THE BLENDING OF SUBSTANDARD-QUALITY COAL THROUGH PRODUCTION SCHEDULING FOR STRIP COAL MINES

by

Joe D. Megeath

A Thesis submitted to the Faculty and the Board of Trustees of the Colorado School of Mines in partial fulfillment of the requirements for the degree of Doctor of Philosophy, Mineral Economics.

Signed: Joe D. Megeath
Joe D. Megeath

Golden, Colorado

Date: April 9, 1976

Approved: Robert E D Woolsey
R.E.D. Woolsey
Thesis Advisor

Approved: J. Paul Mather
J. Paul Mather, Chairman
Department of Mineral
Economics

Golden, Colorado

Date: Apr. 12, 1976

ABSTRACT

Major legislative restrictions have been placed on the burning and mining of coal. These legislative restrictions have increased the concern, on the part of coal mine operators, for careful and accurate production scheduling to obtain marketable coal.

This work develops a model, and a solution procedure, for production scheduling for strip coal mines common to the western United States. The developed model and procedures are much easier to use than currently available techniques. In addition, the solution procedures, when applied to the special case of two simultaneous stripping activities, does not necessarily require extensive computer facilities.

Because of the demand for increased domestic coal production, it is anticipated that this work will be of current and practical value to strip coal mine operations.

TABLE OF CONTENTS

FIGURES

ACKNOWLEDGEMENTS

I would like to express my appreciation to the individual faculty, administrators, and staff of the Colorado School of Mines who have contributed to my studies and this dissertation. Special thanks are given to the members of my committee: Dr. R.E.D. Woolsey, Charles Frush, Dr. Thys Johnson, Dr. Jean Mather, Dr. Karl Newman, and Dr. Hunter Swanson.

Peter Kiewit Sons' Company provided the support and mining data necessary for this project to be completed.

CHAPTER I. INTRODUCTION

During the late 1960's the domestic coal industry was looked upon by many people as a declining industry. Much of this decline was due to the readily available supply of cheap petroleum substitutes as energy sources. This situation has changed dramatically as the actions of the OPEC raised petroleum costs to a higher level.

One of the results of this rise in oil prices and the realization of dependence in the area of energy is increased recognition for the coal industry. The industry now has prices that justify long-term capital expenditures if appropriate long-term contracts and environmental considerations are negotiated. Table I reflects some of the salient cycles and trends that have occurred since 1920.

The year, 1932, was the lowest production <u>and</u> the lowest price year for the period 1920-1974. The highest production year for bituminous coal during the same period was 1947. The number of mines showed a marked downward trend from 1920-1974 reflecting, in part, the increased mechanization and the consolidation of mines to justify the capital

TABLE 1

The Bituminous Coal Mining Industry of the United States for Selected Years

Year	Net Production (mil. tons)	Average Price Per Ton	Employees	No. of Mines
1920	586.7	$ 3.75	639,547	8,921
1932	309.7	1.31	406.380	5,427
1947	630.6	4.16	419,182	8,700
1960	415.5	4.69	169,400	7,865
1970	602.9	6.26	140,140	5,601
1971	552.2	7.07	145,664	5,149
1972	595.4	7.66	149,265	4,879
1973	591.7	8.53	148,121	4,744
1974	603.4	15.72	159,580	5,247

Sources: Energy Facts, 1973, p. 163; Coal News, Nov. 26, 1975, p. 6.

expenditures required. The reduction in the number of mines is also a reflection of the increased reliance on the large western strip mines as opposed to the smaller mines of the Eastern United States. More than half of the coal production of the United States now comes from surface mines. In 1974 surface mines accounted for 54 percent of the total coal production. This is, at least in part, due to the higher per man hour production rate of surface mining techniques. The Bureau of Mines' Weekly Coal Report No. 3036, Nov. 21, 1975 states that surface mines had a 34.98 tons per man-day rate in 1974 while underground mines had a production rate of 10.99 tons per man-day.

The U.S. has vast reserves of coal. The estimates vary considerably, but general agreement is in excess of 3 trillion tons. Of this amount of reserves, approximately 50 percent is mineable under current or reasonably projected technology (Jordan, 1974, p. 33). This places the U.S. in a position with coal that is comparable to the position the Arab States have with oil. The U.S. reserves for coal represent approximately 48 percent of the estimated total world reserves of coal (National Coal Association, 1972).

The national policy of increasing energy independence and the increased cost of petroleum will require that the domestic reserves of coal be produced at a more rapid rate than in the past. One estimate of this demand is shown in Table 1.2.

TABLE 1.2

Estimated Total Demand for Domestic Coal
The Years 1975 - 2000
(millions of tons)

Year	Total Demand
1975	636
1980	740
1985	980
2000	1,418

Source: U.S. Dept. of Interior, United States Energy Through the Year 2000, 1972.

There are other estimates of reserves and demand. The lack of agreement on absolute magnitudes is not significant here. The major point is that the U.S. has vast coal reserves and is facing an increasing demand.

The production from these vast reserves has been hampered by federal leasing policies. Actions of environmental groups have succeeded in blocking the granting of new coal leases on federal land.

The progress toward increasing production has also been slowed by environmental legislation. The legislation affects the coal mining industry on two fronts: (1) the actual disturbance of the land surface and its ultimate restoration, and (2) the emissions from the coal when burned. Only Federal legislation will be considered here.

The primary constraints on coal production relative to the method developed in this dissertation stem from the National Environmental Policy Act of 1969 and the Clean Air Act of 1970. Additional legislation has been proposed including the Surface Mining Control and Reclamation Act of 1974 (which was vetoed by the executive branch) and the Surface Mining Control and Reclamation Act of 1975 (which is under compromise meeting between the House and the Senate).

Concerning the physical disruption of the land, the proposed legislation (1) keeps the amount of disturbed land to a minimum, (2) seeks restoration on an on-going basis, and (3) <u>requires an estimated mine schedule before mining begins</u>. The method developed in Chapters III and IV of this dissertation can be used directly for (3) above.

The second aspect of environmental legislation affecting the coal mining industry is the emissions standards imposed on the burning of the coal. The Clean Air Act of 1970 stipulates that the sulfur dioxide emissions be a maximum of 4 lbs per million BTU of fuel burned for existing power plants and 1.2 lbs per million BTU of fuel burned for new power plants. These are equivalent to 2 lbs and .6 lbs of sulfur per million BTU of fuel burned. For the BTU content of coals common to the western U.S. this would translate to approximately 1 percent sulfur content by weight to be used in new plants (Jordan, 1974 and Gilbert, 1975).

The Department of the Interior through the Bureau of Mines (1974) has estimated the sulfur content of the supply of coal for selected years. These estimates have been weighted by projections of new mining operation in the western areas and the closing of high-sulfur mines in the eastern areas. Table 1.3 shows excerpts from this report for the western area and the total conterminous U.S.

It is obvious from Table 1.3 that much of the coal supply does not meet the standards set by legislation for sulfur content. In fact, according to Buckingham and Homan (1971), "68 percent of the sulfur dioxide emissions in the U.S. comes from coal which provides only 22 percent of the nation's energy".

There are two basic approaches to removing sulfur from coal: (1) stack-gas purification which removes the sulfur after the burning process, and (2) coal desulfurization which removes the sulfur before the burning process (Ferrall, 1972). The other alternative for meeting sulfur constraints is to blend high-sulfur coal with low sulfur coal so that the final product has an acceptable sulfur content.

In addition to the sulfur content of coal, users must concern themselves with other quality characteristics involving possible blending. These quality characteristics are typically BTU content, ash content, and moisture content.

TABLE 1.3

Estimated Sulfur Content of Coal Supply for 1975, 1977, and 1980

	1975				1977				1980			
	Western		Total U.S.		Western		Total U.S.		Western		Total U.S.	
% Sulfur	Mil. Ton	%	Mil. Ton	%	Mil. Ton	%	Mil. Ton	%	Mil. Ton	%	Mil. Ton	%
.1- .5	11.7	15	21.9	3	14.0	15	25.0	3	18.7	14	31.3	4
.6-1.0	48.4	64	253.0	37	60.7	63	279.4	38	83.4	62	28.8	39
1.1-1.3	15.5	20	44.2	7	20.5	21	51.0	7	30.8	23	65.2	8
1.4-1.8	.3	1	58.7	9	.4	1	62.8	9	.5	1	70.2	8
1.9-3.0	.1	-	117.7	17	.3		117.3	16	.5		126.3	15
3.0+	-	-	180.5	27	-	-	191.6	27	-	-	217.1	26
Total	76.0	100	676.0	100	95.9	100	727.0	100	133.9	100	838.9	100

Source: Bureau of Mines, 1974, Assessment of the Impact of Air Quality Requirements on Coal in 1975, 1977, and 1980, p. 161.

Although in many respects the problems in meeting environmental constraints have been highlighted in only the past few years, it must be kept in mind that the concern has been around for a long time. The economist Hotelling wrote in 1931:

> "Contemplation of the world's disappearing supplies of minerals, forests, and other exhaustible assets has led to demands for regulation of their exploitation. The feeling that these products are now too cheap for the good of future generations, that they are being selfishly exploited at too rapid a rate, and that in consequence of their excessive cheapness they are being produced and consumed wastefully has given rise to the conservation movement. The method ordinarily proposed to stop wholesale devastation of irreplaceable natural resources or of natural resources replaceable only with difficulty and long delay, is to forbid production at certain times and in certain regions or to hamper production by insisting that obsolete and inefficient methods be continued."

A similar wording could be found today, as though the idea had just been conceived. The difference today, though, is that restrictive legislation is being successfully passed. The mining industry must now find ways to efficiently work within the framework of the legislation.

The specific surface mining problem to be addressed here has to do with a coal mining property with the following characteristics:

a) Separate and distinct mineable pits with,

b) Many of the mineable pits having at least one quality characteristic which does not meet the required level,

c) The overburden ratio varies substantially from pit to pit,

d) A natural or manmade barrier (that is extremely difficult or costly to cross with stripping equipment) may exist in the property between deposits.

e) The annual required output (tonnage) from the operation has been established.

The particular problem to be investigated is that of production scheduling. Production scheduling, as used in this thesis, can be defined as the chronological ordering of the areas to be mined in such a manner that the resulting blended product meets predetermined constraints on quality and quantity characteristics and equipment utilization costs are minimized.

Time spans for the scheduling of production for a coal mine can be broken into three general categories:

<u>Long-range planning</u> can be defined as concerned with a time span of from 10 years to the life of the mine. Production scheduling for long-range planning can be used to determine feasibility of the entire property

and to provide an overall plan so that an appropriate mining sequence is begun. This type of planning is usually done with relatively scanty information (large-grid drilling patterns). The method of production scheduling in Chapter 4 will be demonstrated and found to be efficient for long-range planning.

Short-range planning can be defined as concerned with a time span of from 2 to 10 years. Short-range planning, as the mine is developed and production is taking place, often involves more detailed information about the quality characteristics and equipment performance. For example, short-range planning during actual production can involve estimates of the tonnage and overburden ratio for each proposed cut of the remaining pits in the property. This information, combined with more core-hole data on quality, can be used to devise a production schedule for a short-range plan. If this information is available, the scheduling method proposed in this thesis can be used to determine the point (if it exists) at which the stripping rate can no longer match the prescribed production rate while meeting the quality constraints. In other words, additional information can allow the method to work efficiency on a cut-by-cut basis instead of a pit-by-pit basis for short-range planning purposes. The method in Chapter IV has, in fact, been used on a producing mine with cut-by-cut information. The resulting analysis revealed the time at which additional stripping capacity would be required as well as the best mining sequence to follow in that particular situation.

Operational planning can be defined as concerned with daily, weekly, or monthly time periods for up to 2 years. Production scheduling for the operational plan is necessary to meet the quality requirements on a timely basis. The operational planning will be marked by substantial lack of flexibility in that certain pits or areas have already been chosen for blending in production. Operational planning should also be marked by additional information including 1) core-hole analysis for quality information from exposed coal, and 2) detailed information on coal thickness and production-haulage capacity. The variations in the quality characteristics can be forseen by this additional data and adjustments in the blending ratios made accordingly. The method developed in this thesis can be used efficiently for production scheduling in operational planning also. The procedure developed here will rapidly reveal problem areas, if they exist, in the operational planning stage.

The method developed in Chapters III and IV can be effectively and efficiently used for production scheduling in 1) long-range planning, 2) short-range planning, and 3) operational planning. The long-range plan production schedule provides guidelines for feasibility and for the appropriate sequence. Large losses can be incurred by beginning with a wrong mining sequence. Such an error will result in some coal being penalized by the contractual buyer or having to be sold to alternative customers at lower prices. The short-range plan production schedule provides guidelines with more complete and detailed information for the shorter planning horizons. Future problem areas can be identified on a timely basis. The operational plan production schedule provides the working plan for very short term mining plans. Although a number of parameters are set for this planning horizon, such as the pits to be developed or mined, a great deal of adjustment can be made in daily, weekly, or monthly blending ratios between pits to maintain feasibility.

The mining property that has been the motivation for the method developed in this dissertation is the Black Butte Coal Company (hereafter referred to as the Black Butte Mine) located in southwestern Wyoming. All actual data, number of pits, and extent of reserves will be altered, while the salient features pertinent to the problems of production scheduling and the developed method are retained. The alterations are solely for the purposes of confidentiality and emphasis of important features.

The proposed Black Butte Mine encompasses approximately 100 square miles of land, in which the underlying coal deposit is neither continuous nor homogeneous. In other words, the coal deposits in this property are separated by significant areas of non-mining land in a polka-dot fashion.

The topography is characterized by a series of north-south running, arid, rocky ridges. These ridges have a steep western face and relatively gentle sloping eastern faces.

The mineable coal deposits in the property are from four different formations: Wasatch, Fort Union, Lance, and Almond. These four formations have significantly different characteristics relative to sulfur and BTU content. The property is also marked by faulting ranging from displacement of a few feet to over 100 feet. Faulting also contributes to changes in the quality characteristics of the coal (Gambill, 1971).

The coal beds are between 5 and 30 feet thick and the overburden varies from 20 feet to a maximum of 150 feet that can be economically removed with current technology. The mineable deposits range from 1 mile to 5 miles in length and are less than 1 mile in width.

In addition, the mainline of a transcontinental railroad runs through the property. Because of the major disruption that would be caused for the heavy rail traffic, a dragline should only cross this railway a maximum of one time.

The rough terrain and the geological factors of the underlying coal deposits give the Black Butte Mine all the characteristics previously mentioned for this type of problem. Contractual requirements of the coal-burning consumer of this coal stipulate an annual output and coal quality characteristics which are violated in many of the individual pits. Careful blending and subsequent production scheduling must take place to meet the quality characteristics which are set by environmental legislation. This dissertation is concerned with the problem of creating a production schedule in a way that blends the coal to meet quality constraints, maintains dragline movements between pits to a minimum, and maintains the stripping activity at a sufficient rate to enable the production of coal at approximately the required rate. This, of course, must be done with a minimum number of draglines and simultaneously producing pits to minimize capital requirements.

In Chapters III and IV a method will be shown that effectively solves this production scheduling problem. Chapter II contains a review of previous work done on production scheduling.

CHAPTER II - SURVEY OF THE LITERATURE

There have been many papers and articles published concerning the use of operations research techniques in the mining industry. On the basis of the literature surveyed, the mining industry seems to have been a relatively late user of these techniques considering their many applications. T.M. Ware, Administrative Vice President of International Minerals, wrote in 1956, "There is more need for OR in mining than in almost any other industry" (Ware, T.M., 1956, p. 119). It would appear from the number of articles on the subject, particularly since 1967, an attempt has been made to meet that need.

A review of operations research techniques in the mining industry through 1967 is contained in Coyle, 1969. Coyle's work was done partially to determine the state-of-the-art at that point in time.

The salient features of some 75 papers are summarized in that paper. In the beginning paragraphs, Coyle gives the following thought about the reviewed literature,

> "...the writer feels that the mining industry has some peculiar structural and economic characteristics (exhaustibility of reserves, for example). In the light of these characteristics, it is felt, therefore, that a literature which showed no major evidence of systematic attempts at an analysis of the industry's special problems, as distinct from the use of more or less standard techniques, could not be taken to indicate a highly developed state of the art".

