, it is necessary to use global optimization.

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APPENDIX I

COARSE PARTICLE SLURRY INSTALLATIONS AND R&D EFFORTS

U.S.S.R.

Current Soviet Coarse Particle Hydrotransport Installations

Detailed information on the slurry systems of Soviet facilities is rather rare. Names and outputs of various hydromines are available, however, from which it is possible to infer that hydraulic hoisting, at least, probably exists in these facilities.

Krasnoarmeyskaya (Red Army) Mine:

Constructed in 1966. Annual production = 1,350,000 metric tons. 25% of output produced by hydromining, 15% by mechanical cutting with hydrotransport, 60% by conventional mining (longwall). Coal which is to be hydraulically transported (-100 mm.) flows from the face to a sump by flume and is pumped one mile in a 420 mm. pipeline to the main shaft. A bubble pump system (1 of 2 in U.S.S.R.) then lifts the slurry 320 meters at a rate of 900 tph through two tapered airlift columns. Design slurry concentration is 10% due to mine water evacuation requirements. A surface slurry line from the shaft to the preparation plant is being planned to replace the current rail system. [81, 86, 129]

Yubileynaya (Jubilee) Mine:

Constructed in 1966. Annual production = 3,250,000 metric tons. Mining method is partly hydraulic, partly hydromechanical, and partly mechanical (longwall and conveyors). Particles are washed down in flumes to an area where they are crushed to -70 mm. and directed to 3 shaft bottom sumps (150 m^3 each). The slurry (solids concentration = 14%) is pumped up three centrifugal pumps and then pipelined 11 kilometers to a preparation plant where it is dewatered and conveyed to the Western Siberia Metallurgical Plant. [81, 86]

Inskaya #2 Mine:

Constructed in 1966. Annual production = 850,000 metric tons. Four thousand metric tons a day of coarse coal are pumped out of the mine in two lines. The slurry is then pipelined 10 kilometers to a preparation plant from which it is transported to the Belovskaya Regional Heat and Electric Plant. [86]

Baydayevskaya-Severnaya #1 Mine:

Has a 10 kilometer coarse-particle surface slurry pipeline. [86] Constructed around 1970. [82]

Other Soviety Hydromines:

Pioner Mine: Annual production = 780,000 metric tons; currently being modernized.
Krasnogorskaya Mine: Annual production = 750,000 mtons.
Tyrganskaya Mine: Anuual production = 300,000 mtons.
IM. 50-letiya SSR Mine: Annual production = 700,000 mtons.
Ordzonikidzeugol #4 Mine: Annual production = 70,000 mtons; contructed in 1960.
Ordzonikidzeugol #105: Annual production = 190,000.
Krasnodonugol ("50th Anniversay of the U.S.S.R.") Mine: Currently being modernized.
Makeyevugol Mine: Railway to preparation plant being replaced with coarse particle slurry line.
Donetskugol Mine: Railway to preparation plant being replaced with coarse particle slurry line. [86]

Soviet Hydrotransport Research Institutes

All-Union Research and Development Institute for Hydraulic Coal Mining (Hydrocoal Institute):

Founded in 1957 to research and develop hydromining methods and technology. Licensed its hydromining system to Mitsui Mining Co. and Kaiser Resources, Ltd. [129]

DonUGI:

Has 4 km. slurry test line. Has studied coal particle degradation during hydrotransport.

UkrNIIGIdrougal:

Has slurry test pipes of 1 to 4 kilometers in length. Has studied coal particle degradation and hydrotransport equipment wear. Its long-range hydraulic transportation department is involved in a methodical program of obtaining headloss and critical velocity data for virtually all possible coal hydrotransport scenarios: particle sizes from .2 mm to 50 mm., a variety of solids' densities, a wide range of particle size distributions, several different pipe diameters, and solids concentrations up to 52%. The Institute is also involved in replacing the short sections of rail between some hydromines and their preparation plants with coarse particle slurry transport systems. [86]

Long Range Hydraulic Transport Laboratory:

Has 12 kilometer, 306 mm. coarse-particle slurry test line. Has done coal particle degradation studies.

JAPAN

Coarse Particle Hydrotransport Installations

As of 1972, Japan had six hydraulic mining operations. What fraction of these utilize slurry pipeline transport has not been determinable. The two hydrotransport descriptions found in the literature survey are reported below.

Furukawa-Yoshima Colliery:

A one-year test program was conducted here in 1962 to prove the feasibility of coarse-particle hydrotransport equipment developed by Japan's Coal Mining Research Center (the Hitachi Hydrohoist). -50 mm. coal particles were transported 1.47 km. of which 253 meters was vertical. The line had a capacity of 100 metric tons per hour. [40, 51]

Mitsui-Sunagawa Colliery:

Hydraulic mining was initiated at the mine in 1965, a Hitachi Hydrohoist system was installed in the Noborikawa section that was to transport -30 mm coal particles 1,700 feet vertically and 6,600 feet horizontally to a preparation plant in a 7" line at a capacity of 95 metric tons per hour. By 1968, however, redesign became necessary as a result of extremely high power costs and wear rates. Instead of pumping all material less than 30 mm. in size, only -.75 mm particles were pumped (9% of the total output) with the remainder being removed by railcars. The 7" line was replaced by an 8" line to reduce pipewear by reducing the slurry speed from 14 fps to 11 fps. 178 metric tons of coal are transported daily by the system, which operates only 3.2 hours per day. Slurry concentration is 27% by weight. [126, 40, 51]

Research Institutes

Japan's Coal Mining Research Center and the Tokyo Institute of Technology started cooperating in 1953 to develop hydrohoisting and hydrotransport systems. An experimental pipefeeder prototype was completed and tested in 1962. The pipefeeder equipment now is supplied by Hitachi under the name Hitachi Hydrohoist. [51] Hitachi has an experimental surface facility involving a 4" 500-foot horizontal line. [40]

PEOPLE'S REPUBLIC OF CHINA

Hydrotransport Installations

China, as of 1977, had 10 hydromines. What percentage of these utilize pipeline hydrotransport is not clear, but two such installations found in the literature survey are described below.

Yang Chuang Mine:

Hydromine with an output in excess of 1 million metric tons per year. Hydrotransport system includes a 500-meter vertical hydrohoisting system lifting -50 mm coal particles. [40]

Lu-Cja-To Mine:

Was designed in 1958 by Polish mining engineers and is decribed as "probably the largest hydromechanized mine in the world." (Planned output = 8,600 metric tpd.) Coal is transported from the face to a screening area where -1 mm particles (35% of total) are separated. These particles are then centrifugally pumped 455 meters to the surface in a 400 mm pipe while the remainder is skip hoisted. Slurry concentration is 10% by weight and 11% by volume. [66]

WEST GERMANY

Coarse Particle Hydrotransport Installations

Gneisenau Mine:

Is one of the large pilot projects anticipating the Hansa Hydromine. Constructed by Ruhrkohle in 1971, coal is mined hydraulically, flumed to a shaft bottom crusher, crushed to -60 mm and then hydrohoisted 700 meters by a 2-chambered pipefeeder. Throughput is 150 m^3/hour; slurry pipeline velocity is 3.2-3.5 m/sec, and pipe diameter is 200 mm. Pipe-feeder chambers are from 150 to 300 mm in diameter and from 200 to 400 meters long. Hydrohoisting concentration is 33%. Recent research (1977) at the mine involved experimentation with the hydrohoisting of shaftborer debris, being carried out by KFA (Kernforschungsanlage Juelich GmbH) for Ruhrkohle AG. [117, 54, 40]

Hansa Hydromine:

The Hansa Hydromine is the culmination of an extensive R&D effort started in 1962 by Ruhrkohle AG with partial funding by the West German government. Coming on line in 1977, Hansa is the first full-scale, completely hydraulic coal mine in West Germany. (Planned output = 3,500 tpd of clean coal.)

Two underground coarse-particle slurry lines lead to shaft bottom from different parts of the mine, one being 2050 m. long, the other 2,830 m. Coal is broken at the face with a hydraulic monitor flumed to the feed station of one of the slurry lines where it is crushed by an 80 tph, 2-roll crusher to a -60 mm size. This coal is injected into one of the slurry lines and transported to one of two 800 m^3 collecting basins at shaft bottom. The slurry is then refluidized, pumped into a 3-chambered pipefeeder and hydrohoisted 2,800 feet.

Horizontal Hydrotransport Details:
particle size = -60 mm.
volumetric concentration = 17 to 50% solids.
pumping rate = two lines each transporting 12 to 14 m^3/minute.
pumping speed = 4 to 5 meters/second.
The 2,050 m line is in 6 m flanged sections 250 mm in diameter has a basalt lining, and has a pumping station with three 250 kw, 6 bar pumps.
The 2,830 m line is in 10 m seamless sections, 250 mm in diameter, constructed of St 35.2 steel, and has a pumping station with four 250 kw, 6 bar pumps.
The slurry lines also have small experimental sections of centrifugally-cast and polyurethane-lined pipe.

Vertical Hydrohoisting Details:
pipefeeder: has three 365 m long chambers each holding 18 m^3 of slurry. Has a 95% efficiency (only 5% of chamber contents is injected into the shaft line as clear water).
volumetric concentration = 50% solids (the slurry was concentrated at the 800 m^3 collecting basins feeding into the pipefeeder).
pumping rate = 450 metric tons of raw coal per hour.
pumping speed = 4 to 5 m/second.
pumps: three low-pressure centrifugal slurry pumps (3 bar, 132 kw) and two high-pressure 8-stage centrifugal water pumps (120 bar, 2,100 kw).
shaft pipelines: four interchangeable 250 mm PV 160 to PN 64 lines--two for water, one for emergency and one for slurry transport.