Coyle concluded that (1) much of the literature was too vague to accurately assess, (2) much unpublished work had been done, and (3) "...the application of OR to the mining industry has not reached a high level of elegance or relevance, though there are many instances to the contrary".

In Coyle's section concerning production and planning he reports that the relatively short-term production schedules described used, basically, simulation and/or linear programming approaches. These two approaches are still the mainstays of production scheduling problems, as evidenced by the more recent literature reviewed by this author.

An example of Coyle's reference to the simulation approach is Dunlap and Jacobs, 1955. That paper is directed toward one aspect of production; namely, the large draglines in the mining of phosphate. A large, sophisticated simulation model of dragline operations was developed to make economic comparisons between various methods of operation.

An example of the use of linear programming is contained in Redmon, 1964. Redmon applied linear programming techniques, in a very basic manner, to (1) an underground mine for blending ores from a stope, and (2) blending ores from 10 different uranium mines to achieve mill consistency and constraints on a daily basis. That paper is also an example of the previously mentioned Coyle remark concerning the use of more or less standard techniques that do not get into an industry's special problems.

The use of operations research techniques for mine design or production scheduling has grown increasingly sophisticated since the literature was reviewed by Coyle. The sophistication is partly because the techniques used are more atuned to the subtle problems unique to the mining industry rather than a patching together of mining problems and techniques known to work in other fields. The computer-aided approach to mining problems from mine production scheduling to ultimate pit limits showed substantial improvements in efficiency by using the block concept in which the ore body is divided into equal volume blocks for analysis (Johnson, T.B., 1969, p. 541).

The most extensive number of papers concerning OR in the mining industry, and in particular, production scheduling, appears in the series of proceedings from the International Symposium on Computer Applications and Operations Research in the Mineral industry. This symposium began in 1961 at the University of Arizona, and has continually grown since that time. The twelfth symposium was held in Golden in 1974. The name of the symposium and its proceedings has had minor changes from year to year, but is generally recognized as the APCOM series.

In the author's opinion, these papers, though prolific, tend to lack the detail of information which would allow verification of either accuracy or practicality. However, most of them are of sufficient extent to make at least subjective judgments concerning their utility.

The growing development of the state of the art referred to by both Ware (1956) and Coyle (1969) can be readily observed in the APCOM series. In the first symposium, there was one paper or lecture on production in the mining industry. That paper by R.F. Hewlett (1961) began with a very basic discussion of models and the modeling process. Hewlett then went on to give three examples of the application of OR techniques to production. The first example was an application of linear programming to blending bauxite ores in Jamaica while minimizing the combined costs of truck and rail haulage. This formulation, which kept silica below a maximum percentage and alumina above a minimum percentage, was a very basic linear programming problem. The second example in that paper utilized simulation and queueing theory to model an underground mine. The third example was again linear programming, but applied to blending uranium ore. Hewlett's paper, if taken as an example of the level of sophistication that was acceptable to the industry, illustrates once again, that OR techniques were not highly developed in the mining industry at that time.

A comparison of the proceeding of APCOM in the first session in 1961 to the proceeding in the last two years (1973 and 1974) reveals a remarkable transition in a little over a decade. The 1973 and 1974 papers from APCOM show a high level of sophistication and, in this author's opinion, techniques that reflect the problems peculiar to the mining industry. Examples of these later papers will be cited later in this section.

The papers from APCOM and other sources, while growing in sophistication, have often tended to be directed toward a particular ore, a particular mining method, or a particular mine. Although the concepts may remain the same in different situations, an approach to one particular mine property can lose its applicability to other mine properties if basic mining methods, ores, or objectives are changed.

Many of the works done on production scheduling and mine design deal primarily with underground methods. An example of this is done by Diest, F.H., and Roberts, M.P., (1972). Their simulation model includes linear programming to meet quality constraints. However, they report that the model is too big to be done on a life of the mine basis.

The literature indicates that the use of simulation as a basic technique for production scheduling in underground methods as well as surface mining generally includes decision variables concerning physical mine parameters. These variables represent such qualities as location of ramps, type of equipment, location of shafts and hoists, and which veins or locations to mine. There are some simulation models that also include the decision of whether the mine should be underground or open-pit. Examples of this are in Wilke (1972), and in Burne (1972) in which various types of equipment or facilities can be tested by having the analyst inject the appropriate data into the respective model.

Mathematical programming has also appeared to have obvious applications in production scheduling with many individual techniques being applied. To develop the proper sequencing, zero-one or integer programming is normally utilized to handle the problems presented by a mining operation. An example of the basic formulation for this can be seen in Avidan (1974) who also applied separable programming to underground gold mines.

Other examples and variations of production scheduling for underground methods include Riester (1964), who developed a huge mathematical programming model (no mention was made of actual application to a real mining situation), McGiddy and Whitfield (1974), who applied the block concept in a way that used the quality characteristics of cut-off points rather than blending averages, and Pye and Burne (1974), who worked with the sublevel caving of copper ore. Pye and Burne built the constraints into the model, but allowed the program constraints to be overridden by the analyst in an interactive computer program. Both of these latter examples are primarily for the purpose of testing various machine characteristics.

Some common problems exist between underground and surface production scheduling (e.g., blending of ores). However, it is difficult to envision a model that can handle both underground and surface general operations without extensive modifications.

The blending of coal and the production scheduling of open-pit coal mining at the Four Corners Power Plant is discussed by Gambill (1971). Much of his article is concerned with physical equipment and handling methods in using blending-stockpiles before the coal is burned. However, some of the principles are the same as for a situation in which no stockpiles are maintained. Gambill's problem was considerably simplified by his report that the BTU characteristic of the coal was well correlated with the other seven quality characteristics involved. Therefore, once BTU was blended properly, the other quality characteristics were assumed to be automatically maintained at acceptable levels. This is not the case in the problem discussed in Chapter III of this dissertation.

A major problem in the methods of production scheduling found in the literature is the size of the resulting models and/or the amount of computer time required to solve the model.

An excellent example of the size of the model that is required to handle blending problems in addition to zero-one variables over a reasonably lengthy planning horizon is found in Imaizumi and Takeda (1971). A very large program connected with the mineral industry has been formulated to determine the best distribution of iron ore to various plants and products taking into account transportation facilities for the Japanese steel industry.

Manula, Falkie, and Su (1973) applied mathematical programming to a model using buckel wheel excavators, belt conveyors, and truck or rail haulage systems. This model, although large and complex, would include no more than one 6-hour shift at a time, which would classify the model as an operational planning guide. No comparison was made between their theoretical formulation and an actual mine operation.

Some authors have seen integer programming as being an infeasible approach to production scheduling because of the size of the model and the computer facilities required. The approach was consequently converted to Lagrange multipliers and a network formulation (Davis and Williams, 1973). No mention was made as to the amount of efficiency gained, however.

The overall size of a long-range production schedule was handled by Pronk Van Hoogeveen, et al, (1972), by packing the data. In their simulation of an open-pit copper mine these authors attempted to reduce the data storage required by giving all blocks with the same quality characteristics the same name. The result still required huge computer storage area. This detailed simulation model took into account decision variables such as ramp location and alternative machine sizes, and would reputedly output ultimate pit limits in addition to the best mining sequence and a short-term mining plan. The size of the model was a disadvantage and it was not addressed to the strip-mining characteristics of a coal operation.

Solving models requiring integer and zero-one programming techniques is known to present computational difficulties in real world situations. The models found in the literature commonly require branch and bound techniques. These techniques are discussed in many operations research textbooks.

A good discussion and explanation can be found in Hillier and Lieberman (1967) and in McMillan (1970).

Special techniques have been added in different works as attempts to overcome particular problems or gain efficiency in modeling real mining problems. Splain and others (1972) developed a model which allows the constraints to be violated with a corresponding penalty factor for the magnitude of the violation in the objective function. It is claimed that the allowable violation and the corresponding penalty are more in keeping with real world situations. The basic penalty function technique is due to Rosenbock (1960). With the amount of information yielded by a basic sensitivity analysis for linear programming, it is questionable whether the violation-penalty formulation offers more realism or information. In addition, although it appears to be more realistic to treat the right-hand-side estimates as stochastic, rather than as point estimates, in this author's opinion it would be difficult to accurately describe or estimate the necessary parameters for stochastic RHS's.

In most of the literature surveyed, the mine schedule is broken into distinct time periods to be analyzed (e.g., quarters, years, or 5-year periods). In works such as Posaner (1974), a standard block concept is initialized, but the time period itself is allowed to vary in scheduling the ore-digging equipment. Each time period is determined by the length of time between each necessary move of a piece of equipment. However, the model is limited to investigating only one period in the future for possible moves.

One of the most detailed explanations of the use of mathematical programming for production scheduling is contained in Johnson (1968). In that work, Johnson has described the extensive formulation required and has developed a systematic solution for a generalized model. That particular paper preceded most of the sophisticated works found in the literature by this author. Johnson's formulation will be more completely described in Chapter III, where it will be used as a point of departure for the development of this author's techniques.

In this author's experience with production scheduling in a topographically complex open-pit coal mine such as described in Chapter I, there are a number of variables or constraints that are common sense to the human mind (e.g., in two seams, you must remove the top seam before you can produce the lower seam). However, these constraints represent complex logic, involved formulation, and increased computation for the computer. Tucker (1961) wrote,

> "Where are the pitfalls? Everywhere, for the unsuspecting wearer of rose-colored glasses. The computer is a tool which has allowed men to raise their sights. For a space vehicle or an airways control system, the problems

> are so vast we could not begin to handle the work without the computer. The cost is a small figure in proportion to the gains. For a working industry, the computer can be an asset when it cuts time and cost, provides information not available previously, and aids management in making decisions. The extent to which these results can be achieved at a realistic expenditure of resources depends on the clear thinking and careful planning of the people who use the computer. The trouble begins when we try to substitute the computer for man's reasoning ability."

In summary, the most complete discussion of production scheduling methods can be found in the APCOM series. The methods used are variations of, or specific applications of, either simulation or mathematical programming.

The techniques and models discussed in the literature have one or more of the following qualities:

1) Large and complex formulation, requiring extensive computer facilities,
2) No actual comparison against real mining situations,
3) Applications to specific mining techniques or situations which are not necessarily compatible with the problems faced in open-pit western United States coal mines.

The Selected References contains additional examples and representations of the examples, methods, and techniques cited in this section.

CHAPTER III - A METHOD OF PRODUCTION SCHEDULING FOR A COAL STRIP MINE

In Chapter I environmental concerns and legislation were discussed. Environmental issues such as sulphur content of coal often make it necessary to carefully blend the coals. This blending will be necessary when at least one area of coal does not meet the standards of quality by itself. In such cases, production scheduling must take into account the fact that at least two pits must be mined simultaneously to use the below-standard coal.

In Chapter II it was shown that many of the existing models for production scheduling are complex in formulation and require substantial computer facilities for the solution of such models. In this Chapter and Chapter IV a method for production scheduling will be presented that requires minimal computer facilities or none at all. This method will be applicable to many coal strip mines including mining properties having complex topography. Complex topography is used here to mean rough terrain, making it difficult to move stripping equipment and possibly the existence of a barrier in the property, such as a river or a railroad track.

Many of the methods and models discussed in Chapter II were concerned with general open-pit, underground, and non-coal mining operations. In these cases the production scheduling procedure must be concerned with the mining of a particular segment only after its surrounding segments or blocks have been removed. The concern for making a segment available for extraction only after it has been "exposed" makes the block concept an efficient tool for production scheduling.

Coal strip mines, on the other hand, are (1) mined out in long strips, and (2) typically made up of coal that does not vary rapidly in quality over distance. The block concept can be used in production scheduling for coal strip mines, however, that concept is not necessarily needed.

In addition, as indicated in Chapter II, many of the production scheduling models are directed toward examination of various types and combinations of equipment (e.g. dragline capacity, size of trucks, type of haulage, etc.). The analysis here will examine the feasibility and sequencing for a production schedule assuming that methods of extraction and haulage have been determined by commonly known mining engineering techniques.

The models currently developed take the objective to be maximization of profit and use this objective directly in the analysis (maximizing the profit function in the case of mathematical programming techniques). Although it is recognized that the maximization of profit is the primary goal of production scheduling, the method developed here will accomplish this in an indirect heuristic manner. The purpose of the method developed here is to determine a production schedule for various pits such that (1) the quality constraints are preserved, and (2) high cost items such as the movement of draglines between pits are kept to a minimal level. At the same time the method (1) simplifies the data requirements, (2) simplifies the formulations, and (3) decreases the computational time needed for solution of the model. The technique used does not necessarily require computer assistance and is, therefore, of particular value to those companies or corporations who do not currently possess large computer facilities or a well-developed software package for production scheduling. It will also be of value to operations involved in preliminary feasibility studies that would not justify the gathering of detailed information or the use of computer time required for any currently developed large-scale model that may be available.

Current Approaches:

The formulation of a general open-pit production scheduling problem using the block concept is exemplified in Johnson, 1968. In Johnson's work the formulation for

T time periods is shown to be:

Maximize

$$\sum_{j=1}^{T} C_j X_j$$

Subject to:

$$\sum_{j=1}^{t} D_j X_j = d$$

$$A_j X_j = b_j \qquad \forall\ j$$

$$\sum_{j=1}^{t} B_{tj} \leq 0 \qquad \forall\ t$$

$$\sum_{rpt} X_h^{rpt} \leq 1_h \qquad \forall\ h$$

$$X_j > 0 \qquad \forall\ j$$

where:

C_j is a profit vector for each block mined in time period j,

X_j is a vector of all blocks reflecting how much to mine in each period j,

D_j is a matrix of coefficients for constraints such as capital and total mine reserves concerning the entire time span of the mine during each period j,

d is a vector of the limits of the constraints (RHS's) for the D_j,

A_j is a matrix of the coefficients of the constraints such as bounds on content of ore for each time period j,

b_j is the vector of limits of the constraints (RHS's) for each time period j. In a coal mine the b_j's may be equal for all j.

B_{tj} is the matrix for the constraints on the allowable mining sequence or the requirement that "surrounding" blocks must be mined first, (see Appendix A)

x_h^{rpt} is the amount of block h mined in period j as material r and processed by method p. The total of this constraint must be less than l_h which is the volume of block h.

Keeping in mind that each line indicated in the formulation above represents entire matrixes, it is relatively easy to observe that this linear programming problem becomes huge as the time periods are made smaller and consequently, the number of the time period, T, becomes larger. Also, the problem becomes rapidly larger as the number of blocks is increased, thus increasing the number of decision variables in each vector X_j.

Johnson has shown that this problem can be solved by decomposition with a network formulation for each subproblem involved. In that method the sequencing constraints make up the subproblem in the decomposition process.

For the strip mining of coal in the Western United States, the above formulation can be substantially shortened. In the usual case of open pit western operations (1) once the coal deposit is reached all material is identifiable and used as coal eliminating the material superscript r, and (2) the coal normally is not subjected to separate processes, eliminating the process superscript p.

The size of current general models such as Johnson's formulation is affected by both the block concept and the equal-length periods concept. However, in the mining of coal, deposits remain relatively homogeneous over large volumes so that the equal size block concept becomes a factor which can contribute unnecessarily to the size of the formulation. Because of the approach to production scheduling as defined in this paper, the deposits, instead of being divided into small (100' x 100' x 100') blocks, can be categorized into uneven, large areas that may even be an entire pit deposit.

The effect of equal-length periods of operation on the size of the formulation can be demonstrated. Considering only the required quality characteristics, production, and the amount of coal in each deposit, the linear constraints and objective function would be formulated as:

(0) (maximize profit) $\sum_j P_{tj}x_{tj}$

Subject to:

(1) (maximum sulphur) $\sum_j s_{tj}x_{tj} \leq S \sum_j x_{tj}$ for each period t

(2) (minimum BTU) $\sum_j b_{tj}x_{tj} \geq B \sum_j x_{tj}$ for each period t

(3) (maximum moisture) $\sum_j m_{tj}x_{tj} \leq M \sum_j x_{tj}$ for each period t

(4) (maximum ash) $\sum_j a_{tj}x_{tj} \leq A \sum_j x_{tj}$ for each period t

(5) (constant production) $\Sigma x_{tj} = K$ for each period t

(6) (tonnage of pit) $\sum_{j} x_{tj} = T_j$ for each pit j

where each time period t is equal in length and

x_{tj} = number of tons mined from pit j in period t,

P_{tj} = profit per ton from pit j in period t,

S_{tj} = average sulphur content of pit j (the subscript t is carried as a matter of continuity on the quality constraints as the sulphur, BTU, moisture, or ash content remains constant over all periods t)

b_{tj}, m_{tj}, a_{tj} = average BTU, moisture, and ash, respectively, for pit j,

S, B, M, A = allowable sulphur, BTU, moisture, and ash content of coal produced (assumed to be constant over life of the mine).

K = required production in each period t,

T_j = tons of coal available for production in pit j.

With only these constraints if a mining property of 20 pits were to be mined over 20 years, and if the periods were made one month in length, the linear programming formulation would contain 4800 decision variables and 1220 constraints. Adding equipment constraints of haulage and stripping capacity available in each period would require at least 480 more constraints.

The portion of the formulation discussed above (0) through (6), only considers the constraints for the blending process, constraints (1) through (4); the production requirement, constraints (5); and the available tonnage in each pit, constraints (6). Left to be considered are the proper sequencing of the pits, costs of dragline movements between pits, natural barriers preventing dragline movement between two subsets of pits, and equipment limitations. These additional items would require binary variables to indicate that movement by a dragline to a pit either takes place or does not take place in any time period. This would substantially increase (1) the number of decision variables, (2) the number of constraints, and (3) the computing time.

Further, binary variables and/or additional constraints would be required to restrain to a predetermined maximum the number of pits being simultaneously obviously should be kept to a minimum in a working coal operation.

Specifically, these additional constraints for costs of dragline movements between pits and limitation of the number of simultaneously producing pits, when combined with formulation (0) through (6), could be formulated as:

(Model I)

Maximize:

(0) $\sum_{tj} P_{tj} x_{tj} - \sum_{tjkL} C_{tjkL} Y_{yjkL}$

Subject to:

(1) (maximum sulphur) $\sum_j s_{tj}x_{tj} \leq S\sum x_{tj}$ for each period t

(2) (minimum BTU) $\sum_j b_{tj}x_{tj} \geq B\sum x_{tj}$ for each period t

(3) (maximum moisture) $\sum_j m_{tj}x_{tj} \leq M\sum x_{tj}$ for each period t

(4) (maximum ash) $\sum_j a_{tj}x_{tj} \leq A\sum x_{tj}$ for each period t

(5) (constant production) $\sum_j x_{tj} = K$ for each period t

(6) (tonnage of pit) $\sum_t x_{tj} = T_j$ for each pit j

(7) $\sum_k Y_{ookL} = 1$

(8) $\sum_{\alpha \in D_L} Y_{(t-1)\alpha KL} = \sum_{\beta \in D_L} Y_{tk,\beta,L} \quad \forall\, t,L,K$

(9) $x_{t,j} \leq M \sum_{K \in D_L} Y_{tjkl} \quad \forall\, t,j,L$

and $Y_{tjkl} = 0,1$

Where:

C_{tjkL} is the cost of moving machine L from pit j to pit k in the time period t,

M is a large number (e.g. 10^{10})

Y_{00kL} is the movement or non-movement of machine L to pit K to <u>begin</u> the mine schedule,

D_L is the set of pits assigned to machine L,

α is the origin of the move of machine L to pit K, β is the termination of the move of machine L from Pit K $(\alpha, \beta \in D_L)$.

In this formulation:

a) As previously shown, constraints (1) through (6) assure quality and production feasibility,

b) Constraints (7) assure that one and only one pit is begun initially by machine L,

c) Constraints (8) assure that if stripping equipment is moved into pit k in a time period, it must be moved from pit k in the following time period. The formulation allows the equipment to remain in the same pit for sequential time periods, but in each time period equipment must be accounted for to "balance" the movements.

d) Constraint (9) forces Y_{tkjl} to take a value of one when pit j is mined. The value of Y_{tkjl} will remain zero whenever possible because of the maximization of the objective function.

This formulation assumes that each piece of stripping equipment, L, is assigned to a specific set of pits, D_L. It also assumes that the stripping equipment will not leave a pit unfinished and return to that pit later.

The final set of constraints required for a general formulation concerns any specific sequencing of pits that must be maintained. For instance, the mining company may need or want to mine one particular pit before another. Sequencing is also required in multi-seam deposits. The set of constraints for sequencing is explained in Johnson (1968) and has the form:

(10) $\sum_{j=1}^{t} B_{tj} \leq 0 \qquad \forall \; t$

This constraint is shown as the third set of constraints in the previous discussion of Johnson's formulation and is shown in Appendix A.

Simplification of Johnson's Model

Model I (0) through (10), is less general than Johnson's formulation shown earlier. Specifically, the formulation (0) through (10) is based on the particulars of strip mining of western coal. Hence, some variables are eliminated such as ore-waste and type of processing for the ore. In addition, the equal size block concept is not a necessity for Model I, (0) - (10).

However, Model I is still very large and requires binary programming techniques. With some modifications in requirements, it will be shown that a model descriptive of many strip coal mines can be made that is smaller and requires less computation time for its solution. To achieve this the concept of equal size blocks will be dropped. The concept of equal-length production periods will also be eliminated.

The equal size blocks concept will be exchanged for generally unequal sized "pits" of homogeneous coal. Each pit, which contains coals of homogeneous quality characteristics, is much larger than the typical block used in many current formulations. The pit may not be the total deposit as in (a) of Figure III-1. Instead, considering elongated deposits, the coal may be roughly homogeneous in two or more horizontal areas such as (b) or they may be realistically divided into vertical pits as in (c) of Figure III-1. Each pit could also be a seam in a multiseam deposit such as depicted in (d) of Figure III-1. Pits such as depicted in (d) would normally be mined in an alternative pattern. The length of each area should be sufficient to allow efficient usage of the stripping equipment. The first step in using the simpler model is to divide the mining property into mineable areas that have an acceptable degree of homogeneity in their quality characteristics.

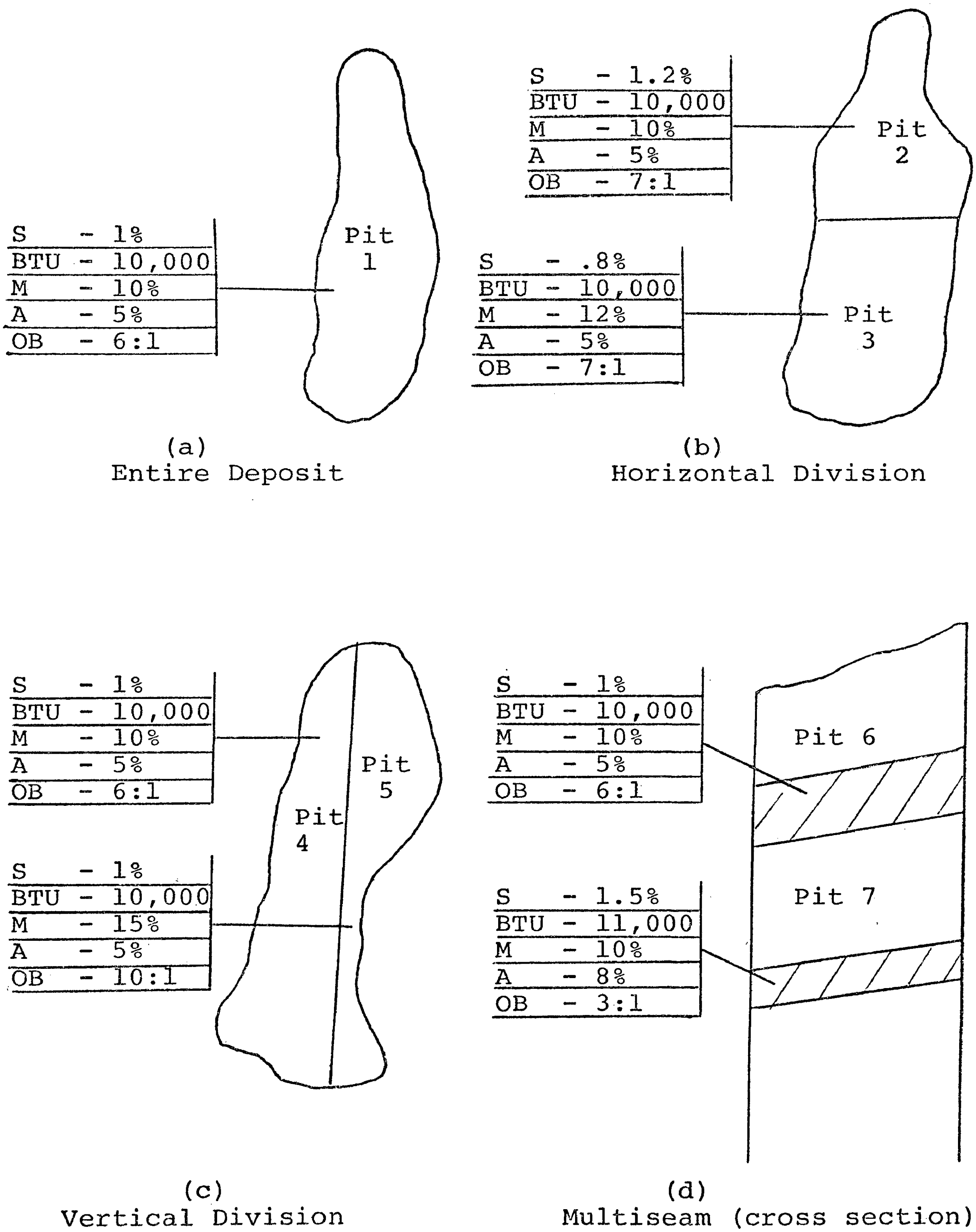
The concept of equal-length production periods can also be eliminated in order to simplify the model. Instead of having production periods of equal length, the production period can be the length of time between dragline movement between pits. In other words, a production period continues until one of the pits being mined is exhausted of coal.

The length of a time period, t, is then dependent upon the amount of coal, T_j, in each of the pits, and the rate of production, R_j, in that pit. The length of the time period can then be expressed as

$$N(T_j, R_j) = \frac{T_j}{R_j} .$$

FIGURE III - 1

Alternative Configurations of
"Homogeneous" Coal Pits



Thus, $N(T_j,R_j)$ will be a variable rather than a constant.

When the two concepts of equal size blocks and equal-length production periods are eliminated, Model I can be further simplified by breaking the production scheduling problem into two segments:

(a) the minimization of the costs of dragline movements between pits, and

(b) the finding of feasible blending combinations of various pits.

First, the minimization of dragline move costs, (a) above, will be examined independently of the blending feasibility problem. Second, the blending feasibility of the various combinations of pits, (b) above, will be analyzed. Third, the dragline movements will be coordinated with the feasibility analysis.

A heuristic solution, easily obtainable, identifying the minimum cost path for each dragline and the corresponding pits assigned to that dragline is suggested by Gavett (1965). In Gavett's solution a matrix is formed reflecting the cost of moving a dragline from any of the affected pits to any other pit. A pit 0 should be included as the construction site of the dragline. The cost of moving a dragline from any pit to the same pit is indicated as an "X". The procedure is as follows:

1. Starting in the row 0, find the smallest figure in that row. Draw an arrow to it and circle it.
2. Follow the column found in the previous step to the first "x". Draw an arrow to and circle the smallest number in that row. Eliminate from further consideration any rows or columns that contain a circled number.
3. Continue this procedure until exactly one circle appears in each row or column. The resulting arrows designate a heuristic solution to minimize the total movement costs.

Gavett reports that this procedure will average only 6 percent difference from the absolute optimal solution total cost. An example of the use of this procedure will be shown in Chapter IV. It should be noted that the above procedure is not always necessary. If the mining property is small enough and contains few enough pits to be assimilated by an analyst, the dragline movement can potentially be done by inspection. In this case, a detailed map of the mine property can visually provide much of the critical information for production scheduling such as:

(1) which pits are not eligible in a particular sequence because of natural barriers,

(2) which sequences of production should be avoided because of the distance involved in moving stripping equipment,

(3) which combinations should be avoided because of excessive distances from the loading station, and

(4) which seams must be mined in proper sequence in a multiple-seam deposit.

These four items, which can be determined readily by observing a map, require extensive formulation and programming for a typical computerized production schedule. The sets of constraints (7) - (10) in Model I represent the formulation that is required to accomplish this using binary variables.

The second segment of the problem, the determination of the feasibility of the potential combinations of pits, is considered next. In Model I, (0) through (4) reflect the linear programming equations necessary to find an optimal solution that is feasible relative to quality requirements over all time periods. That formulation can be altered so that the decision variables, x_j, represent proportions of production rather than tonnages. In addition, consider an individual time period rather than all time periods. The formulation for each individual time period or combination of pits would then be as shown in Model II.

Model II

Maximize:

(A0) $\sum_j P_j x_j$

Subject to:

(A1) $\sum_j s_j x_j \leq S$

(A2) $\sum_j b_j x_j \geq B$

(A3) $\sum_j m_j x_j \leq M$

(A4) $\sum_j a_j x_j \leq A$

(A5) $\sum_j s_j x_j \geq S'$

(A6) $\sum_j b_j x_j \leq B'$

(A7) $\sum_j m_j x_j \leq M'$

(A8) $\sum_j a_j x_j \leq A'$

(A9) $\sum_j x_j = 1$

Further constraints concerning dragline stripping capacity and minimal proportion of production from any individual pit would be:

(A10) $x_j \geq \alpha$ for each j in the combination being evaluated

(A11) $x_j P \leq R_j$ for each j in the combination being evaluated

Where:

S', M', A' are minimal values (possibly zero) of the quality characteristics sulphur , moisture, and ash, B' is a maximal value of the quality characteristics BTU, α is the minimum proportion of production that could be mined from one pit, P is the annual tonnage requirement for total production, and R_j is the maximum tonnage of coal that could be uncovered by the dragline in pit j with overburden OB_j.

For each time period, then, constraints (A1) through (A4) insure that the blend of coal from various pits will meet the quality requirements. Constraints (A5) through (A8) will preclude "high grading" or the blending of very high quality coals in one period at the expense of later time periods which might have no feasible blends satisfying (A1) thorugh (A4). Constraints (A5) through (A8) are necessary because each time period is being analyzed individually rather than all at once as in constraints (0) through (8) of Model I.

Constraints (A10) insure that each of the K pits included in the particular combination under consideration will have a significant proportion of the production. Constraints (A10) also force all K draglines to be used in each time period. Constraints (A11) exclude any proportion that does not allow the stripping activity to stay ahead of production requirements.

The general mathematical formulations, discussed previously have an objective function of maximizing the profit such as (A0). In the model used here it is assumed that either (1) all pits are profitable to mine ($P_j > 0$ for all j) or (2) because of environmental constraints, all coal that is available must be mined. The heuristic solution from the formulation being proposed is based on finding feasible combinations of pits that must be mined. The profit function will not be used in the proposed formulation. The rationale (and the resulting simplification) for changing the objective function will be examined next.

If the quality characteristics of the mining property are sufficiently complex to warrant a sophisticated analysis of the production schedule, then the number of combinations of pits that can be used to meet the standards will be limited. In fact, a method is needed that can efficiently determine if any feasible solution exists. For those mining properties that have many feasible solutions, Gavett's method of finding the least cost dragline movement path could result in a sequence that has merely to be checked for feasibility. Any required modifications in the path should be minor.

In addition, the elaborate mathematical programming model with an objective function of maximizing the profit requires detailed and accurate estimates of current and future prices and costs of many variables. Because these forecasted prices and costs can be as much as 20 or 30 years in the future, the author feels that the solutions obtained are also estimates and cannot be considered to be absolutely optimal solutions. A reasonably accurate heuristic solution to the model reduces to finding a feasible solution. At a later part of this section, the point of maximizing profit will be refined.

A substitute for the profit function will now be developed. Consider the production of coal that must meet the requirements of more than one quality characteristic (e.g., sulphur, BTU, moisture, ash) from the type of mine described in this paper. There is often at least one of the quality characteristics whose overall average (over the total mine property) is very close to the stipulated contractual or environmental requirement. Thus, each individual combination of blended pits should be as close as possible to that stipulated requirement without violating it. If this policy is not followed during each production period, the chances are increased substantially that coal to be produced later will not be able to meet the contractual requirement. Such a quality characteristic will be referred to as the "critical characteristic" in this paper. Under current environmental legislation it will not be atypical for this critical characteristic to be the sulphur content of the coal. For simplification, it will be assumed in this work that sulphur is the critical characteristic even though any of the quality characteristics can be. The critical characteristic will be used in the formulation of an objective function replacing the use of the profit function as the objective.