In November of 1980, the Hansa hydromine ceased production as a result of problems in the hydraulic mining rather than hydraulic transport section. Cited problems: hydromining produced a more dilute slurry than expected, and entrance of tramp material into the slurry lines caused frequent pump breakdowns. [105, 38, 54, 78, 44]

Research and Development Involvement

Ruhrkohle AG:

West Germany's premier coal-producing company. Started a hydromining research and development project that went through four small-scale pilot plants and two large-scale experimental installations (Carl Funke and Gneisenau mines) before ultimately building the Hansa Hydropit. Consortium gathered by Ruhrkohle to develop the Hansa mines: TESCO--Hungarian technical (pipefeeder) consultants. Bergbau AG--mine planning consultants. SIEMAG Transplan GmbH--involved in plant design, procurement and manufacture. Ruhrkohle also supports R&D efforts in coarse-particle hydrotransport applications to shaftborers, drift drivers, and waste-rock stowage. [117, 40]

Krupp:

Is doing extensive R&D on coarse-particle hydrotransport (coal, iron ore, quartz sand, etc.). Has constructed, in conjunction with Bergbau-

Forschung GmbH, a large-scale, coarse-particle coal slurry test facility (1979). [44]

Kernforschungsanlage Juelich GmbH (KFA):

Is involved in many Ruhrkohle projects. It made an early study on the conversion of the old longwall Hansa mine into Ruhrkohle's new hydromine [94, 102] and has recently been involved in shaftboring and drift drivage with hydrotransport debris disposal. [117]

Bergbau-Forshung GmbH (B-F):

Often involved in Ruhrkohle contracts. Has studied optimal hydrotransport system layouts. Is involved in headloss theory research in conjunction with Krupp. Has made a study of steep-seam hydromining and hydrotransport funded by the European Coal and Steel Community (SCSC) and Land Nordrhein-Westfalen. [117]

Steinkohlenbergbauverein:

It is the research and development organization for the West German coal industry. Constructed a large test facility in 1977 specifically aimed at coarse-particle coal hydrotransport research. Studies slurries made up of particles as large as 100 mm with densities ranging from 1.2 to 4.5 metric tons/m^3, running through 80 m horizontal pipes from 100 mm to 350 mm in diameter (4", 6", 8", 10", 12", 14"). Its objective is to develop equipment and design methods to the point where full-scale pilot tests will no longer be necessary before new projects can be put on line. [40] Has also done a study on optimization of coal particle dewatering. [44, 82]

University of Hannover:

As of 1977, was planning a large test facility with pipe diameters up to 20". [40]

UNITED STATES OF AMERICA

Coarse-Particle Hydrotransport Installations

Robinson Run Mine:

In 1973 the Consolidation Coal Co. drew together the various components of the coarse-particle hydrotransport system it has been developing since 1970 and made a pilot program in this mine. Coal mined by a continuous miner is fed into a mobile injection hopper. This injector receives the coal in a 300-gallon mixing sump in which a submerged roller crusher reduces the particles to -4". The sump receives water and up to 5.3 tons of coal per minute to form a slurry with a volumetric concentration of 30%. This slurry is fed into a vertical-shaft centrifugal pump driven by a 350 HP motor which pumps 3000 gallons per minute into a 650 foot flexible 10" rubber

rubber pipeline. This flexibility allows the mobile injection hopper unit to follow the movements of the continuous miner for up to 650'. The rubber line then links up to a stationary variable-speed booster pump and a rigid 8" steel line in which the slump travels at a speed of 12 fps, 2,950 feet horizontally and 115 feet vertically to a preparation plant. Design throughput = 600 tph; maximum throughput yet attained = 480 tph. [26, 27, 75, 76, 93]

Loveridge Mine:

Having come on line in 1980, it is the Consolidation Coal Co.'s first full-scale, commercial installation which utilizes coarse-particle hydrotransport. The coal comes from two sources: (1) a continuous miner which feeds into a flexible system such as that described above in the Robinson Run Mine, and (2) a conveyor belt carrying the output from a longwall section and a continuous development section. The flexible continuous mining system, however, operated for only about a year until it was necessary to switch to a different development entry system which was not compatible with the hydrotransport line.

The output (peak rate = 20 tpm) from the conveyor belt, on the other hand, is crushed to -100 mm, made into a slurry, and injected into a 14" pipeline which carries it 1000 feet to the Multiple Feed Station. A dredge reclaimer takes the slurry from this sump to seven centrifugal pumps in series (2 varispeed, 5 fixed) which send it up through a borehole 900 feet to a surface pump station (1 receiving tank and 6 centrifugal pumps, 2 of which are varispeed). The slurry is then transported in a 12" line 2.4 miles over a 450-foot hill to the preparation plant. [77]

Research and Development

The following list contains only the major participants in coal hydrotransport since those having a subsidiary interest are rather numerous.

U.S. Department of Energy; U.S. Bureau of Mines:

The U.S. government started its interest in hydraulic transport in 1950 and since then has produced more than 20 publications and granted 11 contracts. [82] Most prominent in its recent efforts is the Hydraulic Transport Research Facility in Bruceton, Penn. This research facility is the largest of its kind in the world. Its objective is to "obtain engineering data for the design of efficient, economical, and reliable raw coal hydraulic haulage systems from the face to the wash plant." [35] To do so, it runs a variety of size distributions at several concentrations through a "once-through" loop of either 150 mm, 300 mm, or 430 mm line (respectively 194 m, 203 m, and 203 m long) including horizontal, vertical and, excepting the 150 mm line, diagonal sections. The government has been involved in other projects as well. These have included studies in layout optimization, control and automation of coarse hydrotransport systems, coarse coal transport feasibility instrumentation, and development of coal injector and booster pump devices. [35]

Colorado School of Mines Research Institute:

This facility has a variety of loops. Two of the loops, a 100' length of 2" pipe and 600' length of 6" pipe were the ones on which the 1957 American Gilsonite line was designed. [69] Another loop of 6" pipe transports coal up to 2" in diameter through 400 horizontal feet and a 65-foot vertical segment. A final loop sends coal up to 2" in diameter through 400 horizontal feet of 8" pipe. [82]

Consolidation Coal Company:

Consol based its coarse particle transport system on data from two experimental loops: one at Ponca City, OK and the other at its Humphrey #7 mine. The Ponca City line was originally used to obtain design data for Consol's 108-mile Ohio line. Ponca City: this permanent installation has two 38,000 gallon head tanks, a 10" variable speed centrifugal pump which can handle -2.5" solids, 650 feet of 8" pipe, and 200 feet of 8" rubber hose. [26]

Humphrey #7: This temporary facility had a one-pass length of 400 feet and a loop of 700 feet, each including 200 feet of 10" rubber hose. These lines were used primarily to test Consol's mixing sump characteristics and control devices. [26, 76]

Island Creek Coal Company:

Has a coarse-particle pipeline project at its Virginia Pocahontas #3 preparation plant. It is testing a high-pressure feeder which apparently is able to continuously feed -6" coal at a rate of 600 mtph into a 2.07 MPa pipeline capable of transporting coarse coal slurry up to 16 kilometers. Besides the feeder, the project also has a 10" pipeline which runs for 153 meters horizontally and 35 m vertically. The pilot facility was engineered by Kamyr, Inc. [40, 122]

Peabody Coal Company:

Peabody is considering a coarse-particle hydrotransport system which would connect four coal mines to a common preparation plant. Particle size = -50 mm; pipe diameter = 508 mm; volumetric concentration = 25%; weight concentration = 33%; slurry velocity = 4.1 m/sec.; pressure drop = 6 meters/ 100 meters. The total distance covered by the lines may be as much as thirty miles and would use four Kamyr, Inc. rotary feeders, each with a capacity of 910,000 kg/hour (1000 mtph).

GREAT BRITAIN

Research and Development Involvement

British Hydromechanics Research Association:

Contructed a coarse-particle hydrotransport test facility in 1977 with funding provided by manufacturers, users, and the National Coal Board's

Mining Research and Development Establishment (MRDE). Will study pump and pipe wear and flow characteristics of vertical and a variety of inclined lines.

Main loop: 200 mm pipe diameter, 100 meters long.
Branch loops: 250 mm pipe diameter, 50 meters long; 150 mm pipe diameter, 42.5 meters long.
Vertical section: 200 mm pipe, 13 meters high.
All loops can be tilted to create gradients of up to 1:4 [40]

Mining Research and Development Establishment of the National Coal Board:

Is involved in a project developing a system where surface dirt from several mines is pumped out to sea by a coarse-particle slurry line; also involved in the project are: (1) Transport and Road Research Laboratory (to give technical support); (2) British Hydromechanics Research Association (to perform trials in their test facility); and (3) Warren Springs Laboratory (to provide theoretical analysis and small-scale test trials). A pilot project was constructed in 1978 at the Horden Colliery. The following year, 250,000 tons of shale were pumped.

Pipe diameter = 254 mm
Particle size = 38 mm
Pipe length = 1.7 km
Slurry speed = 4 m/sec
Throughput = 140 mtph.

The slurry pipe is lined with basalt, but wear is nevertheless very severe. [40, 90, 117]

Transport and Road Research Laboratory:

Not only is the TRRL a major force in the dirt-disposal project described above, but it is also involved in other hydrotransport research: in 1977 the British Department of Environment contracted it to study the economics and technical feasibility of hydrotransport, particularly that of coarse particles. It was to use a commercially-owned test facility with pipes up to 200 mm in diameter in order to run experiments on a variety of solids up to 40 mm in size. [117]

Warren Spring Laboratory:

Is studying the hydrotransport of limestone aggregates and colliery spoil. Facility characteristics: particle size = -50 mm; pipe diameter = 102 mm and 157 mm; volumetric concentration = -27%; minimum slurry velocity = 2 to 3 m/sec.

AUSTRALIA

Research and Development Involvement

Commonwealth Scientific and Industrial Research Organization (CSIRO):

The Pipeline Engineering and Research Group determined that long-distance, coarse-particle hydrotransport would be the best system for moving coal from Australia's interior to the sea. Is planning a prototype slurry transport test facility to be located at a mine in New South Wales. The test line would have a diameter of 300 mm in which a slurry with a weight concentration of 70% would travel at 1 meter/second. A laboratory model was undergoing tests at CSIRO's Division of Mineral Engineering in 1980. [2]

B.E. Boyle and Associates:

Have developed a 25 mm high-head, rotary ram pump prototype. Would pump a slurry with a weight concentration of 83% made up of coarse particles in a stabilizing, fine slurry. Also has done a feasibility study for a 50-km coarse-particle "stabilized" slurry line from the inland to an off-shore shiploader at Port Kemble which would transport -50 mm coking coal. [8] Another study considered a 276-km pipeline from the Ulan Colliery to the Port of Newcastle. [2]

University of Ashton:

Dr. Elliott experimented (1970) with coarse-particle, "stabilized" slurries at a facility having a 1.61 km pipeline with a diameter of 101.6 mm. Weight concentration = 59%; 100% of particles are -12.7 mm; 39% were less than -3.2 mm; flow is laminar up to 2.1 m/sec and no settlement was noticed at velocity as low as .6 m/sec. [8]

CANADA

Is building a hydrotransport test facility which will handle particles up to 19 mm in diameter.

APPENDIX II

CHRONOLOGY

1850s	1856:	German patent granted to Louis Schwartzkoppf for the first hydraulic dredge.
	Late 1850s:	Use of jet elevators in California gold fields.
1870s	1873:	U.S. patent for jet elevator granted to J. H. Martin.
	1876:	U.S. patent for hydraulic dredging granted to Alexander von Schmidt.
1880s		Hydraulic stowage comes into practice.
		Mine fire extinguished using a slurry line.
	1889:	Patent claim by Wallace C. Andrews for the fine solid/liquid pipeline flow powered by gravity or pumps. (Patent granted in 1891.)
1900s	1900:	High pressure water and solids pumps used for overburden removal in phosphate mines for first time.
	1905:	Donelly gets patent for system which pumps anthracite silt away from coal breaker.
	1906:	Early attempt by Blatch to obtain concentration vs. velocity and concentration vs. headloss correlations.
1910s	1913-1924:	Gilbert Bell runs 660-yard 8" pipeline carrying 5" coal particles travelling at 4' per second. Proposes a similar 130-mile coarse-coal pipeline.
1920s	1920:	Zimmer writes article containing one of first references to hydrohoisting and comments on Bell's slurry line.
	1921:	130-mile, 175-mile, and 322-mile coal slurry lines from Pennsylvania proposed.
	Late 1920s:	1-mile-long anthracite sludge slurry line constructed in Pennsylvania.
1930s		Slurry pipeline transport of phosphate matrix and fine coal refuse becomes common practice.
	1931:	Article by Mikumo--one of first indication that Japanese are involved in slurry transport.
	1935:	Article by Cambrette--one of first indications that French are interested in slurry transport.
		Muchnik starts hydraulic mining experimentation in the U.S.S.R.
	1936:	U.S.B.M. produces article on the hydrotransport of sand and gravel in dredging.
	1937:	Construction of first hydraulic mine in U.S.S.R.
	1939:	Howard, Caldwell, and O'Brien articles on sand transport--associated with dredging.

1940s	1944:	Wuensch article suggests heavy-medium, coarse-particle hydrotransport and hydrohoisting.
	1946:	Wilson article--one of first articles to attempt to quantify slurry transport behavior rather than give anecdotes of past experience.
	1947:	Wuensch article suggests device for hydrohoisting injection. First continuous miner constructed.
	1949:	Clear Springs phosphate matrix line constructed in Florida (5 miles lond). First Soviet hydromine in commerical production.
1950s	1950:	Infamous conveyor fire tragedy spurs British interest in hydrotransport. U.S. Bureau of Mines starts involvement in underground and surface hydrotransport.
	1951:	Consol completes large demonstration facility for the Ohio Pipeline project (17,000 ft. line, -3/8" particles). Inco transport concentrates 7.5 miles in a fine particle slurry line. World's first lockhopper feeder for hoisting installed in Winsconsin zinc mine (injects -4" particles into a 10" line). First U.S.B.M. paper presented--considers a 100-mile coarse particle slurry line.
	1952:	Britain's National Coal Board has symposium on hydrotransport, revealing its interest in hydraulic hoisting, underground hydrotransport, and long-range surface hydrotransport. Durand's headloss correlation was presented (based on data obtained at SOGREAH, a French engineering firm, facilities). Other papers on feeder design and hydrotransport mechanics were advanced. 6-mile French fine-particle coal line constructed (16 mesh, 105 million tpy)--Lorraine Coal Company.
	1953:	Poles start research on lockhopper feeder for hydraulic hoisting. Japanese form an institution to develop hydrotransport and hydrohoisting systems. Completely-automated Soviet hydromine constructed.
	1955:	Several slurry headloss formulae presented (Newitt, Metzner, Spell).
	1956:	Soviets schedule 69 hydraulic mining units to be activated over the 1956-1960 5-year plan. Great Britain starts experimental coal mine, Woodend.
	1957:	Poles complete lockhopper feeder prototype (for -80 mm particles) and install it in the Debiensko colliery. Germans start investigating hydraulic mining. Soviets start a 38 mile, 12" coal pipeline (220 tph, 38% solids concentration, 4.8 fps slurry velocity). Soviet Hydrocoal Institute founded--objective is to develop hydromining and hydrotransport technology.

1957: Anaconda builds 14-mile, fine-particle, gravity slurry line in El Salvador (6" pipe diameter and 2,700 tpd capacity). Construction started on the 72 mile American Gilsonite Pipeline.

1958: Poles design China's Lu-Cja-To mine (hydrohoisting of -1 mm particles). Construction started in 1960. Output = 8,600 metric tons per day.
Czech's develop a feeder for hydraulic hoisting.
German's start hydromining and hydrohoisting experimentation at the Robert Müser mine.
Two Polish hydrocollieries constructed: the Sjerza and the Andaluzja.
Dutch construct an experimental surface hydrotransport facility (DSM Laboratory).
Consol brings its Ohio pipeline on line (108 miles, 1.3 million tpy).
3-mile soot and cinder disposal line contructed in Scotland (-50mm), based on French (Sogreah) hydraulic studies.

1959: Britain develops a pipefeeder and installs it in the experimental Woodend mine. Starts hydromining trials at the Trelewis mine.
Hydrohoist constructed at Devillaine mine by Neyrpic (funded by CERCHAR--French Coal Mining Research Center): -3 1/4" particles, 590' lift, 215' horizontal transport, 50-60 tph.

1960s

1960: Poles develop centrifugal pumps that can handle -80 mm particles in 70 meter hoisting.

1962: Poles develop a high-pressure, non-centrifugal pump (1500 psi).
Germans start the research and development program that ultimately results in the Hansa hydromine in 1977 (4 pilot plants constructed between 1962 and 1970 and two large-scale plants since 1971).
Japanese install experimental pipefeeder equipment in the Furukawa-Yoshima Colliery (100 tph; 1.47 km. long; -50 mm particle size; 253 meters vertical hoisting).

1963: By 1963 there are several hydrotransport mines in the U.S. S.R. and Poland, and hydrohoisting is common in Europe and Australia.
Australians investigate their own hydrohoist system (Unisearch, Ltd.)--3" particles in 4" line.
Mitsui Mining Co. starts hydraulic mining in its Sunagawa Colliery.
Consol's Ohio fine-particle slurry line closed down due to large rail rate reduction. Is still maintained in ready state.

1964: U.S.B.M. constructs an experimental lockhopper feeder installation.
The Rugby limestone fine particle slurry line is started (57 miles; 1.7 million tpy).
Haulage bottleneck resulting from shuttlecar problems becomes apparent in U.S. room-and-pillar coal mining.

1965: Japan places a hydrohoist system (-30 mm particles) in the Sunagawa Colliery (1700' vertical hydrohoisting, 6,600' horizontal hydrotransport).
Germans start work in Dahlhausen Tiefbau experimental mine.
Consol starts investigating coarse-particle hydrotransport as a source of continuous haulage.

1966: Consol obtains a patent for continuous hydraulic haulage.
U.S.B.M. constructs lockhopper test facility.
Canada announces plans for long-range pipelines.
Three large Soviet hydromines come on line: Inskaya #2, Red Army Mine, and the Jubilee mine--all involving horizontal and vertical coarse-particle hydrotransport.

1967: The Savage River concentrate fine-particle slurry line starts (53 miles; 2.25 million tpy).

1968: Japanese revise Sunagawa mine system to transport -.75 mm particles instead of -50 mm particles as a result of extreme wear.

1969: Tesco (A Hungarian engineering firm) develops a pipefeeder to handle -60 mm particles--is eventually installed in Germany's Gneisenau mine.
Consol establishes a formal R&D program for hydraulic continuous haulage.
U.S.B.M. constructs a large-scale, experimental, coarse-particle hydrotransport installation.

1970s

1970: Black Mesa fine-particle coal slurry line goes on line (273 miles; 4.8 million tpy).

1971: Calveras fine-particle limestone slurry line starts (17 miles, 1.5 million tpy).
Germany tests hydromining and hydrohoisting in its Gneisenau mine using two Tesco pipefeeders (-60 mm particles in a 200 mm pipe up a 700 m shaft).
Waipipi fine-particle iron sand pipeline starts (6 miles; 1 million typ).

1972: Japan has six hydromines by 1972.
Ruhrkohle experiments with hydromining at the Carl Funke mine (-1 mm coal particles transported 5 km horizontally and 700 m vertically).
Bougainville fine-particle, copper-concentrate pipeline comes on line (17 miles, 1 million tpy).
Irian Jaya fine-particle copper concentrate pipeline comes on line (69 miles, .3 million tpy).

1973: U.S.B.M. starts continuous haulage R & D program by contracting feasibility study to the Colorado School of Mines Research Institute.
Coarse-particle coal slurry pumping starts at Consol's Robinson Run mine.

1974: Pena Colorado fine-particle iron concentrate pipeline comes on line (28 miles, 1.8 million tpy).
Pinto Valley fine-particle copper concentrate pipeline comes on line (11 miles, .4 million tpy).