The previous objective function was specified as

(A0) maximize $\sum_j P_j x_j$.

This can be replaced by

(A00) maximize $\sum_j s_j x_j$.

The objective function (A00), for each period being analyzed, will attempt to make each blend of pits involved contain the maximum amount of sulphur without violating constraints (A1) - (A11). The new objective function (A00) and the constraints (A1) - (A11), Model III, are concerned only with a single production period.

In any given period, then, K pits can be analyzed by Model III. If a solution to the model does not exist, this particular combination of K pits should be eliminated from further consideration. At this point the cost of moving one of the draglines to its current position from its previous position should be increased to a very large number. The new resulting path of that dragline will yield a different combination of K pits to be analyzed by (A00) - (A11).

If a solution does exist for Model III, the optimal answer will be the percent of production that should be mined from each of the K pits involved. These percentages,

x_1^*, x_2^*, $x_3^*,\ldots,x_K^*$, for K pits in production simultaneously, can be applied to the tonnages of coal in pits 1,2,3,...,K, respectively, to determine the pit that is depleted of coal first.

For instance, suppose:

$K = 4$

$T_1 = 100$ T

$T_2 = 200$ T

$T_3 = 400$ T

$T_4 = 400$ T

$P = 600$ tpy

$X_1^* = .20$

$X_2^* = .25$

$X_3^* = .40$

$X_4^* = .15$

Then, the pits will be mined at the annualized rates of:

pit 1 = .20 X 600 = 120 tpy

pit 2 = .25 X 600 = 150 tpy

pit 3 = .40 X 600 = 240 tpy

pit 4 = .15 X 600 = 90 tpy.

Hence, pit 1 will be depleted first after $\frac{100}{120}$ =.83 years. This particular time period would be .83 years long. The tonnage mined from each pit and the remaining tonnage at the end of the period would be:

Pit	Beginning Reserves	Mined in This Period	Ending Reserves
1	100 T	100 T	depleted
2	200 T	124.5 T	75.5 T
3	400 T	199.2 T	200.8 T
4	400 T	74.7 T	325.3 T

The depleted pit will be eliminated from the combination being considered. The next pit in the path of the dragline assigned to the depleted pit is added to the remaining pits to make up the next set of k pits to be considered for feasibility. The original k-1 pits are carried forward to this new combination with the decreased amount of coal in each after appropriate production from the previous combination. In this way there are always k pits being considered during any production period. This procedure is repeated until all coal is depleted.

In summary the general procedure is:

a) Find the best path for each of the K draglines required (Gavett's method can be used).

b) Take the next K undepleted pits from these paths and solve Model II. If a feasible solution does not exist, increase the cost amount of the movement cost matrix (Gavett's method) for the appropriate dragline to a very large number. Return to step (a). If a feasible solution exists, go to step (c).

c) Using the optimal solution (percentages) from (b), delete the appropriate amount of tonnage from each of the K pits in the combination until at least one pit is depleted.

d) Eliminate depleted pits from consideration and return to step (b).

The above procedure can be used to find a production schedule using any number of pits in production at one time. Its efficiency, though, makes substantial gains when only two pits are to be in production at any one time. Under this condition, which is not uncommon, the above procedure leads to a method of solution which does not necessarily even need computer aid. The special case of two pits under production simultaneously (two draglines being operated simultaneously) and the resulting simplification of the production schedule model and solution will be discussed in Chapter IV.

CHAPTER IV - A METHOD OF FINDING OPTIMAL PRODUCTION SCHEDULES WHEN EXACTLY TWO DRAGLINES ARE REQUIRED AND EXACTLY TWO PITS ARE BEING BLENDED

In Chapter III it was shown that the general formulation of a production schedule model as depicted by Johnson can be simplified for the strip mining of coal. It was also proposed in Chapter III that the elimination of the equal size block concept and the equal-length production period can result in further simplification of the general model. The maximization of a profit function was dropped and replaced by the maximization of the critical characteristic function. Further, the final generalized formulation in Chapter III, Model III does not require a computationally time-consuming binrary programming code as do other formulations.

It will be shown in this section that the final formulation of Chapter III, Model III, results in a very efficient method of solution for the special case of two pits being blended simultaneously. This will be under the assumption that economic analysis has shown that the mining property can be operated most efficiently using exactly two pieces of stripping equipment (e.g. draglines).

In particular, the model being considered here concerns a coal mining property whose various deposits of coal do not all meet the required quality characteristic standards. Thus, blending is mandatory.

That part of Model III from Chapter III concerning the quality characteristics, A(00)-A(6), would appear for K=2 as:

(BO) Maximize $s_i x_i + s_j x_j$

Subject to:

(B1) $s_i x_i + s_j x_j \leq S$

(B2) $b_i x_i + b_j x_j \geq B$

(B3) $m_i x_i + m_j x_j \leq M$

(B4) $a_i x_i + a_j x_j \leq A$

(B5) $x_i + x_j = 1$

(B6) $x_i \quad x_j \geq 0$

Where i and j are any two pits in the deposit and $i \neq j$.

It will be shown that constraints (Bl) - (b6) can be revised so that many of the questions concerning feasibility of the blending of two particular pits can be made by visual inspection. Let q_i denote any quality characteristic (sulphur, BTU, moisture, or ash content) of pit i. A manipulation of q_i will be shown that reduces the quality parameters to number that

1) Center around zero,
2) indicate better-than required levels with a negative value and,
3) indicate worse-than-required levels with a positive value.

In this fashion large numbers will not have to be analyzed in determining feasibility. Instead, one will have a group of standardized data, centered around zero, so that required blending ratios (values of x_i or x_j) can be found quickly and simply or often by inspection.

The following theorems are required to attain the desired standardization described above.

Theorem IV-1: If the inequality

$$\sum_j a_j x_j \leq Z \text{ (for Z 0)}$$

is satisfied by the values $x_1, x_2, \ldots, x_n$ where $\sum_j x_j = 1$, then the same values of x_j also satisfy

$$\sum_j \left(\frac{a_j}{z} - 1\right) x_j \leq 0$$

Proof:

$\sum_j a_j x_j \leq Z$ may be written as

$$= \frac{1}{Z} \sum_j a_j x_j \leq 1, \text{ and by transposing the right hand side,}$$

$$= \frac{1}{Z} \sum_j a_j x_j - 1 \leq 0, \text{ and since } \sum x_j = 1,$$

$$= \frac{1}{Z} \sum_j a_j x_j - \sum_j x_j \leq 0 \text{ and}$$

$$= \sum_j \left(a_j/z\ x_j\right) - \sum_j x_j \leq 0, \text{ and}$$

$$= \sum_j (a_j/z\ x_j - x_j) \leq 0, \text{ hence}$$

$$= \sum_j (a_j/z - 1)\ x_j \leq 0$$

Theorem IV-2: If the inequality

$$\sum_j a_j x_j \geq Z \text{ (for } Z > 0)$$

is satisfied by the values $x_1, x_2, \ldots, x_n$ where $\sum_j x_j = 1$, then the same values of x_j also satisfy

$$\sum_j (1 - a_j/z)\ x_j \leq 0$$

Proof: $\sum_j a_j x_j \geq z$

$$= \frac{1}{z} \sum_j a_j x_j \geq 1$$

$$= \frac{1}{z} \sum_j (a_j x_j) - 1 \geq 0$$

$$= \frac{1}{z} \sum_j a_j x_j - \sum_j x_j \geq 0$$

$$= \sum_j a_j/z\ x_j - \sum x_j \geq 0$$

$$= \sum_j x_j - \sum_j a_j/z\ x_j) \leq 0$$

$$= \sum_j (x_j - a_j/z\ x_j) \leq 0$$

$$= \sum_j (1 - a_j/z)\ x_j \leq 0$$

Let q_i be a new constant derived from $\overline{q}_i$ using Theorem IV-1 and Theorem IV-2. $\overline{q}_i$ will be referred to as the "standardized"

q_i. The desired standardization to be used is:

(1) for all equal-to-or-less-than quality constraints, the value of quality q in pit i is standardized by

$$\bar{q}_i = \frac{q_i}{Z(q)} - 1 \text{ (using Theorem IV-1)}$$

where q_i is the average of the particular quality, sulphur (s_i) or moisture (m_i) or ash (a_i), in pit i and Z(q) is the required maximum of quality q. The Z(q)'s are also the right-hand-sides of (B1)-(B4). For instance, if the established maximum for sulphur is 0.6 percent and the average sulphur content of pit 3 is 0.38 percent, then

$$s_3 = .38$$

$$Z(s) = .6 \text{ and}$$

the standardized $\bar{s}_3 = \frac{.38}{.6} - 1$

$$= -.37.$$

(2) for all equal-to-or-greater-than quality constraints, the average value of quality q in pit i is standardized by

$$\bar{q}_i = 1 - \frac{q_i}{Z(q)} \quad \text{(from Theorem IV-2)}$$

where q would normally be BTU. For example, if the established minimum for BTU is 10000 and the average BTU of pit 3 is 9500 then,

$$b_3 = 9500,$$

$$Z(b) = 10000 \text{ and,}$$

$$\bar{b}_3 = 1 - 9500/10000$$

$$= .05.$$

(3) The overburden ratio of each pit can be converted to a production rate and made a smaller value by

$$\overline{OB}_i = Z(OB)/OB_i,$$

Z(OB) is the overburden ratio that would allow the stripping equipment to uncover coal at a rate equal to 50 percent of the required production for the case of two simultaneous stripping activities. Suppose the required annual coal production rate is 2 mtpy, a specified dragline can move an overburden on 4.8:1 at a coal production rate of 1 mtpy (50 percent of the required 2 mtpy), and the overburden of pit 3 is 5.5:1. Then, K(OB) = 4.8 and

$$\overline{OB}_3 = \frac{4.8}{5.5}$$

$$= .87.$$

Using Theorem IV-1 and Theorem IV-2 and the above, (1) and (2), standardizing procedures, Formulation (BO) - (B6) can be expressed as:

Model IV:

(C0) maximize $\bar{s}_i x_i + \bar{s}_j x_j$

subject to:

(C1) $\bar{s}_i x_i + \bar{s}_j x_j \leq 0$

(C2) $\bar{b}_i x_i + \bar{b}_j x_j \leq 0$

(C3) $\bar{m}_i x_i + \bar{m}_j x_j \leq 0$

(C4) $\bar{a}_i x_i + \bar{a}_j x_j \leq 0$

(C5) $x_i + x_j = 1$

(C6) $x_i, x_j \geq 0.$

The pits can now be arranged in descending order according to the critical characteristic (sulphur).

As the pits are arranged in descending order, let the pits that have a standardized sulphur content equal to or greater than zero be designated by the subscript i, $\bar{s}_i \geq 0$. The pits that have a standardized sulphur content less than zero will be designated by the subscript j, $\bar{s}_j < 0$. The pits are now divided into two subsets.

Consider the objective function,

$$\text{(CO) maximize } \bar{s}_i x_i + \bar{s}_j x_j,$$

and the first constraint,

$$\text{(C1) } \bar{s}_i x_i + \bar{s}_j x_j \leq 0.$$

It is obvious from (C1) that all combinations of pits in which both pits have positive $\bar{s}_i$ can be eliminated from further consideration. It is also apparent from (CO) and (C1) that (CO) is optimal when (C1) is a binding constraint, in other words, when

$$\bar{s}_i x_i + \bar{s}_j x_j = 0.$$

When the above condition is met [(C1) is binding], the optimal solution is

$$\bar{s}_j x_j = -\bar{s}_i x_i$$

$$\frac{-\bar{s}_j}{\bar{s}_i} = \frac{x_i}{x_j}$$

If the ratio x_i/x_j is feasible for the remaining constraints, then this should be the ratio of blending used since the first constraint (sulphur in this instance) is the critical quality. It remains, then to determine either a) x_i/x_j is a feasible combination or b) x_i/x_j is not feasible and if not, what ratio (if any) would be feasible. The question of feasibility is limited at this point to the constraints (C2)-(C6).

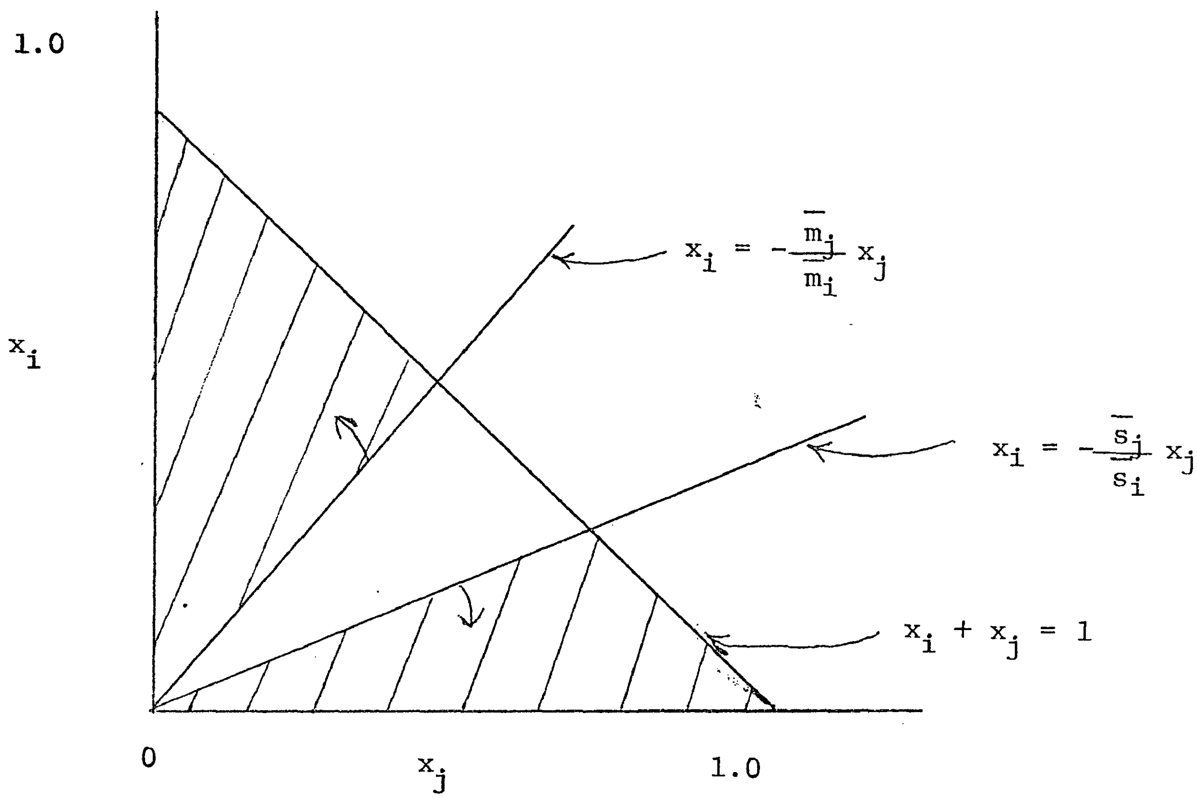
The feasibility of the quality constraints can be shown geometrically. Using the above relationship, when the constraint concerning the critical value (sulphur) is binding,

$$-\bar{s}_j/\bar{s}_i = x_i/x_j.$$

Figure IV-1 depicts, geometrically for two variables, the relationship between the critical characteristic constraint and the constraint involving any other quality, say, M. It can be seen in Figure IV-1, if the sulphur ratio of $\bar{s}_i$ to $\bar{s}_j$ is $\bar{s}_i/\bar{s}_j$, then the slope of the sulphur constraint is $-\bar{s}_j/\bar{s}_i$ and the feasible region lies below this line. Remembering that the i^{th} pit has positive $\bar{s}$ and the j^{th} pit has negative $\bar{s}$, suppose that $\bar{m}_i$ is negative and $\bar{m}_j$ is positive. The slope for this constraint is $-m_j/m_i$, however, the feasibility region for the constraint, M, lies above this line. The two areas of feasibility do not overlap and no feasible solution exists. If $\bar{m}_i$ is negative and $\bar{m}_j$ is positive,

Figure IV - I

Constraints Causing Nonfeasibility



then, $-\bar{s}_j/\bar{s}_i$ must be equal to or greater than $-\bar{m}_j/\bar{m}_i$ or $-\bar{s}_i/\bar{s}_j \leq \bar{m}_i/\bar{m}_j$ in order for the feasible region to be nonempty.