1976: Australia's Broken Hill Proprietary studies coarse-particle, long-distance, coking coal hydrotransport and its effect on size and coking quality.
Las Truchas fine-particle iron concentrate pipeline comes on line (17 miles, 1.3 million tpy).
Sierra Grande fine-particle iron concentrate pipeline comes on line (20 miles, 2.1 million tpy).

1977: Ruhrkohle's Hansa hydromine comes on line.

1978: Hydro-shaftboring R&D started at Gneisenau.

1979: U.S.D.O.E.'s Hydraulic Transport Research Facility comes on line.

1980: Consol's Loveridge mine pumps its first coal.
Hansa Hydromine closes down.

APPENDIX III

MODIFIED PROGRAM OUTPUT

The output of the modified Transflux program uses a somewhat esoteric nomenclature in some areas. The explanatory notes below will help clarify some of the confusing points.

1. Economic Input Data Section. This section displays the data which is used to calculate the costs shown later in the program. The Transflux report collected cost data, ran regressions on them, and came up with cost functions of the form: unit cost = Ax^B. A and B are fixed constants whose values depend on the type of equipment being considered and "x" is a variable representing the size or capacity of the equipment. The economic input data section, therefore, shows the A and B values, the lifetime, and the price escalation factor (used to update the cost information) for each piece of equipment.

2. Degrading Factor For Slurry. Centrifugal pumps are less efficient when pumping slurry than water. To adjust their specification efficiencies, the program uses the equation:

$$HR = \frac{\text{Head Pumping Slurry}}{\text{Head Pumping Water}} = 1-K\,\frac{C_v}{.2}$$

where C_v is the concentration by volume and K is the "degrading factor."

3. Velocity-Diameter Conversion Factor. This is the constant used in the slurry velocity equation $V_S = 2.516\ (D)^{.5}$

4. Surge and Initial Slurry Flow at the Face. In this section, feeder-breaker and slurry-mixer power consumption data were given as zero since these units were assumed to be equivalent to a conventional system's stageloader and thus not included in the cost analysis.

5. Surge Capacity. All pumps have a small surge tank feeding their inlet. In some legs, however, a large surge tank is included in the design to smooth out variations in the input flow so that a steady output flow can be created. If no such surge tank is included in the leg in question, the program will give an output figure of "0" under the surge capacity heading. If such a surge tank is necessary, however, its capacity will be given under the heading. If any water is added at these surge tanks to dilute the slurry, the magnitude of the inflow will appear under the "Added Water" title.

6. Pump: Type No. The Transflux program considers eight different types of slurry pumps. The head-throughput characteristics of all these pumps are plotted and divided into sections to form a carpet plot. The computer then selects that section on the carpet plot which fits the system characteristic best. The number of this section is then printed in the output under the "Type No" heading.

7. Surge, Slurry, Pump, and Pipe Specifications at the Shaft, Section #1, Leg #1. This section describes the hydrohoisting leg of the slurry system and is different from the others in that it has seven pumps instead of one. The costs quoted in the output are for all seven pumping units as a group; the horsepower, power consumption, max. power demand, power consumed per shift, and dynamic head, however, are per-pump figures.

8. Final Output At Top of the Shaft. This section gives the total cost per ton figures and breaks it down into its components. It must be noted that if one adds up the total variable costs per ton of each of the legs, the resulting figure does not equal the total variable cost per ton given in this section. This is due to the fact that the cost analysis in the first (face) leg does not include the cost of water consumed, but this cost is included in the final section's variable cost per ton figure.

COAL SLURRY HAULAGE EVALUATION PROGRAM

INPUT DATA

TYPE 4 (for diagram, see page 59)

```
LONGWALL DATA
SEAM THICKNESS (FT)..............................=   5.5Ø
LONGWALL FACE LENGTH (FT)........................=5ØØ.ØØ
SHEARER DRUM PENETRATION (FT) ...................=   2.5Ø
SHEARER SPEED (FT/MN) ...........................=  15.ØØ
FEEDER BREAKER SURGE CAPACITY (TONS).............=   9.ØØ
FEEDER BREAKER OUTPUT RATE (TONS/MIN)............=  2Ø.ØØ


        LONGWALL MINING CYCLE FOR    5.ØØ  FOOT FACE ADVANCE

CUTTING TIME FOR FULL FACE (MIN).........= 65.71
SHEARER FIRST REPOSITION TIME (MIN)......=  1.ØØ
SHEARER SECOND REPOSITION TIME (MIN).....=  1.ØØ
TOTAL CYCLE TIME (MIN)...................= 67.71
SHEARER CUTTING RATE (TONS/MIN)..........=  8.89

              ECONOMIC INPUT DATA

        PARAMETERS OF INVESTMENT EQUATIONS :
   FLEXIBLE PIPES :
      A...................................= Ø.1ØØ95E Ø4
      B...................................= Ø.2Ø45ØE ØØ
      LIFE...(YEARS)......................= Ø.5ØØØØE Ø1
      PRICE ESCALATION FACTOR.............= Ø.15461E Ø1
   RIGID PIPES :
      A...................................= Ø.5Ø378E Ø1
      B...................................= Ø.99429E ØØ
      LIFE...(YEARS)......................= Ø.5ØØØØE Ø1
      PRICE ESCALATION FACTOR.............= Ø.15461E Ø1
   CENTRIFUGAL SLURRY PUMPS :
      A...................................= Ø.13244E Ø1
      B...................................= Ø.11223E Ø1
      LIFE...(YEARS)......................= Ø.15ØØØE Ø2
      PRICE ESCALATION FACTOR.............= Ø.15392E Ø1
   CENTRIFUGAL SLURRY PUMPS FOR LIFT :
      A...................................= Ø.63426E Ø2
      B...................................= Ø.7483ØE ØØ
      LIFE...(YEARS)......................= Ø.15ØØØE Ø2
      PRICE ESCALATION FACTOR.............= Ø.14485E Ø1
   VARIABLE SPEED DRIVES :
      A...................................= Ø.4476ØE Ø4
      B...................................= Ø.3Ø88ØE ØØ
      LIFE...(YEARS)......................= Ø.15ØØØE Ø2
      PRICE ESCALATION FACTOR.............= Ø.15392E Ø1
   ELECTRIC MOTORS :
      A...................................= Ø.1113ØE Ø3
      B...................................= Ø.89168E ØØ
      LIFE...(YEARS)......................= Ø.15ØØØE Ø2
      PRICE ESCALATION FACTOR.............= Ø.15392E Ø1
   VALVES :
      A...................................= Ø.2Ø82ØE Ø3
      B...................................= Ø.15445E Ø1
      LIFE...(YEARS)......................= Ø.15ØØØE Ø2
      PRICE ESCALATION FACTOR.............= Ø.15392E Ø1
   SURGE TANKS :
      A...................................= Ø.37983E Ø2
      B...................................= Ø.5567ØE ØØ
      LIFE...(YEARS) :
         AT PANELS .......................= Ø.5ØØØØE ØØ
```

```
        AT SUBMAINS......................= Ø.5ØØØØE Ø1
        AT MAINS AND SHAFT BOTTOM.......= Ø.2ØØØØE Ø2
    PRICE ESCALATION FACTOR.............= Ø.16169E Ø1

NO. OF WORK DAYS/YEAR...................= Ø.22ØØØE Ø3
NO. OF SHIFTS/DAY.......................= Ø.3ØØØØE Ø1
TOT. TIME FOR SHIFT CHANGES AND
LUNCH BREAK PER SHIFT...(MIN)...........= Ø.4ØØØØE Ø2
ESTIMATED POWER RATE ($/KWHR)...........= Ø.4ØØØØE-Ø1
WATER HANDLING COST($/1ØØØ GALS)........= Ø.4ØØØØE-Ø1
WATER PURCHASE COST ($/1ØØØ GALS).......= Ø.2ØØØØE ØØ
MAINTENANCE CHARGE FACTOR...............= Ø.1ØØØØE ØØ
```

AVAILABLE PIPES
INSIDE DIAMETERS IN INCHES

4.Ø3	6.Ø6	7.98	1Ø.Ø2	12.ØØ
13.25	15.25	17.25	19.25	23.25
29.ØØ	36.ØØ			

INPUT CONSTANTS

```
ROUGHNESS PARAMETER..............................=  Ø.ØØØ15
ROUGHNESS FACTOR FOR FLEXIBLE PIPE...............=  2.ØØ
KINEMATIC VISCOSITY.(SQ.FT/SEC)..................=  Ø.ØØØØ1216
GRAVITATIONAL ACCELERATION..(FT/SQ.SEC.).........= 32.17Ø
DEGRADING FACTOR FOR SLURRY......................=  Ø.215
MAX. LIMIT FOR SOLIDS CONC. (IN WEIGHT) IN SLURRY=  Ø.45Ø
RATIO OF MIN. PERMISSIBLE VEL. TO CRITICAL VEL...=  1.Ø1Ø
VELOCITY-DIAMETER CONVERSION FACTOR..............=  2.516
RATIO OF MAX. PERMISSIBLE VEL. TO CRITICAL VEL...=  2.ØØØ
```

2 SECTION MINE , TYPE 4

SURGE AND INITIAL SLURRY FLOW AT THE FACE

```
SURGE CAPACITY :
FEEDER BREAKER (TONS)........................=  9.ØØ
TOTAL SURGE (TONS)...........................=  9.ØØ
SURGE DISCHARGE RATE (TPM)...................=  8.62
SURGE DISCHARGE DURATION/CYCLE (MIN).........= 67.76
NUMBER OF CYCLES/SHIFT.......................=  6.5Ø
POWER :
FEEDER BREAKER MAX. DEMAND (KW)..............=  Ø.ØØ
FEEDER BREAKER POW. CONS./CYCLE (KWH)........=  Ø.ØØ
SLURRY MIXER MAX. DEMAND (KW)................=  Ø.ØØ
SLURRY MIXER POW. CONS./CYCLE (KWH)..........=  Ø.ØØ
```

INITIAL SLURRY CHARACTERISTICS

```
FLOW RATE (GPM) ..................................=  4Ø44.187
SOLIDS CONCENTRATION IN WEIGHT ...................=     Ø.45Ø
SOLIDS CONCENTRATION IN VOLUME ...................=     Ø.375
WATER CONSUMPTION (GPM) ..........................=  2526.177
```

2 SECTION MINE , TYPE 4

```
SLURRY ,PUMP, AND PIPE SPECIFICATIONS
BETWEEN THE SLURRY MIXER AND THE BOOSTER PUMP
SLURRY:
FLOW RATE  (GPM)...................=  4044.187
FLOW DURATION/CYCLE (MIN).........=    67.758
NUMBER OF CYCLES/SHIFT............=     6.498
WATER CONSUMPTION  (GPM)..........=  2526.177
SOLIDS CONC. IN WT. ..............=     0.450
SOLIDS CONC. IN VOL...............=     0.375
VELOCITY  (FT/SEC)................=     9.410
SP. GRAVITY.......................=     1.136
PUMP:
TYPE NO...........................=  23
HORSE POWER.......................=   238.463
POWER CONSUMPTION (KWH/TON.COAL)..=     0.344
POW. CONS. /CYCLE (KWH)...........=   200.894
MAX. POW. DEMAND (KW).............=   177.893
DYNAMIC HEAD  (FT.)...............=    95.919
EFFICIENCY........................=     0.467
PUMP COST ($).....................= 22764.539
PUMP VAR. DRIVE COST ($)..........= 40018.242
COST OF VALVES ($)................= 34679.055
MOTOR COST  ($)...................= 27548.660
PIPES:
INSIDE DIAMETER (INCH)
    FLEXIBLE PIPE.................=    13.250
    RIGID PIPE....................=    13.250
LENGTH  (FT.)
    FLEXIBLE PIPE.................=    50.000
    RIGID PIPE....................=  2450.000
COST:
    FLEXIBLE PIPE ($).............= 23448.695
    RIGID PIPE ($)................=249145.438
NO OF PARALLEL SYSTEMS AT FACE....=     1.000
TOTAL HEAD FOR WATER  (FT)........=    44.012
TOTAL HEAD FOR SLURRY (FT)........=    57.215
COST PER TON OF COAL ($)..........=     0.041

 2  SECTION MINE   , TYPE   4


SURGE, SLURRY, PUMP, AND PIPE SPECIFICATIONS
          AT PANEL ENTRY   ,  SECTION NO:   1


SURGE :
SURGE CAPACITY (1000 GALS.)..=      0.000
COST OF SURGE ($)............=      0.000
INPUT TO SURGE:
SLURRY INFLOW..(GPM).........=   4044.187
INPUT DURATION/CYCLE (MIN)...=     67.758
NUMBER OF CYCLES/SHIFT.......=      6.498
OUTPUT :
COAL THROUGHPUT/SHIFT (TONS).=   3797.211
SLURRY OUTFLOW (GPM).........=   4044.187
ADDED WATER...(GPM)..........=      0.000
FLOW DURATION/CYCLE..(MIN)...=     67.758
NUMBER OF CYCLES/SHIFT.......=      6.498
OUTGOING SLURRY :
SOLIDS CONC. IN WT...........=      0.450
SOLIDS CONC. IN VOL..........=      0.375
VELOCITY (FT/SEC)............=      9.410
SP GRAVITY...................=      1.136
```

```
PUMP:
TYPE NO......................=23
COST OF PUMPS ($)............= 22764.539
COST OF VAR. DRIVES ($)......= 39991.016
COST OF MOTORS ($)...........= 27494.555
COST OF VALVES ($)...........= 34679.055
NO. OF PUMPS SERIAL..........= 1
HORSE POWER..................=   237.938
POW. CONS. (KWH/TON. COAL)...=     0.343
MAX. POWER DEMAND (KW).......=   177.502
POW. CONS./SHIFT (KWH).......=  1302.516
DYNAMIC HEAD (FT)............=    95.725
EFFICIENCY...................=     0.467
PIPE :
INSIDE DIAMETER (INCH).......=    13.250
LENGTH (FT)..................=  2500.000
COST OF PIPE ($).............=254230.031
TOTAL HEAD FOR WATER (FT)....=    43.923
TOTAL HEAD FOR SLURRY (FT)...=    57.099
COST PER TON OF COAL ($) ....=     0.040

 2  SECTION MINE   , TYPE   4


SURGE, SLURRY, PUMP, AND PIPE SPECIFICATIONS
          AT PANEL ENTRY   ,  SECTION NO:   2


SURGE :
SURGE CAPACITY (1000 GALS.)..=     0.000
COST OF SURGE ($)............=     0.000
INPUT TO SURGE:
SLURRY INFLOW..(GPM).........=  4044.187
INPUT DURATION/CYCLE (MIN)...=    67.758
NUMBER OF CYCLES/SHIFT.......=     6.498
OUTPUT :
COAL THROUGHPUT/SHIFT (TONS).=  3797.211
SLURRY OUTFLOW (GPM).........=  4044.187
ADDED WATER...(GPM)..........=     0.000
FLOW DURATION/CYCLE..(MIN)...=    67.758
NUMBER OF CYCLES/SHIFT.......=     6.498
OUTGOING SLURRY :
SOLIDS CONC. IN WT...........=     0.450
SOLIDS CONC. IN VOL..........=     0.375
VELOCITY (FT/SEC)............=     9.410
SP GRAVITY...................=     1.136
PUMP:
TYPE NO......................=23
COST OF PUMPS ($)............= 22764.539
COST OF VAR. DRIVES ($)......= 39991.016
COST OF MOTORS ($)...........= 27494.555
COST OF VALVES ($)...........= 34679.055
NO. OF PUMPS SERIAL..........= 1
HORSE POWER..................=   237.938
POW. CONS. (KWH/TON. COAL)...=     0.343
MAX. POWER DEMAND (KW).......=   177.502
POW. CONS./SHIFT (KWH).......=  1302.516
DYNAMIC HEAD (FT)............=    95.725
EFFICIENCY...................=     0.467
PIPE :
INSIDE DIAMETER (INCH).......=    13.250
LENGTH (FT)..................=  2500.000
COST OF PIPE ($).............=254230.031
TOTAL HEAD FOR WATER (FT)....=    43.923
TOTAL HEAD FOR SLURRY (FT)...=    57.099
COST PER TON OF COAL ($) ....=     0.040

 2  SECTION MINE   , TYPE   4
```

```
SURGE, SLURRY, PUMP, AND PIPE SPECIFICATIONS
AT SUBMAIN,  SECTION NO:  1  LEG NO:  1


SURGE :
SURGE CAPACITY (1000 GALS.)..=     0.000
COST OF SURGE ($)............=     0.000
INPUT TO SURGE:
SLURRY INFLOW..(GPM).........=  8088.374
INPUT DURATION/CYCLE (MIN)...=    67.758
NUMBER OF CYCLES/SHIFT.......=     6.498
OUTPUT :
COAL THROUGHPUT/SHIFT (TONS).=  7594.423
SLURRY OUTFLOW (GPM).........=  8088.374
ADDED WATER...(GPM)..........=     0.000
FLOW DURATION/CYCLE..(MIN)...=    67.758
NUMBER OF CYCLES/SHIFT.......=     6.498
OUTGOING SLURRY :
SOLIDS CONC. IN WT...........=     0.450
SOLIDS CONC. IN VOL..........=     0.375
VELOCITY (FT/SEC)............=    11.104
SP GRAVITY...................=     1.136
PUMP:
TYPE NO......................=36
COST OF PUMPS ($)............= 49556.953
COST OF VAR. DRIVES ($)......= 47287.258
COST OF MOTORS ($)...........= 44607.586
COST OF VALVES ($)...........= 52122.547
NO. OF PUMPS SERIAL..........= 1
HORSE POWER..................=   409.407
POW. CONS. (KWH/TON. COAL)...=     0.295
MAX. POWER DEMAND (KW).......=   305.418
POW. CONS./SHIFT (KWH).......=  2241.170
DYNAMIC HEAD (FT)............=    76.826
EFFICIENCY...................=     0.435
PIPE :
INSIDE DIAMETER (INCH).......=    17.250
LENGTH (FT)..................=  2000.000
COST OF PIPE ($).............=264384.313
TOTAL HEAD FOR WATER (FT)....=    35.251
TOTAL HEAD FOR SLURRY (FT)...=    45.826
COST PER TON OF COAL ($) ....=     0.026

 2  SECTION MINE  , TYPE   4


SURGE, SLURRY, PUMP, AND PIPE SPECIFICATIONS
AT MAIN , SECTION NO:  1  LEG NO:  1


SURGE :
SURGE CAPACITY (1000 GALS.)..=   148.294
COST OF SURGE ($)............= 46455.422
INPUT TO SURGE:
SLURRY INFLOW..(GPM).........=  8088.374
INPUT DURATION/CYCLE (MIN)...=    67.758
NUMBER OF CYCLES/SHIFT.......=     6.498
OUTPUT :
COAL THROUGHPUT/SHIFT (TONS).=  7594.420
SLURRY OUTFLOW (GPM).........=  7688.011
ADDED WATER...(GPM)..........=   268.901
FLOW DURATION/CYCLE..(MIN)...=   240.000
NUMBER OF CYCLES/SHIFT.......=     2.000
OUTGOING SLURRY :
SOLIDS CONC. IN WT...........=     0.436
SOLIDS CONC. IN VOL..........=     0.362
```

```
VELOCITY (FT/SEC)............=     10.554
SP GRAVITY...................=      1.131
PUMP:
TYPE NO......................=56
COST OF PUMPS ($)............= 74291.484
COST OF VAR. DRIVES ($)......= 60746.055
COST OF MOTORS ($)...........= 91939.016
COST OF VALVES ($)...........= 52122.547
NO. OF PUMPS SERIAL..........= 1
HORSE POWER..................=   921.303
POW. CONS. (KWH/TON. COAL)...=     0.724
MAX. POWER DEMAND (KW).......=   687.292
POW. CONS./SHIFT (KWH).......=  5498.337
DYNAMIC HEAD (FT)............=   178.381
EFFICIENCY...................=     0.425
PIPE :
INSIDE DIAMETER (INCH).......=    17.250
LENGTH (FT)..................=  5280.000
COST OF PIPE ($).............=697974.625
TOTAL HEAD FOR WATER (FT)....=    84.387
TOTAL HEAD FOR SLURRY (FT)...=   108.920
COST PER TON OF COAL ($) ....=     0.065

 2  SECTION MINE  , TYPE   4


SURGE, SLURRY, PUMP, AND PIPE SPECIFICATIONS
AT MAIN , SECTION NO:  1  LEG NO:  2


SURGE :
SURGE CAPACITY (1000 GALS.)..=     0.000
COST OF SURGE ($)............=     0.000
INPUT TO SURGE:
SLURRY INFLOW..(GPM).........=  7688.011
INPUT DURATION/CYCLE (MIN)...=   240.000
NUMBER OF CYCLES/SHIFT.......=     2.000
OUTPUT :
COAL THROUGHPUT/SHIFT (TONS).=  7594.420
SLURRY OUTFLOW (GPM).........=  7688.011
ADDED WATER...(GPM)..........=     0.000
FLOW DURATION/CYCLE..(MIN)...=   240.000
NUMBER OF CYCLES/SHIFT.......=     2.000
OUTGOING SLURRY :
SOLIDS CONC. IN WT...........=     0.436
SOLIDS CONC. IN VOL..........=     0.362
VELOCITY (FT/SEC)............=    10.554
SP GRAVITY...................=     1.131
PUMP:
TYPE NO......................=56
COST OF PUMPS ($)............= 74291.484
COST OF VAR. DRIVES ($)......= 60746.055
COST OF MOTORS ($)...........= 91939.016
COST OF VALVES ($)...........= 52122.547
NO. OF PUMPS SERIAL..........= 1
HORSE POWER..................=   921.303
POW. CONS. (KWH/TON. COAL)...=     0.724
MAX. POWER DEMAND (KW).......=   687.292
POW. CONS./SHIFT (KWH).......=  5498.337
DYNAMIC HEAD (FT)............=   178.381
EFFICIENCY...................=     0.425
PIPE :
INSIDE DIAMETER (INCH).......=    17.250
LENGTH (FT)..................=  5280.000
COST OF PIPE ($).............=697974.625
TOTAL HEAD FOR WATER (FT)....=    84.387
TOTAL HEAD FOR SLURRY (FT)...=   108.920
COST PER TON OF COAL ($) ....=     0.064
```

```
 2  SECTION MINE  , TYPE   4


SURGE, SLURRY, PUMP, AND PIPE SPECIFICATIONS
AT THE SHAFT, SECTION NO:  1  LEG NO:  1

SURGE :
SURGE CAPACITY (1ØØØ GALS.)..=      Ø.ØØØ
COST OF SURGE ($)............=      Ø.ØØØ
INPUT TO SURGE:
SLURRY INFLOW..(GPM).........=   7688.Ø11
INPUT DURATION/CYCLE (MIN)...=    24Ø.ØØØ
NUMBER OF CYCLES/SHIFT.......=      2.ØØØ
OUTPUT :
COAL THROUGHPUT/SHIFT (TONS).=   7594.42Ø
SLURRY OUTFLOW (GPM).........=  1Ø113.883
ADDED WATER...(GPM)..........=   2425.872
FLOW DURATION/CYCLE..(MIN)...=    24Ø.ØØØ
NUMBER OF CYCLES/SHIFT.......=      2.ØØØ
OUTGOING SLURRY :
SOLIDS CONC. IN WT...........=      Ø.341
SOLIDS CONC. IN VOL..........=      Ø.275
VELOCITY (FT/SEC)............=     11.149
SP GRAVITY...................=      1.1ØØ
PUMP:
TYPE NO......................=57
COST OF PUMPS ($)............=6385Ø2.375
COST OF VAR. DRIVES ($)......=452377.375
COST OF MOTORS ($)...........=769537.125
COST OF VALVES ($)...........=432221.188
NO. OF PUMPS SERIAL..........= 7
HORSE POWER..................=  1125.8Ø9
POW. CONS. (KWH/TON. COAL)...=     6.193
MAX. POWER DEMAND (KW).......=  5878.972
POW. CONS./SHIFT (KWH).......= 47Ø31.766
DYNAMIC HEAD (FT)............=   2Ø6.8Ø3
EFFICIENCY...................=     Ø.516
PIPE :
INSIDE DIAMETER (INCH).......=    19.25Ø
LENGTH (FT)..................=  1ØØØ.ØØØ
COST OF PIPE ($).............=147426.5ØØ
TOTAL HEAD FOR WATER (FT)....=   145.Ø84
TOTAL HEAD FOR SLURRY (FT)...=   145.59Ø
COST PER TON OF COAL ($) ....=     Ø.298

 2  SECTION MINE  , TYPE   4


FINAL OUTPUT AT TOP OF THE SHAFT

COAL PRODUCTION (TONS/SHIFT)........=  7594.423
SLURRY OUTPUT RATE.(GPM)............= 1Ø113.883
MAX.  WATER DEMAND..... (GPM).......=  7747.127
TOTAL WATER CONS. (1ØØØ GAL/SHIFT)..=  3517.955
WATER RECLAMATION RATE (%)..........=    88.ØØØ
MAKE-UP WATER PRICE ($/1ØØØ GAL.)...=     Ø.2ØØ
COST OF HANDLING WATER ($/1ØØØ GAL).=     Ø.Ø4Ø
ESTIMATED POWER RATE ($/KWHR).......=     Ø.Ø4Ø
CALCULATED POWER RATE ($/KWHR)......=     Ø.Ø4Ø
TOTAL POWER DEMAND (KW).............=  8269.764
TOTAL POWER CONS./SHIFT (KWH).......= 65485.414
SLURRY CHARACTERISTICS:
SOLIDS CONC. IN WT..................=     Ø.341
SOLIDS CONC. IN VOL.................=     Ø.275
SPECIFIC GRAVITY....................=     1.1ØØ
COST ESTIMATION :
```

```
TOTAL VAR. INVESTMENT (1000. $).....=  6452.151
VAR. CAPITAL & MAINT. COST ($/TON)..=     0.178
POWER COST/TON...($)................=     0.345
WATER COST ($/TON)..................=     0.030
TOTAL VAR. COST ($/TON).............=     0.552
```