On the other hand, suppose that another quality constraint (take moisture, again) has the characteristics, $\bar{m}_i \geq 0$ and $\bar{m}_j \leq 0$. Figure IV-2 depicts this geometrically. Now the feasible region for the second constraint lies below the line with slope $-\bar{m}_j/\bar{m}_i$. Hence, the feasible regions described by the two constraints will always overlap. If $-\bar{s}_j/\bar{s}_i \leq -\bar{m}_j/\bar{m}_i$, the ratio $\bar{s}_j/\bar{s}_i$ is also feasible for the other quality constraint. However, if $-\bar{s}_j/\bar{s}_i \geq -\bar{m}_j/\bar{m}_i$, then the ratio $\bar{m}_j/\bar{m}_i$ is the highest ratio that is feasible for both. Therefore, if m_i is positive and m_j is negative and 1) $-\bar{s}_i/\bar{s}_j \geq -\bar{m}_i/\bar{m}_j$, then $-\bar{s}_i/\bar{s}_j$ is a feasible ratio, but if

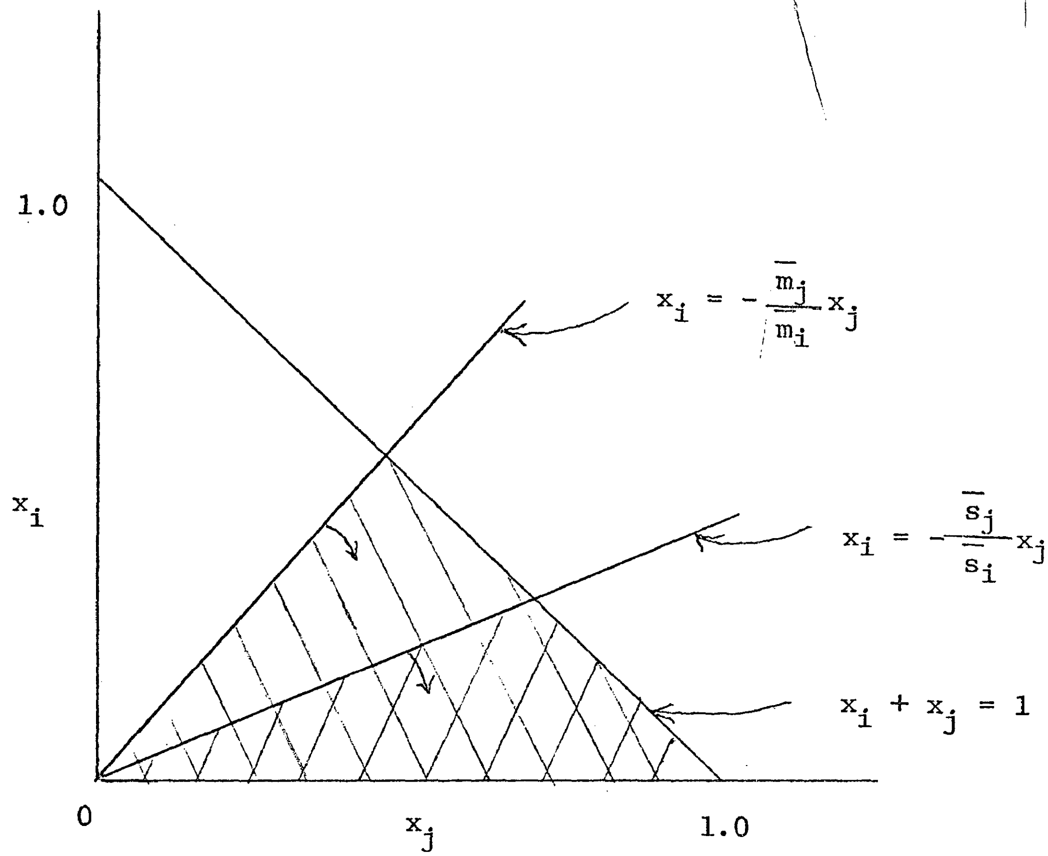
2) $-\bar{s}_i/\bar{s}_j < -\bar{m}_i/\bar{m}_j$

then, $-\bar{m}_i/\bar{m}_j$ is the highest common feasible ratio.

The production rate requirement follows a pattern different from the quality standard requirements. Recall that the standardized overburden ratio, $\overline{OB}_i$, represents the proportion of 50 percent of the annual tonnage requirement of coal that can be uncovered by a particular dragline or other stripping equipment in pit i with overburden OB_i. If the stripping activity is to operate at a rate equivalent to the

Figure IV - 2

Constraints With Overlapping Feasibility



production activity, then it must be uncovering an amount of coal equal to the annualized production requirement. There are three situations that can exist.

Case I: If the ratio of production that is desired or required by the quality constraints is equal to the standardized overburden ratio of two pits,

$x_i/x_j = \overline{OB}_i/\overline{OB}_j$,

then both draglines can operate at maximum speed and the ratio x_i/x_i is feasible if that ratio results in the required annual production rate. This can be determined by:

$$\overline{OB}_i + \overline{OB}_j \geq 2 \ ?$$

The value 2, in the above inequality, represents the "standardized" production rate in that if $\overline{OB}_i = 1$, then the dragline assigned to pit i can uncover 50 percent of the required annual coal production.

Case II: If $x_i/x_j < \overline{OB}_i/\overline{OB}_j$, then pit j with its better-than-required sulphur content, must be producing at a maximum rate and pit i decreased accordingly in its production rate. Because pit i has positive sulphur, the ratio x_i/x_j is a maximum and cannot be increased. The feasibility of the annual production requirement can be determined by the decision question in this case:

$$\overline{OB}_j + x_i/x_j \ \overline{OB}_j \geq 2 \ ?$$

If this is not true, the pits i and j cannot be used as a combination because the allowable stripping capacity is less than the production rate.

Case III: If $x_i/x_j > \overline{OB}_i/\overline{OB}_j$, then pit i must be producing at a maximum rate and pit j adjusted accordingly in its production rate. The feasibility of the annual production requirement can be determined in this case by:

$$\overline{OB}_i + x_j/x_i \ \overline{OB}_i \geq 2?$$

In this case if the answer to the above is no, the product or value $(x_j/x_i)(\overline{OB}_i)$ can be increased to as much as $\overline{OB}_j$. This is because the j^{th} pit has better-than-required sulphur. Hence, more production can be taken from pit j without violating (C1) in Model IV. If this optimal increase is utilized and the value of the left side of the inequality still does not equal or exceed two, then pits i and j cannot maintain the required production rate. The two pits, i and j, would then be removed from further consideration as a blending combination.

The solution to Model IV can be found in a stepwise procedure. This can be done in such a way that the zero-one programming elements concerning dragline movement and sequencing are handled without applying direct binary programming algorithms and their required computer time.

The preliminary procedure involves:

a) determining, from core-hole analyses, mineable pits of homogeneous coal such as in Figure III-1.

b) finding the average of the quality characteristics and the overburden in each of the pits from (a), and

c) standardizing the average of each quality characteristic in each pit per the standardization procedure discussed previously.

Assuming that sulphur is the critical characteristic, the stepwise procedure for two simultaneous stripping activities is:

Step 1) Find a least-cost path for each dragline (Gavett's method is used here) from the appropriate cost matrix of moves between pits.

Step 2) Find, among the pits remaining for consideration, the next pair of pits from step 1. (a) If both pits have positive $\bar{s}$, go to step 10. (b) If both of the pits have negative $\bar{s}$, go to step 3. (c) If one pit has positive $\bar{s}$ and the other negative $\bar{s}$, designate them $\bar{s}_i$ and $\bar{s}_j$, respectively, and go to step 4.

Step 3) A subjective decision must be made at this point. (a) If it is felt that enough "slack" exists in the critical characteristic that the remaining pits can be successfully blended, substitute the next quality characteristic for $\bar{s}$ and go to step 4. (b) If it is felt that this blend would result in too much high-sulphur coal in the remaining pits, go to step 10.

Step 4) Calculate the ratio $|\bar{s}_i/\bar{s}_j|$ for this pair. If any standardized value is zero, convert that value to a very small positive number, say, .0001.

Note: This step determines the ratio of production, x_j/x_i, which would result in a usable blended sulphur content.

Step 5) Compare the remaining pairs of standardized data, $\bar{q}_i$ vs. $\bar{q}_j$, (BTU, moisture, ash) for this combination. If each of these remaining pairs of data are both negative go to step 8.

Note: This step insures that the ratio from step 4 will be a feasible blending ratio since the remaining quality characteristics are all better than the limiting constraints. Any pair of quality characteristics (q_i, q_j) that are both negative need not be considered in further steps.

Step 6) Compare the remaining pairs of standardized data for this combination. If any pair of these values are both positive go to step 10. Eliminate the combination i, j from further consideration as a possible blend.

Note: This step rejects any combination of pits in which both have the same quality characteristic that is over the constraint since it is impossible to make a combination of such pits that would be feasible. The standardized data makes this step very rapid by the fact that one is merely looking for any pair of positive values.

Step 7-a) For any remaining pairs of standardized data in which the characteristic for the i^{th} pit is negative and the characteristic for the j^{th} pit is positive calculate:

$$\left|\bar{s}_i/\bar{s}_j\right| \overset{?}{<} \left|\bar{q}_i/\bar{q}_j\right| .$$

If the inequality is not true, eliminate i, j as a possible combination and go to step 10.

Note: This step eliminates combinations which are infeasible in that no combination of these two pits would satisfy both the sulphur constraint and the constraint for quality characteristic q.

Step 7-b) For any remaining pairs of quality variables in which the characteristic for the i^{th} pit is positive calculate:

$$\left|\bar{s}_i/\bar{s}_j\right| \overset{?}{>} \left|\bar{q}_i/\bar{q}_j\right| ,$$

If the inequality is not true, replace $\bar{s}_i/\bar{s}_j$ with $\bar{q}_i/\bar{q}_j$, and either return to step 7-a if more quality pairs remain to be examined or go to step 8 if no quality pairs remained unexamined.

Note: The new ratio, $\left|\bar{q}_i/\bar{q}_j\right|$, will result in a sulphur (the critical value) better than required. The combination i, j can be feasibly blended then, but should be used only if necessary, or if its use does not unnecessarily hamper the feasibility of other combinations.

Step 8) For the overburden or stripping capacity constraint, calculate:

$$\left|\bar{s}_i/\bar{s}_j\right| \overset{?}{\leq} OB_j/OB_i$$

Then, if the above inequality is true, go to 8-a. If the inequality is not true go to 8-b.

Step 8-a) Calculate:

$$OB_i + \left|s_i/s_j\right| \quad (OB_i) \overset{?}{\geq} 2.$$

If this inequality is true, the two pits i and j represent a feasible combination. Go to step 9. If this inequality is not true, the product $\left|s_i/s_j\right|$ (OB_i) can be increased to as much as OB_j to attain the total sum of 2, but this will result in a better-than-required condition for the quality characteristics. The analyst can choose to "high grade" this combination by replacing the ratio $\left|\bar{s}_i/\bar{s}_j\right|$ by Max $(OB_i/OB_j,\ q_i/q_j$ from step 7-a) (or any ratio between the ratio and the above maximum ratio which will meet the production requirement.)

Step 8-b) Calculate:

$$OB_j + \left|\bar{s}_j/\bar{s}_i\right| \quad OB_j \overset{?}{\geq} 2$$

If this inequality is true, then pits i and j represents a feasible combination. If the inequality is not true, eliminate i, j as a possible blending combination and go to Step 10.

Note: This step eliminates those combinations in which the ratios necessary to meet the quality constraint will not allow the necessary production rate to be maintained by an equal amount of uncovered coal.

Step 9) Distribute the tonnage, T_i and T_j, from this combination according to the ratio $\bar{s}_j:\bar{s}_i$ until at least one pit is depleted. Eliminate the depleted pit from further consideration and take the remaining tonnage from the other pit to step 2.

Step 10) Increase the cost of moving one dragline to its current position from its previous position to a very large number. Return to step 1.

This procedure is followed until a) all pits with positive characteristics are depleted in step 9 in which case the remaining pits would be checked against step 8 for feasibility of production rate, or b) a high tonnage of coal is left unproduced and infeasible in which case the analyst should investigate other alternatives to choices utilizing step 7-b and step 8-a. If this discloses no improvement, the property is not mineable with the 2 specified draglines. Elimination of the pit with the highest critical value and a repetition of the procedure should reveal the extent of necessary additional stripping equipment and/or the amount of coal to be produced for alternative purposes (with less stringent standards).

Demonstration of the Method

To demonstrate the steps involve, a simplified mining property will be utilized. Figure IV-3 represents the dispersion and data required for this demonstration problem. Suppose that two identical draglines are being proposed for the project, each of which could uncover .5 mtpy in an overburden of 10:1. The required annual production rate is 1.0 mtpy. Also, suppose the natural barrier is a railroad track with heavy traffic. The basic requirements of the produced coal will be taken to be:

Sulphur - maximum of 1.0%

BTU - minimum of 10,000

Moisture - maximum of 10%

Ash - maximum of 10%

Production - 1.0 mtpy

The standardized data for the 6 pits involved would be as shown in Table 4.1. The data there is arranged in descending order of sulphur content.

Figure IV- 3

LAYOUT FOR LOCATION AND PERTINENT DATA FOR PITS

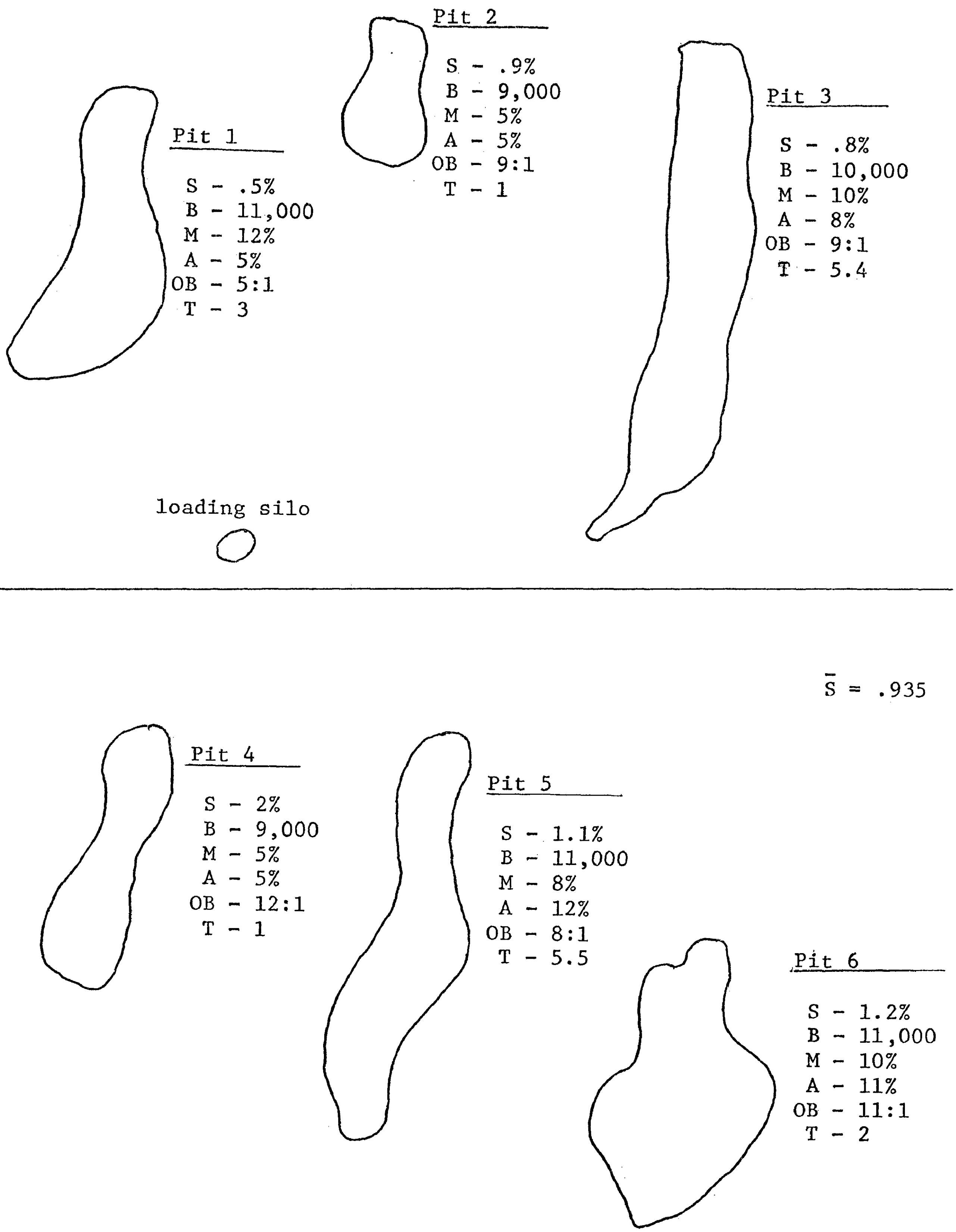


TABLE 4.1

Standardized Data for Figure IV-1

Pit	Sulphur	BTU	Moisture	Ash	Overburden
4	+1.0	+.1	-.5	-.5	.8
6	+ .2	-.1	0	+.1	.9
5	+ .1	-.1	-.2	+.2	1.2
2	- .1	+.1	-.5	-.5	1.1
3	- .2	0	0	-.2	1.1
1	- .5	-.1	+.2	-.5	2.0

FIGURE IV - 4

Cost Matrices for Dragline Movements Between Pits
(Separated by Natural Barrier) and Resulting Paths

Cost Matrix North Pits

From Pit \ To Pit	0	1	2	3
0	x	10	20	15
1	99	x	5	10
2	99	5	x	4
3	99	10	5	x

Cost Matrix South Pits

From Pit \ To Pit	0	4	5	6
0	x	3	5	7
4	99	x	6	9
5	99	6	x	4
6	99	9	6	x

Path for north dragline:

Construction to 1 to 2 to 3

Path for south dragline:

Construction to 4 to 5 to 6

(Gavett's Method)

Using the above two paths (Step 1), Step 2 and Step 4 make the absolute value of the ratio, 1/-.5 = 2.0.