APPENDIX IV

COSTING

Conventional Haulage System

I. Panel Conveyors:

Haulage Requirements

Peak Load = 475 tph/longwall unit
Haulage Distance < 5000'
22 Operating hours/day
220 Operating days/year

Cost Data

Conveyor Equipment Life = 20 yrs.
Conveyor Belt Life = 5 yrs.
Power Cost = $.04/kw-hr (1981)

Conveyor Characteristics

Belt Width = 36 inches
Belt Speed = 350 feet per minute
Troughing Angle = 35°
Capacity = 550 tph
Belt Material = Medium strength, 4-ply RMA 120 Grade II, Vulcanized Splice, 1/8" Top Cover, 1/16" Bottom Cover.
Belt Drive = 150 Horsepower
Extensible
(Sources: SME Handbook [64], U.S.B.M. Information Circulars [31, 32, 59])

Capital Costs (For One Panel):

Conveyor Equipment (deck, idlers, drive, terminals etc.)	$352,200
Conveyor Belt	302,800
Conveyor Fire Protection, Two-Inch Water Line, Extension Component	83,900
Total	$738,900 (1981)

Power Costs (For One Panel):

(150 HP)(.746 kw/HP)(22 hrs/day)(220 days/yr)($.04/kw-hr) = $21,700/yr (1981

Maintenance Costs (For One Panel):

(10%)(Total Capital Costs) = (.10)($738,900) = $73,900/year (1981).

The longwall mine has two panels so the above costs are multiplied by two:

Total Equipment Cost = $872,200
Total Belt Cost = $604,000
Total Power Cost = $43,400/yr
Total Maintenance Cost = $147,800/yr (1981 prices)

(Sources: SME Handbook [64], U.S.B.M. Information Circulars [31, 32, 59])

II. Submain Conveyor:

<u>Haulage Requirements</u>

Peak Load = 950 tph
Haulage Distance = 2000'
22 Operating hours/day
220 Operating days/yr.

<u>Conveyor Characteristics</u>

Belt Width = 48 inches
Belt Speed = 400 feet per minute
Troughing Angle = 35^{o}
Capacity = 1,155 tph
Belt Material = Medium Strength
Belt Drive = 150 Horsepower

<u>Cost Data</u>

Conveyor Equipment Life = 20 yrs.
Belt Life = 5 yrs.
Power Cost = $.04/kw-hr.

Capital Cost:

Conveyor Equipment	$251,000
Conveyor Belt	194,600
Conveyor Fire Protection, Two-Inch Water Line	30,000
Total	$476,000 (1981)

Power Costs:

(150 HP)(.746 kw/HP)(22 hrs/day)(220 days/yr)($.04/kw-hr) = $21,700/yr (198

Maintenance Costs:

(10%)(Total Capital Costs) = (.10)($476,000) = $47,600/yr (1981)

(Sources: SME Handbook [64], U.S.B.M. Information Circulars [31, 32, 59])

III. Main Conveyor:

<u>Haulage Requirements</u>

Peak Load = 950 tph
Haulage Distance = 2 miles
22 Operating hours/day
220 Operating days/yr.

<u>Conveyor Characteristics</u>

Distance Covered by Three 4000-Foot Conveyors, Each Equipped with 200 HP Drive.
Belt Width = 48 inches
Belt Speed = 400 fpm
Troughing Angle = 35^{o}
Capacity = 1,155 tph
Belt Material = Medium Strength
(Sources: SME Handbook [64], Link's Feasibility Study [67])

<u>Cost Data</u>

Conveyor Equipment Life = 20 yrs.
Conveyor Belt Life = 5 yrs.
Power Cost = $.04/kw-hr.