Following steps 5 and 6, it can be readily seen that the pairs (+.1, -.1), (-.5, +.2), and (-.5, -.5) are neither all both negative nor any both positive. (For example, though, the combination of pits 4 and 2 could not be made because both have a positive value for BTU.)

Step 7-a would be applied to the moisture characteristic. The comparison of the ratios

$$1.0/-.5 < -.5/+.2$$

$$2 < 2.5,$$

is true so that this combination is still feasible. Notice that with the small standardized numbers, these comparisons can be made visually. However, they will be written out here for demonstration purposes.

Step 7-b would be applied to the BTU data by comparison of:

$$\left|1.0/-.5\right| \geq \left|+.1/-.1\right|$$

$$2 \geq 1$$

This inequality is true so that the BTU value is also feasible.

Step 8-a applies to the overburden data since

$$\left|1.0/-.5\right| \leq \left|2.0/.8\right|$$

$$2 \leq 2.5.$$

We then calculate,

$$.8 + 2\ (.8) = 2.4 > 2.$$

Applying step 9 with a sulphur ratio of 2:1, then, the appropriate tonnage distribution would be 1 ton from pit 4 per 2 tons from pit 1. Pit 4 has only one ton so this feasible combination would use 1 ton from pit 4 and 2 tons from pit 1 leaving 1 ton remaining in pit 1. This remaining tonnage is returned to step 2.

Pit 5 is now used to blend with the remainder of pit 1. From steps 2 and 4 the ratio

$$\left|+.1/-.5\right| = .2$$

is found. Steps 5 and 6 show neither all remaining pairs of quality data negative nor any pair of both positives.

Step 7-2 is applied to the moisture data by

$.2 \leq 1$

in which the inequality is true. Step 7-b applies to the ash data and the inequality

$.2 \not\geq .4$

is not true. Hence the sulphur ratio, 1:5, is replaced by the ash ratio, 2:5.

In step 8 the comparison

$.4 \leq 2/1.2$

is made (Note that the ash ratio has replaced the sulphur ratio) and steps 8-a shows

$1.2 + .4\ (1.2) + 1.68 < 2.$

The blending ratio can be increased from .4 to as much as 1 (the moisture ratio from step 7-2). A ratio of 2/3 will meet the production requirement. Step 9 then requires a ratio of 3:2 which distributes as 1 ton from pit 1 and 1.5 tons from pit 5. The remaining tonnage from pit 5 is returned to step 2.

From steps 2 and 4, the ratio of sulphur for pits 5 and 2 is

$$\begin{aligned} \left|s_i/s_j\right| &= \left|+.1/-.1\right| \\ &= 1. \end{aligned}$$

Step 5 removes moisture from further consideration and step 7-a shows the inequality to be true for the BTU standard:

$$1. \leq \left|-.1/+.1\right|$$

Step 7-b is utilized for the ash standard by the inequality

$$1 \geq \left|+.2/-.5\right|$$

$$1 \geq .4$$

which is a true inequality. For the required production rate, step 8 and 8-b show

$$1 > 1.1/1.2$$

and $1.1 + (1)(1.1) = 2.2 \geq 2.$

Hence, pits 5 and 2 are feasible, and the tonnage is distributed by step 9 in the ratio 1:1 or 1 ton from pit 5 and 1 ton from pit 2. This depletes the coal deposit in pit 2 and the remaining tonnage from pit 5 is returned to step 2.

The only pit remaining above the barrier is pit 3. The ratio of sulphur from steps 2 and 4 for pits 5 and 3 is

$$\left|+.1/-.2\right| = .5$$

Looking through the remaining pairs of variables, it can be seen that ash is the only point of concern. Using steps 7-b it is found that

$$.5 \geq \left|+.2/-.2\right|$$

is not true. Hence, the ash ratio, 1:1, is substituted for the sulphur ratio of 1:2.

With the new ratio from step 7-b, step 8 becomes

$|1 \leq 1.1/1.2.|$

Since this inequality is not true, step 8-b shows

$1.2 + (1)(1.2) = 2.4 \geq 2.$

Therefore, pits 3 and 5 represent a feasible combination and the tonnage distribution of 1:1 would remove 3 tons from pit 5 (depleting pit 5) and 3 tons from pit 3). The remaining tonnage from pit 3 is taken back to step 2.

The analyst would then begin step 2 with pits 3 and 6. Step 4 would yield a sulphur ratio of

$|+.2/-.2| = 1.$

By inspection (steps 5 and 6) it can be seen that the quality characteristic ash is the only variable of concern. Using step 7-b we find

$1 > 1/2.$

Hence, the sulphur ratio is feasible for all quality constraints.

Going to step 8:

$1 < 1.1/.9$

and hence, to 8-a:

$.9 + (1)(.9) = 1.8 < 2.$

The product $(1)(.9) = .9$ can be increased to OB_j or 1.1. If this is done,

$.9 + 1.1 = 2 \geq 2.$

then the sulphur ratio of 1:1 is replaced by the overburden ratio of .9:1.1 for tonnage distribution in step 9.

The above ratio applied to the tonnages yields 2 tons from pit 6 and 2.4 tons from pit 3.

The results of this are summarized in Table 4.2.

TABLE 4.2

Production Schedule for Demonstration Problem

COMBINATIONS				BEGIN	TOTAL
PIT	TONS	PIT	TONS	YEAR	TONS
4	1	1	2	0	3
5	1.5	1	1	3	2
5	1	2	1	5	2
5	3	3	3	7	6
6	2	3	2.4	13	4.4

This demonstrates the mechnics of the method. We will now look at an example of its use in a real situation.

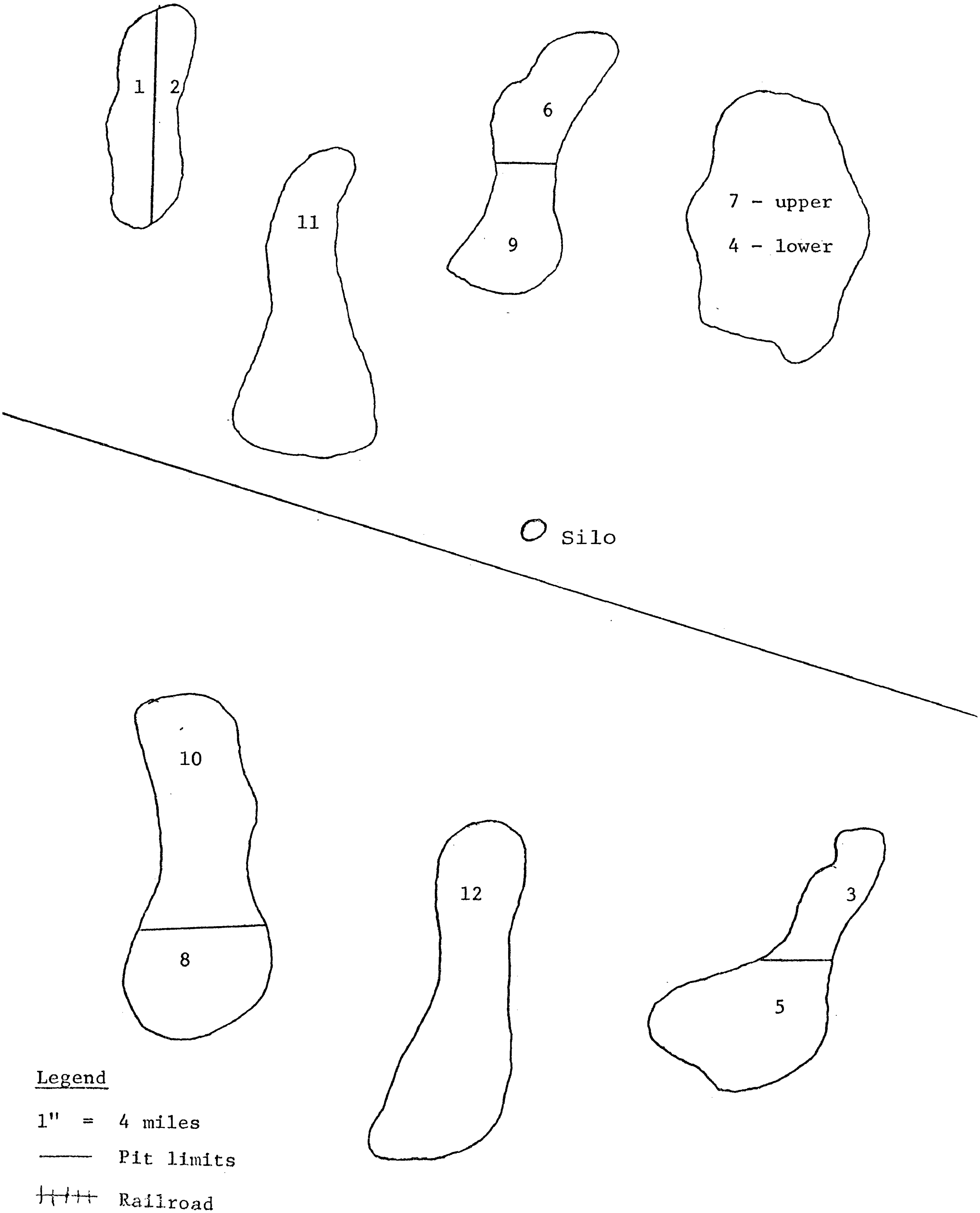
Example of Method for Mine Property

The model for this example is derived from actual working or proposed coal strip mines in the Western United States. The configuration and data have been changed to retain the confidentiality of the actual properties. TMC will be the fictitious mine name. Figure IV-5 is an outline of the mining property to be analyzed. The separate and distinct deposits have been divided into areas or pits of homogeneous

FIGURE IV-5

Mining Property of TMC With 12

Pits of Homogeneous Coal



coal quality as reflected in Figure IV-5. The averaged data for the pits is shown in Table 4.4.

The TMC property is made up of coal deposits with wide variations in quality from pit to pit. The deposits are separated by at least one mile and the surface area is rough and broken. A barrier, the railroad, also makes it extremely costly to move a dragline from one side of the barrier to the other. Assume the relevant contractual requirements for this property are as shown in Table 4.3.

TABLE 4.3

Production Requirements for TMC

Item	Requirement	
Sulphur	Maximum	1.0%
BTU	Minimum	10,000
Moisture	Maximum	10.0%
Ash	Maximum	10.0%
Annual Production	4 Million Tons	

TMC wants to mine this property with only two draglines. They are now considering two draglines that can each uncover 2 MTPY of coal with an overburden ratio of 8 to 1.

The standardized data for tables 4.3 and 4.4 is contained in Table 4.5. This table is arranged in descending order for the sulphur characteristic. Working each side of the barrier with a dragline, the previously discussed method would yield a schedule as shown in Table 4.6.

TABLE 4.4

Averaged Data for TMC by Pit

Pit	S	B(000's)	M	A	OB	Million Tons
1	2.0%	10.1	10%	5%	10:1	4.0
2	1.8	10.2	8	8	9:1	1.0
3	1.4	10.2	6	12	9:1	5.0
4	1.2	10.5	5	4	10:1	2.0
5	1.2	9.5	7	5	6:1	7.0
6	1.1	9.8	11	8	8:1	3.0
7	1.1	10.3	12	9	8:1	4.0
8	.9	10.5	6	12	8:1	3.0
9	.7	12.0	12	6	6:1	2.1
10	.7	9.7	5	10	6:1	7.0
11	.5	14.5	7	7	6:1	8.0
12	.4	10.0	9	12	4:1	8.0
Total Property	.95%	10.74	7.81%	8.63	6.9:1	54.1

Step 1, finding a least-cost path for each of the two draglines, is developed in Figure IV-6 using Gavett's method.

Table 4.6 follows the steps and procedures previously shown in this section. The procedure begins by selecting pits 1 and 12 to combine. Let dragline A be that placed on the upper side of the barrier and dragline B be that placed on the lower side of the barrier.

Table 4.6 shows that a total of 10.8 million tons can be mined from the combination A, 1 (dragline A in pit 1) and b,12 (dragline B in pit 12) in the ratio 1 ton from pit 1 per 1.7 tons from pit 12. Pit 1 will then be exhausted. Pit 2 would be the next choice for dragline A according to Figure IV-6. The stepwise procedure shows the combination A,2 and B,12 can be feasibly mined. The ratio of production is 1 ton from pit 2 per 1.2 tons from pit 12. This will yield 2.2 million tons of acceptable coal. Both pit 2 and pit 12 will be exhausted.