Capital Costs (For All Three Belts):

Conveyor Equipment	$1,306,400
Conveyor Belt	1,167,900
Conveyor Fire Protection and Two-Inch Water Line	171,400
Total	$2,645,700 (1981)

Power Costs (For All Three Belts):

(3)(200 HP)(.746 kw/HP)(22 hrs/day)(220 days/yr)($.04/kw-hr) = $86,700/yr. (1981)

Maintenance Costs (For All Three Belts):

(10%)(Total Capital Costs) = (.10)($2,645,700) = $264,600/yr (1981)

(Sources: SME Handbook [64], Link's Feasibility Study [67])

IV. Shaft-Bottom Coal Loading Pocket:

Scale-Up of Loading Pocket in Link's Feasibility Study [67]
Loading Pocket Capacity = (5,016,000 tpy/1,100,000 tpy)(360 tons)
= 1650 ton capacity
Excavation Volume = (1650 tons of coal)(2000 lbs/ton)(1 ft^3/83 lbs coal)(1 yd^3/27 ft^3) = 1470 yd^3
Excavation Cost, Concrete Lining, and Chute Gates = $323,000 (1981)
Maintenance Cost = (10%)(Capital Costs) = (.10)(323,000) = $32,300/yr (1981)

(Sources: Link's Feasibility Study [67] and Underground Mining Methods Handbook*)

V. Production/Service Shaft:

Diameter = 26' ID
Depth = 1000'
Excavation Costs = $1,444,000
Concrete Lining = $436,000
Pocket Excavation = $70,000
Station Support = $58,000
Pocket Concrete = $24,000

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*William A. Hustrulid (Ed.), Underground Mining Methods Handbook, Society of Mining Engineers, Littleton, CO, 1982, 1754pp.

Steel = $200,000
Shaft Support = $850,000
Mobilization = $400,000
Collar Installation = $300,000.
Total Production/Service Shaft Cost = $3,782,000 (1977)
= $5,742,000 (1981)

(Source: Underground Mining Methods Handbook)

VI. Service and Production Hoists:

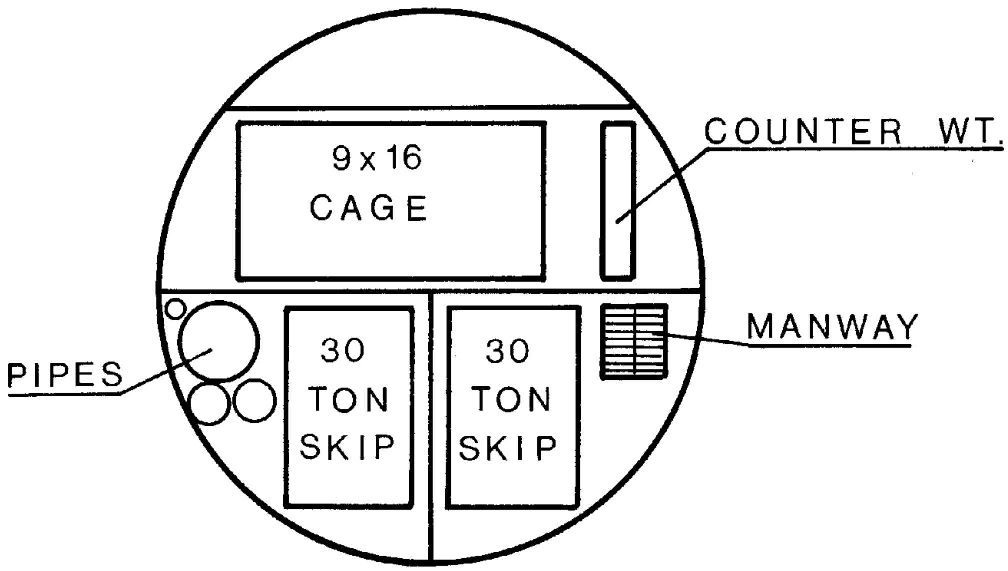

Figure 13. Conventional System's Shaft Cross-Section.

Service Hoist:
Counter-Weighted Friction Hoist
Hoist Power = 500 HP
Hoist Capacity = 9'x16' cage

Capital Cost = $1,154,000 (1981)
Maintenance Cost = (10%)(Capital Cost) = $115,000/yr (1981).
Power Cost = (500 HP)(.746 kw/HP)(22 hrs/day)(220 days/yr)($.04/kw-hr)
= $72,200/yr (1981)

Production Hoist:
Two-Skip Friction Hoist
Hoist Power = 2,500 HP
Skip Capacity = 2x30-ton skips

Capital Costs:	
Main Hoist (Friction) 2500 HP (1860 kw)	$1,118,000
Hoist/Tail Ropes	123,000
Rope Handling Equipment	6,000
2xSkips (30 t)	112,000
Rope Hardware	27,000
Dumping Scrolls	13,000
Measuring Pockets and Scale	141,000
Sheaves	33,000
Spill Pocket	10,000
	$1,583,000 (1977)
	$2,293,000 (1981)

Power Costs:

(2,500 HP)(.746 kw/HP)(22 hrs/day)(220 days/yr)($.04/kw-hr)
= $361,000/yr (1981)

Maintenance Costs:

(10%)(Capital Costs) = $229,300/yr (1981)

Total Hoist Capital Cost = 3,447,000 (1981)
Total Hoist Maintenance Cost = $345,000/yr (1981)
Total Hoist Power Cost = $433,300/yr (1981)

(Source: Unadjusted cost data from Underground Mining Methods Handbook)

VII. Power Supply System:

Description:
Power Factor = .85
Primary Substation = 2x3000 kva units 69,000 v/7200 v
Secondary Substations = 1x3000 kva units 7200 v/440 v
2x1000 kva units "
8x300 kva units "
1x200 kva units "
1x150 kva units "

Capital Costs:

Primary Substation Cost	$336,000
Secondary Substation Cost	901,000
Total	$1,237,000 (1981)

Maintenance Costs:
(1%)(Capital Costs) = $12,000/yr (1981

(Source: Unadjusted Costs Found in U.S.B.M. Information Circulars [31, 32, 59])

Slurry Haulage System

I. Panel Conveyors:

Costs obtained from modified Transflux Program (see Appendix III).
Capital Cost = $1,554,000 (1981)
Power Cost = [(1306 kw-hr/shift)(2)+(1303 kw-hr/shift)(2)](3shifts/day)(220 days/yr)($.04/kw-hr) = $138,000/yr (1981)

II. Submain Haulage

Capital Cost = $458,000 (1981)
Power Cost = (2241 kw-hr/shift)(35 shifts/day)(220 days/yr)($.04/kw-hr) = $59,000/yr (1981)

III. Main Haulage

Capital Cost = $1,954,000 (1981)
Power Cost = (10,997 kw-hr/shift)(3 shifts/day)(220 days/yr)($.04/kw-hr) = $290,000/yr (1981)

IV. Hydrohoist

Capital Cost = $2,440,000 (1981)
Power Cost = $1,242,000/yr (1981)

V. Service Hoist

Description:
1000' depth
Cage Size = 9'x20'
Shaft Diameter = 22' ID
Counterbalanced Friction Service Hoist = 600 HP

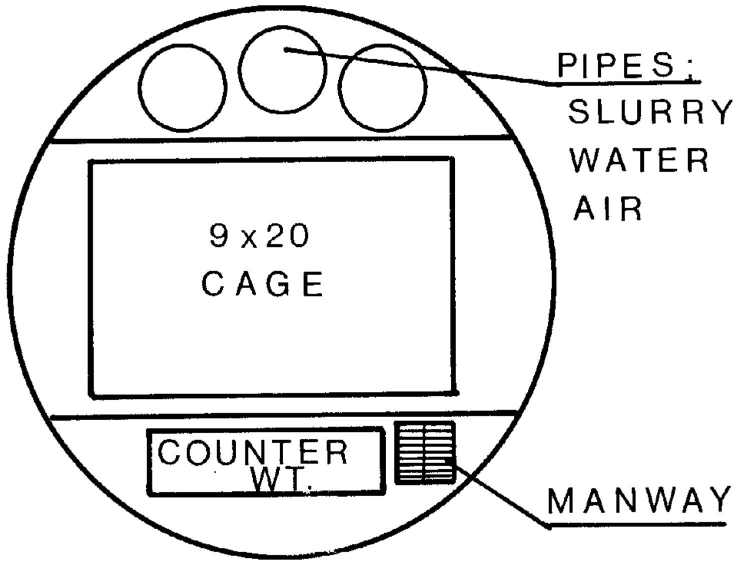

Figure 14. Slurry System's Shaft Cross-Section

Capital Cost:

Main Hoist (Friction, 600 HP)	$779,000 (1977)
Hoist/Tail Ropes	55,000
9'x20' Cage	94,000
Rope Handling Equipment	6,000
Rope Hardware	18,000
Cage Chains	5,000
Sheaves	39,000
	$996,000 (1977)
	$1,448,000 (1981)

Maintenance Cost:

(10%)(Capital Cost) = $144,000/yr (1981)

Power Cost:

(600 HP)(.746 kw/HP)(22 hrs/day)(220 days/yr)($.04/kw-hr)
= 86,700/yr (1981)

(Source: Underground Mining Methods Handbook)

VI. Shaft Bottom Emergency Sump:

Volume = Contents of Vertical Pipe = 2,021 ft^3
Capital Cost = $13,000 (1981)
Maintenance Cost = $1,000/yr (1981)

(Source: Transflux Report [112])

VII. Service Shaft:

Diameter = 22 feet
Depth = 1000'

Capital Costs:

Excavation Costs	$1,051,000 (1977)
Concrete Lining	407,000
Pocket Excavation	55,000
Station Support	52,000
Pocket Concrete	20,000
Steel	200,000
Shaft Support	740,000
Mobilization and Collar Installation	630,000
	$3,155,000 (1977)
	$4,680,000 (1981)

Maintenance Cost = $46,800/yr (1981)

(Source: <u>Underground Mining Methods Handbook</u>)

VIII. Water Supply System:

The water supply system is a duplicate of the slurry system except for the pumping stations. When the first slurry line gets worn and has to be changed, the water line and slurry line are switched. This way the water pipeline is essentially obtained for free since ordinarily the worn slurry lines are discarded.

Capital Costs:

Pipe	-
Pumps and Drives	$326,000 (1981)
Modulating Control Valves	78,000
	$404,000 (1981)

Maintenance Costs:
(5%)($404,000) = $20,000/yr (1981)

Power Costs included in slurry pipeline's power costs.

IX. Power Supply System

Primary Substation = 2x6250 kva units 69000 v/7200
Secondary Substation = 1x7500 kva unit 7200/400 v
4x1000 "
6x300 "
1x200 "
2x150 "

Primary Substation Capital Costs =	$542,000	(1981)
Secondary Substation Capital Costs =	1,162,000	
	$1,704,000	(1981)

Maintenance Cost = (1%)(Capital Cost) = $17,000/yr (1981)

(Source: Unadjusted costs from U.S.B.M. Information Circulars [31, 32, 59])

X. Control and Automation System:

Level of Automation. Lines and pumps are automatically flushed, shut-off, and restarted in reaction to the production status at the face.

Capital Costs:		
Hardware Costs	$313,000	(1977)
Panel Costs	205,000	
	$518,000	(1977)
	$750,000	(1981)

Maintenance Cost = (10%)(Capital Costs) = $75,000/yr (1981)

(Source: Unadjusted costs from Transflux Control and Automation Report [111])

Slurry System Cost Per Ton Calculations
(With Delay Modifications):

Delay Factor Calculations:

$$\frac{\text{Longwall Delay Time}}{\text{Total Time-Travel Time}} = \frac{32{,}764+2{,}860}{175{,}200-14{,}600} = .2218$$

external conveyor = 32,764
external (power) = 2,860

With the conventional system we used a different delay factor of .3375 which took into account delays throughout the entire mine system, both longwall delay times and external delay times. Since in the slurry system we do not know the reliability of the slurry lines, we will take only the longwall delay times into account and then see what level of reliability would be necessary for the slurry system to reach in order to compare with the conventional system in terms of cost/ton.

I. Panel Haulage:

One Leg:

Capital and Maintenance Cost/Ton = Total Cost per ton of Coal - Power Costs/ton
= (.059/ton + \$.040/ton) - \$.0275/ton
= \$.0715/ton

But this is based on perfect performance so we must apply the delay factor:

Capital and Maintenance Cost/Ton = \$.0715/ton/(1-.2218)
= \$.0919/ton

Power Cost = (.344 kw-hr/ton + .343 kw-hr/ton)(.04/kw-hr)
= \$.0275/ton

Total Costs/Ton = \$.0275/ton + \$.0848/ton = \$.1194/ton

II. Submain Haulage:

Power Cost/Ton = (.295 kw-hr/ton)(\$.04/kw-hr) = \$.0118/ton
Capital and Maintenance Costs = Total Cost/Ton - Power Cost/Ton
= \$.0261-\$.0118/Ton = \$.0142/Ton
Applying Delay Factor = Capital and Maintenance Cost/Ton
= \$.0142/(1-.2218) = \$.0182/Ton
Total Costs/Ton = \$.0118/Ton + \$.0178/Ton = \$.0300/Ton

III. Main Haulage (Two Legs):

Power Cost/Ton = (.724 kw-hr/ton + .724 kw-hr/ton)($.04/kw-hr) = $.0579/Ton
Capital and Maintenance Costs = Total Cost/Ton - Power Cost/Ton
= ($.065/Ton + $.064/Ton) - .0579/Ton
= $.0711/Ton
Applying Delay Factor = Capital and Maintenance Cost/Ton
= ($.0711/Ton)/(1-.2218) = $.0914/Ton
Total Cost/Ton = $.0579/Ton + $.0893/Ton = $.1493/Ton

IV. Hydrohoisting:

Power Cost/Ton = (6.193 kw-hr/Ton)($.04/kw-hr) = $.2477/Ton
Capital and Maintenance Costs = Total Cost/Ton - Power Cost/Ton
= $.298/Ton - $.2477/Ton = $.0503/Ton
Applying Delay Factor = Capital and Maintenance Cost/Ton
= ($.0503/Ton)/(1-.2218) = $.0646/Ton
Total Cost/Ton = $.2477/Ton +$.0632/Ton = $.3123/Ton

V. Service Hoist Costs:

C = (1-.2218)(7594 Ton/Shift)(220 days/yr)(3 shifts/day)
Annual Capital Cost = $1,443,000/20 yrs = $72,150/yr
Annual Power Cost = $86,655/yr
Annual Maintenance Cost = $44,000/yr
Capital Cost = ($72,150/yr)/C = $.0185/Ton
Maintenance Cost = ($144,000/yr)/C = $.0369/Ton
Power Cost = ($86,655/yr)/(220x3x7594) = $.0173/Ton
Total = $.0727/Ton

VI. Shaft:

Annual Cost = $4,680,000/20 yrs = $234,000/yr
Annual Maintenance Cost = 1% of Capital Cost = $46,800/yr
Capital Cost/Ton = $234,000/C = $.0600/Ton
Maintenance Cost/Ton = $46,800/C = $.0120/Ton
Total = $.0720/Ton

VII. Power Supply:

Annual Capital Cost = $1,704,000/20 yrs = $85,200/yr
Annual Maintenance Cost = 1% of Capital Cost = $17,040/yr

Capital Cost/Ton = $85,200/C = $.0218/Ton
Maintenance Cost/Ton = $17,040/C = $.0044/Ton
Total = $.0262/Ton

VIII. Shaft Bottom Sump:

Annual Capital Cost = $13,000/20 yrs = $650/yr
Annual Maintenance Cost = $1,300/yr
Capital Cost/Ton = $650/C = $.0002/Ton
Maintenance Cost/Ton = $1,300/C = $.003/Ton
Total = $.0005/Ton

IX. Water Supply System:

Annual Capital Cost = $404,000/20 yrs = $20,200/yr
Annual Maintenance Cost = $20,000/yr
Capital Cost/Ton = $20,000/C = $.0052/Ton
Maintenance Cost/Ton = $20,000/C = $.0051/Ton
Total = .0103/Ton

X. Control and Automation:

Annual Capital Cost = $750,000/20 yrs = $7,500/yr
Annual Maintenance Cost = $75,000/yr
Capital Cost/Ton = $.0096/Ton
Maintenance Cost/Ton = $.0192/Ton
Total = $.0288/Ton

TOTAL COST PER TON = $.8215/Ton

Conventional System Cost Per Ton Calculations (With Delay Modifications):

Delay Factor Calculations:

$$\frac{54{,}210 \text{ (Delay Time)}}{175{,}200 \text{ (Total Time)} - 14{,}600 \text{ (Travel Time)}} = .3375$$

Delay Time:

Longwall Delay	32,764 min
External Conveyor =	18,586 min
Other External	2,860 min
	54,210

I. Panel Haulage: (For 1 leg since it will yield same results in cost/ton calculations as using 2 legs)

Conveyor Equipment = $436,000/20 yrs = $21,805/yr
Belt = $302,800/5 = $60,560/yr
Maintenance = $73,890/yr
Power = $21,660/yr

A = (1-.3375)(3797 t/shift)(3 shifts/day)(220 days/yr)

Capital Costs/ton = ($21,805/yr + $60,560/yr)/A = $.0496/Ton
Maintenance Cost/Ton = ($73,890/yr)/A = $.0445/Ton
Power Costs/Ton = $21,660/(220 days/yr)(3 shifts/day)(3797 t/shift)
= $.0086/Ton
Total = $.1027/Ton

II. Submain Haulage:

Conveyor Equipment = $476,100/20 yrs = $23,805/yr
Belt = $194,643/5 = $38,929/yr
Maintenance = $47,600/yr
Power = $21,660/yr

B = (1-.3375)(7594 t/shift)(3 shifts/day)(220 days/yr)

Capital Costs/Ton = ($23,805 + 38,929/yr)/B = $.0189/ton
Maintenance Costs/Ton = ($47,600/yr) = $.0143/Ton
Power Costs/Ton = ($21,660/yr)/[(220 days/yr)(3 shift/day)(7594 t/shift)]
= $.0375/ton
Total = $.0375/Ton

III. Main Haulage (3 Belts):

Conveyor Equipment = $1,477,807/20 yrs = $73,890/yr
Belt = $1,167,858/5 = $233,572/yr
Maintenance = $264,600/yr
Power = $86,700/yr
Capital Cost/Ton = (73,890/yr + 233,572/yr)/B = $.0926/ton   B = 3,320,476.5
Maintenance Costs/Ton = $264,600/B = $.0797/Ton
Power = ($86,700/yr)/(220x3x7594) = $.0173/ton
Total = $.1896/Ton

IV. Hoisting:

Capital Equipment = $3,447,000/20 yrs = $172,350/yr
Maintenance = $344,700/yr
Power = $433,204/yr
Capital Costs/Ton = ($172,350/yr)/B = $.0519/Ton
Maintenance Costs/Ton = ($344,700/yr)/B = $.1038/Ton
Power Costs/Ton = ($433,264/yr)/(220x3x7594) = $.0864/Ton
Total = $.2421/Ton

V. Shaft:

Capital Equipment = ($5,742,000/20 yrs) = $287,100/yr
Maintenance = $57,420/yr
Capital Cost/Ton = ($287,100/yr)/B = $.0865/Ton
Maintenance Cost/Ton = ($57,420/yr)/B = $.0173/Ton
Total $.1038/Ton

VI. Power Supply:

Capital Equipment = ($1,237,000/20 yrs) = $61,850/yr
Maintenance = $12,000/yr
Capital Costs/Ton = $61,850/B = $.0036/Ton
Total = $.0222/Ton

VII. Loading Pocket

Capital Equipment = $323,000/20 yrs = $16,150/yr
Maintenance Cost = $32,000/yr
Capital Costs/Ton = ($16,150/yr)/B = $.0049/Ton
Maintenance Costs/Ton = ($32,000/yr)/B = $.096/Ton
Total = $.145/Ton

TOTAL COSTS PER TON= $.7124/Ton