Figure IV-6 reveals that the next move would now be A from 2 to 11 and B from 12 to 3. In Table 4.6 this move is part of a sequence marked "omit". The procedure rapidly shows that the above move leads to an infeasible situation. The 3 lines involved to discover this are:

Move A TO	and	Move B TO
11		3
11		5
9		5

TABLE 4.5

Standardized Data for TMC by Pit

Pit	$\bar{S}$	$\bar{B}$	$\bar{M}$	$\bar{A}$	$\overline{OB}$	Million Tons
1	+1.0	-.01	0	-.5	.8	4.0
2	+ .7	-.02	-.2	-.2	.9	1.0
3	+ .4	-.02	-.4	+.2	.9	5.0
4	+ .2	-.05	-.5	-.6	.8	2.0
5	+ .2	+.05	-.3	-.5	1.3	7.0
6	+ .1	+.02	+.1	-.2	1.0	3.0
7	+ .1	-.03	+.2	-.1	1.0	4.0
8	- .1	-.05	-.4	+.2	1.0	3.0
9	- .3	-.20	+.2	-.4	1.3	2.1
10	- .3	+.03	-.5	0	1.3	7.0
11	- .5	-.45	-.3	-.3	1.3	8.0
12	- .6	0	-.1	+.1	2.0	8.0

TABLE 4.6

Life-of-Mine Production Schedule for TMC

Draglin A		Dragline B		Sulphur	Ratio	Total
Pit	Tons	Pit	Tons	Ratio	Used	Tons
1	4.0	12	6.8	1.7:1	1:1.7	10.8
2	1.0	12	1.2	1.2:1	1:1.2	2.2
11	6.0	3	5.0	1.25:1	1.2:1	11.0
11	2.0	5	3.6	2.5:1	1:1.8	5.6
9	2.1	5	3.1	1.5:1	1:1.5	5.2
11	3.9	5	7.0	2.5:1	1:1.8	10.9
11	4.1	3	3.4	1.25:1	1.2:1	7.5
9	2.1	3	1.6	1:1.3	1.3:1	3.7
7	4.0	10	4.0	1:3	1:1	
4	2.0	10	3.0	1:1.5	1:1.5	13.0
6	3.0	8	3.0	1:1	1:1	6.0

OMIT

FIGURE IV-6

TMC Cost Matrices for Dragline Movements

Between Pits and Resulting Paths

(0000's)

Dragline A

From/to	1	2	4	6	7	9	11
0	5	7	100	10	12	8	6
1	X	0	100	12	14	11	5
2	0	X	100	11	13	10	4
4	14	13	X	5	100	7	10
6	12	11	100	X	5	6	8
7	14	13	0	5	X	5	10
9	11	10	100	1	5	X	4
11	7	6	100	5	10	4	X

Path: 1 to 2 to 11 to 9 to 6 to 7 and 4

Dragline B

From/to	3	5	8	10	12
0	8	10	9	6	1
3	X	0	12	11	7
5	0	X	13	12	6
8	12	13	X	0	8
10	11	12	0	X	9
12	6	7	8	9	X

Path: 12 to 3 to 5 to 10 to 8

This leaves .3 million tons in Pit 5. The remaining alternative blends for 5 are B,5 and A,6; B,5 and A,7; B,5 and A,4. A visual inspection of Table 4.5 shows that by step 2 none of these are feasible, though. In fact, the only pit that can be feasibly combined with Pit 5 is Pit 11.

The use of step 10 would result in a change in the cost matrix of dragline movements and the resulting path. The cost of moving dragline B from pit 12 to pit 3 is changed to a large number (100). The resulting change in path for dragline B for Figure IV-6 is shown below.

from/to	3	5	8	10	12
0	8	10	9	6	1
3	x	0	12	11	7
5	0	x	13	12	6
8	12	13	x	0	8
10	11	12	0	x	9
12	100	7	8	9	x

Path: 3 to 12 to 5 to 3 to 10 to 8.

The move of dragline B from pit 12 to pit 3 is omitted and dragline B is from pit 12 to pit 5 instead, per the above change. As is shown in Table 4.6, this move results in a schedule for production that uses all of the coal available in the property. The procedure shows

Move A TO	and	Move B TO
1		12
2		12
11		5
11		3
9		3

will result in feasible combinations for blending. Note that the combinations A,11 and B,5; A,11 and B,3 are not in the ratio of the respective sulphur content. Step 8 in the method shows that feasible changes have to be made in the ratio to attain the required annual production rate in these two combinations.

The logical move for dragline A is now from Pit 9 to the multiseam deposit containing Pit 7 (the upper seam) and Pit 4 (the lower seam). This will be mined by taking, in each cut, the overburden from Pit 7 followed by production of Pit 7 followed by taking the overburden from Pit 4 followed by production from Pit 4. The method is versatile enough to handle this type of sequencing problem.

Table 4.6 shows that the two seams (Pit 7 and Pit 4) can be combined with Pit 10. Pits 7 and 10 are blended on a 1 to 1 ratio. The following cut of Pit 4 is blended with Pit 10 on a 1:1.5 ratio. This sequence is followed until Pits 4, 7, and 10 are exhausted.

The remaining pits, Pit 6 and Pit 8, can be blended on a 1:1 ratio. This provides a long-range production schedule and the conclusion that the TMC property can be mined with the proposed dragline capacity.

Table 4.6 proves that the TMC mine can be produced to meet contractual requirements using all the coal. Further refinements can be made to obtain a heuristic solution to the maximum, present-value profit. The sequence shown in Table 4.5 can be rearranged following any line in which both pits in the combination are simultaneously exhausted.

Suppose, for instance, the combination A,11 and B,5 provides higher profits than the combinations A,1 and B,12; A,7 - A,4 and B,10; or A,6 and B,8. These higher profits could be (1) because of lower cost, perhaps short cycle times, or (2) because of higher revenues, perhaps a bonus is paid for higher BTU content. The higher profits at the beginning of the mining operation could result in a higher present value for the mining property in total. In this case the sequence

Move A TO	and	Move B TO
11		5
11		3
9		3

followed by the next highest profit block would result in a heuristic profit maximization.

In this section a method has been developed which produces a production schedule for a strip coal mine using two pieces of stripping equipment. Table 4.6 required approximately 60 minutes of manual computational time to complete. No computer time was used as opposed to the methods discussed in Sections II and III which required extensive computer facilities. The nonlinear programming methods used by these previous methods have been effectively circumvented.

In Section V, refinements, variations, and extensions will be discussed.

CHAPTER V - VARIATIONS AND EXTENSIONS FOR THE PRODUCTION SCHEDULING METHOD

In the demonstration and the example presented in Chapter IV, a production schedule was developed on a life-of-the mine basis. As was indicated in Chapter I, the proposed method from Chapter IV is also versatile enough to handle short-range planning and operational planning production schedules. The method is not limited to long-range planning production schedules.

During the development and use of the method for production scheduling discussed in this paper it was found that some slack must be allowed for local variability of coal quality. The original data was divided into mineable pits of homogeneous coal quality. The quality within these pits cannot realistically be expected to be strictly constant. There will be some variance of the quality characteristics within each pit.

Overall, if 1) two pits are being blended strictly according to the ratio set for the average of the critical value, 2) the critical value follows a symmetric distribution, and 3) the blended average is set at the exact maximum or minimum acceptable critical value then, the blending of all of the two pits in total will result in acceptable coal. However, for any time periods less than the entire amount of both pits, approximately 50 percent of them will not meet the required standard for that quality characteristic. The amount of variance for the blended coal will depend on the variance of the coal in each pit and the measurement time period (daily, weekly, monthly, annually, etc.).

One way to account for this variation is to use the worst core sample in each pit instead of the average for the analysis. This would certainly result in acceptably blended coal, but almost all of the coal would be "over-standard". This procedure, obviously, could not be used in those situations in which the overall average of the mining property is anywhere close to the contractual requirements.

The suggested procedure for accounting for local variability is to use a standard for each quality characteristic, K(q), that is better than the contractual requirement for that quality characteristics. In this way, some slack is built into the procedure for those situations in which the coal quality in both pits is simultaneously locally worse than average. The amount of this increase/decrease is a subjective decision. It depends upon:

1) the degree of the penalty incurred for not meeting the standard,
2) the proximity of the average of the quality characteristic to the required standard, and
3) the stipulated time period for measuring the quality characteristic.

As a general rule, the standard deviation of the quality characteristic values within each pit can be calculated by standard procedures. The amount of "slack" can then be set at one standard deviation. This, plus the ability to alter the blending ratio for any operational planning period, will yield favorable results in the author's experience.

Of course, a complete and closely centered drill pattern for core-hole analysis will yield information to help prevent unpredicted changes in coal characteristics. A great deal of work is being done currently in the area of Geostatistics to develop adequate measures of the size and pattern of drill patterns required. Future work in Geostatistics directed toward coal mining could be beneficial to production scheduling techniques.

The method for production scheduling, as presented in Chapter IV is limited to two simultaneous stripping activities. This limitation can be partially removed by two different approaches.

In the first approach, suppose there is a large pit of coal with homogeneous quality characteristics. This pit could be of low quality coal, but it does not necessarily have to be. The situation could be one in which it has been found that the mining property cannot be fully produced with the current or proposed two pieces of stripping equipment. In this situation a third stripping and production activity could be put into the large pit previously described. A pre-determined percentage of the annual production requirement could then come from this pit. The remainder of the mining property production schedule could be devised using the method presented in Chapter IV. In this technique the required quality standards and production rate would be adjusted for the set amount of tonnage coming from the one large pit. Further, the method for production scheduling developed in Chapter IV would show when this third activity must begin.

The second approach is broader in scope than the above described first approach. When a production schedule has been devised, such as in Table 4.6, each line represents a feasible combination of two pits and the required blending ratio. The method can be directly expanded, then, to accomodate four stripping/production activities simultaneously. A pair of stripping equipment can be used on the combination indicated in each line of the production schedule. This situation could, become necessary 1) if it is felt that four small draglines would be more efficient than two very large draglines, or 2) if the production requirement is substantially increased either during planning or during actual production.

The concept that each line represents a feasible blending ratio can also be used to advantage with only two draglines in the operational planning stage. Typically, a strip coal mine is opened at the outcrop and progresses into higher overburdens at each cut thereafter. The maximum stripping rate at the outset, then, could uncover more coal than is required by the required annual production rate. This will leave, at times, excess stripping capacity. If distances between pits allow, the draglines can be operated on a cycling basis from pit to pit and a reserve of exposed coal built to allow for times when higher overburdens must be moved. In Table 4.6, for instance, dragline A could remove one cut from Pit 1 while dragline B removes one cut from Pit 12. While the coal is being removed from Pits 1 and 12, dragline A would remove one cut from Pit 11 and dragline B would remove one cut from Pit 5. The two draglines would then return to Pits 1 and 12 while the production activity takes the coal from Pits 1 and 5 in the proper ratio. This cycling operation could continue as long as feasible to prevent idle time on the draglines.

The decision to use the cycling pattern for draglines would depend on the cost of dragline idle time compared to the cost of moving the dragline repeatedly between pits. The decision for cycling would also depend on the tonnage of exposed coal desired by management.

The material and method presented here have been shown to need no computer facilities. However, computer assistance can be utilized. Appendix B is a computer program written in FORTRAN IV for generating the necessary average for each pit from the core data available and creating the standardized data used in Chapter IV.

In summary, a method of production scheduling for strip coal mines has been developed. This method works efficiently in meeting quality constraints imposed by environmental legislation concerning the emissions from burned coal Extensive computer facilities can be used, but are not necessary for the use of this method.

The method is versatile in that it can be used for planning periods of varying length. It can also be used with general or detailed information and data. The versatility can also be expanded to include more than two simultaneously producing pits under certain conditions.

A method that very efficiently works with exactly two simultaneously producing coal pits was demonstrated in

Chapter IV. Further work may develop a similar technique for more than two simultaneously producing coal pits under very general conditions.

Further work may include the development of geostatistics techniques to provide more useful input data for production scheduling. This could include a comparison of the cost of gathering more core analysis data vs. the estimated dollar value of that data to better mine planning.

It is hoped that this dissertation will not only improve the economics of strip coal mine operation now, but also will provide insights for further improvements.

APPENDICES

APPENDIX A

A Model for Sequencing Block Extraction

An excerpt from "Optimum Open Pit Mining Production Schduling" a dissertation by Thys B. Johnson for California University, May, 1968.

For the purpose of explaining the detailed structure of the matrices B_{it} it is convenient to change from the single subscript notation $h = 1,2,\ldots,H$ for each block, to a more natural 3-dimensional notation $i = 1,2,\ldots,I$, $j = 1,2,\ldots,J$ and $k = 1,2,\ldots,K$, as shown in Figure 2.6. Our block volume variables X_h^{rpt} now are designated by X_{ijk}^{rpt}.

Consider the sequence extraction constraints for block (232) (directly below block (132)) assuming we must remove the five blocks restricting it. Also assume the schedule duration to be two periods, blocks to be classifiable as ore or waste, and that it is possible to treat the ore by two different methods. Then we have the following constraints which illustrate the structural form of the B_{ip} matrices.

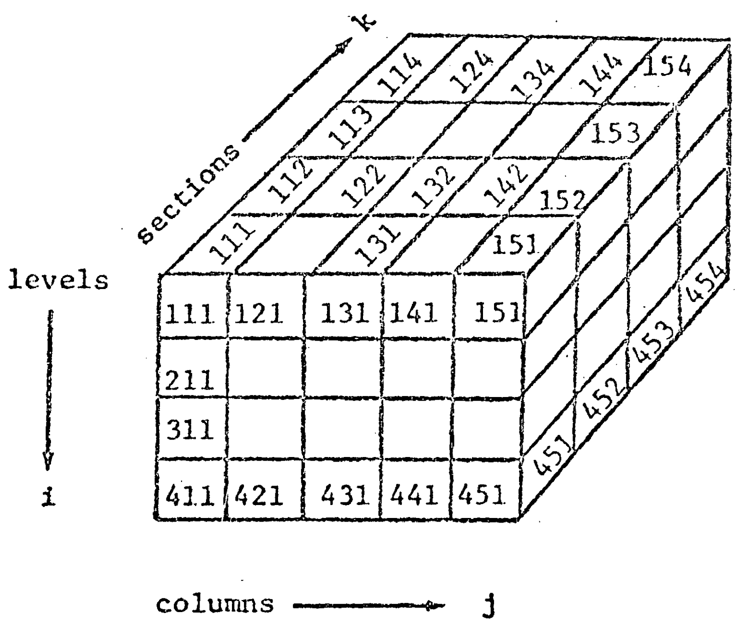


FIGURE 2.6: BLOCK STRUCTURE

(2.3.2)

$$
\begin{aligned}
&-X_{131}^{111} - X_{131}^{211} - X_{131}^{121} - X_{131}^{221} + \sum_{r=1}^{2}\sum_{p=1}^{2} X_{232}^{rp1} \le 0\\
&-X_{122}^{111} - X_{122}^{211} - X_{122}^{121} - X_{122}^{221} + \sum_{r=1}^{2}\sum_{p=1}^{2} X_{232}^{rp1} \le 0\\
&-X_{132}^{111} - X_{132}^{211} - X_{132}^{121} - X_{132}^{221} + \sum_{r=1}^{2}\sum_{p=1}^{2} X_{232}^{rp1} \le 0\\
&-X_{142}^{111} - X_{142}^{211} - X_{142}^{121} - X_{142}^{221} + \sum_{r=1}^{2}\sum_{p=1}^{2} X_{232}^{rp1} \le 0\\
&-X_{133}^{111} - X_{133}^{211} - X_{133}^{121} - X_{133}^{221} + \sum_{r=1}^{2}\sum_{p=1}^{2} X_{232}^{rp1} \le 0
\end{aligned}
$$

$$
\begin{aligned}
&-X_{131}^{112} - X_{131}^{212} - X_{131}^{122} - X_{131}^{222} + \sum_{r,p} X_{232}^{rp2} \le 0\\
&-X_{122}^{112} - X_{122}^{212} - X_{122}^{122} - X_{122}^{222} + \sum_{r,p} X_{232}^{rp2} \le 0\\
&-X_{132}^{112} - X_{132}^{212} - X_{132}^{122} - X_{132}^{222} + \sum_{r,p} X_{232}^{rp2} \le 0\\
&-X_{142}^{112} - X_{142}^{212} - X_{142}^{122} - X_{142}^{222} + \sum_{r,p} X_{232}^{rp2} \le 0\\
&-X_{133}^{112} - X_{133}^{212} - X_{133}^{122} - X_{133}^{222} + \sum_{r,p} X_{232}^{rp2} \le 0
\end{aligned}
$$

PERIOD 1

PERIOD 2

There relationship express the fact that to get at block (232) in Period 1, blocks (122), (131), (132), (133), and (142) must be removed in Period 1 irrespective of what is done with the material after it is mined. To obtain block (232) in Period 2, the sequence of blocks (122)-(142) can be removed in Period 1 or Period 2.

It is easily seen from the coefficient matrix of (2.3.2) that each B_{ip} can be expressed as:

$$B_{ip} = [E \; E \; E \; E]$$

by re-arrangement of the columns, where each E is a matrix with exactly one - 1 and one +1 in each row, and all other elements are zero. Thus E is the <u>transpose</u> of the node arc incidence matrix of a network-flow problem.

APPENDIX B

FORTRAN IV program for averaging characteristics of pits and calculating standardized characteristics at up to 75 pits.

```
      PROGRAM GREG (INPUT,OUTPUT,TAPES=INPUT,TAPE6=OUTPUT)
      DIMENSION SPLIT(75),SSULF(75),SOB(75),SBTU(75),SASH(75),
     -SMOIS(75),XPIT(75),SULF(75),XOB(75),BTU(75),ASH(75),
     -XMOIS(75(,XCYC(75),XTON(75)
C
C     *********************************************************
C     *****TABLE OF FORMAT STATEMENTS
1     FORMAT (F2.2,F2.1,F5.0,F2.2,F2.2)
2     FORMAT (A6,F3.2,F3.1,F5.0,2F4.2,F4.1,F3.1)
3     FORMAT (A6)
6     FORMAT (1H-,8X,8HPIT NAME,6X,7HSULPHUR,6X,11HOVER BURDEN,
     -6X,6HB,T.U.,6X,3HASH,6X,8HMOISTURE,6X,10HCYCLE TIME,6X,
     -10HTONS(MILS))
7     FORMAT (9X,A6,8X,F5.2,10X,F5.1,9X,F6.0,5X,F5.1,6X,F5.1,
     -10X,F6.1,10X,F5.1)
11    FORMAT(9X,A6,9X,F4.2,11X,F4.1,10X,F5.0,5X,F5.1,7X,F5.1,
     -10X,F5.1,11X,F4.1)
12    FORMAT (1H1,49X,21HSTANDARDIZATION TABLE)
13    FORMAT (1H-,11X,8HPIT NAME,10X,7HSULPHUR,10X,11HOVER
     -BURDEN,9X,6HB,T.U.,13X,3HASH,13X,8HMOISTURE)
14    FORMAT (44X,32HAVERAGE OF RAW DATA BY PIT NAMES)
15    FORMAT (12X,A6,12X,F5.2,14X,F5.1,12X,F6.2,12X,F5.1,
     -13X,F5.2)
16    FORMAT (5X,5HPITS,A6,5H AND,A6,42H ARE NOT COMPATIBLE
     -DUE TO PRODUCTION RATES)
17    FORMAT (5X,5HPITS,A6,5H AND,A6,36H ARE NOT COMPATIBLE
     -DUE TO BTU RATIO)
18    FORMAT (5X,%HPITS,A6,5H AND,A6,38H ARE NOT COMPATIBLE
     -DUE TO ASH CONTENTS)
19    FORMAT (5X,5HPITS,A6,5H AND,A6,43H ARE NOT COMPATIBLE
     -DUE TO MOISTURE CONTENT)
20    FORMAT(1H1)
C     *********************************************************
L=0
I=0
COUNT=0
TSULF=0
TBTU=0
TASH=0
TMOIS=0
SWITCH=0
```

```
C     READ RAW DATA
C     *FIRST DATA CARD MUST BE CONSTANTS USED FOR STANDARDI-
      *ZATION
C     *LAST PIT NAME SHOULD BE 999999 SHOING END OF DATA FILE,
C     *ALL DATA FROM ONE PIT AREA MUST BE GROUPED TOGETHER WITH
      *PIT NAME IN THE FIRST SIX COLUMNES.
      *DATA FOR OVER BURDEN, CYCLE TIME, AND TONAGE CAN BE
      *PUNCHED ON ONE OR ALL OF THE DATA CARDS FOR ONE PIT
      *NAME.

      READ(5,1) CSULF,COB,CBTU,CASH,CMOIS
      READ(5,2) PIT,RSULF,OB,RBTU,RASH,RMOIS,CYC,TON

65    WRITE(6,20)
      WRITE(6,14)
      WRITE(6,6)
      LINES=4
45    TPIT=PIT
      TOB=OB
      TCYC=CYC
      TTON=TON
25    COUNT=COUNT+1
      TSULF=TSULF=RSULF
      TBTU=TBTU+RBTU
      TASH=TASH+RASH
      TMOIS=TMOIS+RMOIS
      READ(5,2)PIT,RSULF,OB,RBTU,RASH,RMOIS,CYC,TON
      IF(TPIT,EQ,PIT)GO TO 25
      L=L+1
      XPIT(L)=TPIT
      SULF(L)=TSULF/COUNT
      XOB(L)=TOB
      BTU(L)=TBTU/COUNT
      ASH(L)=TASH/COUNT
      XMOIX(L)=TMOIS/COUNT
      XCYC(L)=TCYC
      XTON(L)-TTON
      COUNT=0
      TSULF=0
      TBTU=0
      TASH=0
      TMOIS=0
      IF (PIT.EQ.6H999999)GO TO 35
      GO TO 45
35    CONTINUE
      ***********************************************************
      *****SORT ROUTINE SULPHUR
      * SORTS BY FULPHUR WHEN IT IS CONSIDERED THE MAJOR CONSTRANT.
      ***********************************************************
```

```
      I=L
      DO 2000 N=1,I
      DO 2000 M=N,I
      IF (SULF(N),BE,SULF(M))GO TO 2000
      STORE=XPIT(N)
      XPIT(N)-XPIT(M)
      XPIT(M)=STORE
      STORE=SULF(N)
      SULF(N)-SULF(M)
      SULF(M)-STORE
      STORE=X)B(N)
      XOB(N)=XOB(M)
      XOB(M)=STORE
      STORE=X)B(N)
      XOB(N)=XOB(M)
      XOB(M)=STORE
      STORE=BTU(N)
      BTU(N)=BTU(N)
      BTU(M)=STORE
      STORE=ASH(N)
      ASH(N)=ASH(M)
      ASH(M)=STORE
      STORE=XMOIS(N)
      XMOIS(N)=XMOIS(M)
      XMOIS(M)=STORE
      STORE=XCYC(N)
      XCYC(N)=XCYC(M)
      XCYC(M)=STORE
2000  CONTINUE
      TO TO 2002
200   CONTINUE
2002  CONTINUE
      DO 500 L=1,I
      IF(LINES,EQ,53)GO TO 95
      GO TO 105
 25   WRITE(6,20)
      WRITE(6,14)
      LINES=4
 105  WRITE(6,11)XPIT(L),SULF(L),XOB(L),BTU(L),ASH(L),XMOIS(L),
     -XCYC(L),XTON(L)
      LINES=LINES+1
 500  CONTINUE
      ******************************************************
      *****SET UP STANDARDS TABLE                          *
      ******************************************************
      WRITE (6,12)
      WRITE(6,13)
      LINES=4
      DO 1000 I=1,L
```

```
      IF(LINES,EQ.53)GO TO 75
      go to 85
75    WRITE(6,12)
      WRITE(6,13)
      LINES=4
85    SPIT(I)=XPIT(I)
      SSULF(I)-(S*LF(I)/CSULF)-1
      IF(SSULF(I).EQ.0)SSULF(I)=-.001
      SOB(I)-(C)B/XOB(I))
      SBTU(I)=1-(BTU(I)/CBTU)
      IF (SBTU(I).BQ.0)SBTU(I)=-.001
      SASH(I)=((ASH(I)/100.)/CASH)-1
      IF (SASH(I).EQ.)0SASH(I)=-.001
      SMOIS(I)-((XMOIS(I)/100.)/CMOIS)-1
      IF (SMOIS(1).EQ.0)SMOIS(I)=-.001
      WRITE (6,15)SPIT(I),SSULF(I),SOB(I),SBTU(I),SASH(I),
     -SMOIS(I)
      IF(SSULF(I))501,501,502
502   M=I
501   LINES=LINES+1
1000  CONTINUE
      WRITE(6,20)
```

SELECTED REFERENCES

Averitt, P., 1967, Coal Resources of the United States, January 1, 1967: U.S. Geol. Survey Bull. 1275, 112 p.

Avidan, J.S., 1974, Long Range Planning of Gold Mining Using Integer and Separable Programming: Twelfth Intern. Symp. on the Application of Computers and Math. in the Minerals Ind., pp. C95-C102, Golden, Colorado School of Mines.

Axelson, H.H., 1964, A Practical Approach to Computer Utilization in Mine Planning: Quarterly of the Colorado School of Mines, Vol. 59, No. 4, pp. 593-622.

Bosman, H.C.W., 1973, Mine Evaluation and Production Scheduling (MEPS): Canadian Inst. Mining Metall. Trans., Vol. 76, pp. 123-127.

Buckingham, P.A. and Homan, H.R., 1971, Sulfur and the Energy Industry: Hydrocarbon Processing, Vol. 50, No. 8, p. 122.

Burne, G.C.S., 1972, Long Term Mine Production Scheduling: Tenth Internat. Symp. on the Application of Computer Methods in the Mineral Ind., Johannesburg, South African Inst. of Mining and Metall.

Clayton, A., 1974, Marketing--Coal Preparation--Mechanization: Mining Engineer, Vol. 134, No. 168, pp. 74-80.

Coyle, R.G., 1969, Review of the Literature on Operational Research in the Mining Industry: Inst. Mining Metall. Trans., Vol. 78, Sec. A., pp. A1-A9.

CSM Alumni Foundation, 1975, Mineral Economics Research Aided by Peter Kiewit Sons' Co.: The Mines Magazine, Vol. 65, No. 5, p. 22.

Davis, R.E. and Williams, Carl, 1973, Optization Procedures for Open Pit Mine Scheduling: Eleventh Intern. Symp. on Computer Applications in the Minerals Inc., pp. Cl-C18, Tucson, University of Arizona.

Diest, F.H., Salamon, M.D.G., MacGillivray, R.B., Hawkins, D.F., Pirow, P.C., 1971, Computer-Assisted Evaluation of Coal Reserves: Ninth Intern. Symp. on Techniques for Decision Making in the Mineral Ind., Special Vol. 12, pp. 24-74, Montreal, Canadian Inst. of Mining and Metall.

Diest, F.H. and Roberts, M.P., 1972, Aids to Planning Production in Coal Mines: Tenth Internat. Symp. on the Application of Computer Methods in the Mineral Ind., Johannesburg, South African Inst. of Mining and Metall.

Dunlap, J.W. and Jacobs, H.H., 1955, Strip Mining Phosphate Rock with Large Walking Draglines: Ops. Res., Vol. 3, No. 1, pp. 20-33.

Dupree, W. and West, J.A., 1972, United States Energy Through the Year 2000: U.S. Dept. of Interior, 53 p.

Erlandsson, Anders, 1972, An Application of Linear Programming to Investment Analysis: Tenth Intern. Symp. on the Application of Computer Methods in the Mineral Ind., Johannesburg, South African Inst. of Mining and Metall.

Ferrall, J.F., 1972, Desulfurization by Hydrogenation of Coal in Solution: Master's thesis, Colorado School of Mines.

Gambill, T.L., 1971, Coal Blending at the Navajo Mine - Four Corners Power Plant, New Mexico: Ninth Intern. Symp. on Techniques for Decision Making in the Mineral Ind., Special Vol. 12, pp. 196-204, Montreal, Canadian Inst. of Mining and Metall.

Gavett, W.J., 1965, Three Heuristic Rules for Sequencing Jobs to a Single Production Facility: Management Science, V. 11, No. 8, pp. B166-176.

Gazolla, E.A., De Oliveira Filko, F., Gazolla, G.A., Lobo, J.P.F., Yashek, M., 1973, Shovel Production Scheduling Model: Eleventh Intern. Symp. on Computer Applications in the Mineral Ind., pp. A63-A47, Tucson, University of Arizona.

Gilbert, A.G., 1975, Environmental Attitudes and Legislation: Mining Engineering, Vol. 27, No. 3, pp. 67-68.

Hewlett, R.F., 1961, Mathematical Models and Their Application to Production Problems: Computers and Computer Applications in the Mineral Industry, Univ. of Ariz., Tucson, pp. 0-1 to 0-23.

Hillier, F.S. and Lieberman, G.J., 1967, Introduction to Operations Research: San Francisco, Holden-Day, Inc., 632 p.

Hotelling, H., 1931, The Economics of Exhausible Resources: Jour. Polit. Econ., pp. 137-175.

Imaizumi, M. and Takeda, K., 1971, A Summary of a Study on the Optimal Distribution of Iron Ores: Ninth Intern. Symp. on Techniques for Decision-Making in the Mineral Ind., Special Vol. 12, Montreal, Canadian Inst. of Mining and Metall.

Jackson, Ivan, 1964, The Application of a Set of Regression Equations to a Mining System: Quarterly of the Colorado School of Mines, Vol. 59, No. 4, pp. 757-768.

Johnson, Thys B., 1968, Optimum Open Pit Mine Production Scheduling: Ph.D. Dissertation, California University, Berkeley, California.

__________, 1969, Optimum Open-Pit Ming Production Scheduling: A Decade of Digital Computing in the Mineral Industry, pp. 539-562, New York, AIMME.

__________, 1971, Optimum Design of an Open Pit--An Application in Uranium: Ninth Intern. Symp. on Techniques for Decision Making in the Mineral Ind., Special Vol. 12, Montreal, Canadian Inst. of Mining and Metall.

Jordan, P.R., 1974, Comeback for Coal: Our Natural Resources: The Choices Ahead, U.S. Dept. of the Interior Conservation Yearbook Series, No. 10.

Lambert, C. and Mutmansky, Jan, 1973, Application of Integer Programming to Effect Optimum Truck and Shovel Selection in Open Pit Mines: Eleventh Intern. Symp. on Computer Applications in the Mineral Ind., pp. A75-A105, Tucson, University of Arizona.

Lerchs, H. and Grossman, I.F., 1965, Optimum Design of Open-Pit Mines: Canadian Inst. Mining Metall. Trans., Vol. 68.

Manala, C.B., Falkie, T.V., and Su, Y.L., 1973, OPMHA - A Total Systems Simulator for Open Pit Mining: Eleventh Intern. Symp. on Computer Applications in the Minerals Ind., pp. A37-A62, Tucson, University of Arizona.

McGiddy, T.C., and Whitfield, D.B., 1974, A Computer System for the Evaluation and Planning of Coal Mines (CMEPS): Twelth Intern. Symp. on the Application of Computers and Math. in the Minerals Ind., Golden, Colorado School of Mines.

Megeath, J.D., 1975, How to Use Statistics: San Francisco, Canfield Press, 307 p.

Melaikev, N.V., 1972, The Role of Coal in the Energy Fuel Resources in the USSR: Canadian Inst. Mining Metall. Trans., Vol. 75, pp. 63-67.

Meyer, Manfred, 1969, Applying Linear Programming to the Design of Ultimate Pit Limits: Management Sci., Vol. 16, No. 2, pp. B121-B135.

National Coal Association, 1972, Bituminous Coal Facts: 95 p.

National Coal Association, 1975, Coal News, V. 4293, p. 6.

Onorofskie, S.A., 1971, Effective Utilization of Stripping Equipment: Master's Thesis, Colorado School of Mines, 96 p.

Posaner, F.M., 1974, Operations Research Techniques Applied to Mine Production Planning: Ph.D. Dissertation, University of Adelaide.

Pronk Van Hoogeveen, L.A.J., Cutland, J.R., and Weir, M., 1972, An Open Pit Design System for Stratiform Ore-bodies: Tenth Internat. Symp. on the Application of Computer Methods in the Mineral Ind., Johannesburg, South African Inst. of Mining and Metall.

Pye, R.D. and Burne, G.C.S., 1974, A Generalized Production Scheduling System: Twelth Intern. Symp. on the Application of Computers and Math. in the Minerals Ind., pp. 622-646, Golden, Colorado School of Mines.

Redmon, D., 1964, Determining Mine-Production Schedules by Linear Programming: U.S. Bureau of Mines RI 6441.

Riester, W., 1964, The Application of Mathematical Optimization Methods for the Planning of Production in the German Hard Coal Mining Industry: Quarterly of the Colorado School of Mines, Vol. 59, No. 4, pp. 641-670.

Rosenbeck, H.H., 1960, An Automatic Method for Finding the Greatest or Least Value of Function: Comput. Jour. Vol. 4, pp. 175184.

Splaine, M., Atkinson, D.C., Davison, W., and Smith, L., 1972, Optimizing Medium-Term Operations Plans for a Group of Copper Mines: Tenth Internat. Symp. on the Application of Computer Methods in the Mineral Ind., Johannesburg, South African Inst. of Mining and Metall.

Tucker, H., 1961, Computer Pitfalls: Computers and Computer Applications in the Mineral Industry, Univ. of Ariz., Tucson, pp. T-1 - T-9.

U.S. Congress, House of Representatives Committee on Interior and Insular Affairs, 1974, Surface Mining Control and Reclamation Act of 1974: H.R. No. 93-1072, 93rd Congress, 2nd Session, 293 p.

__________, House of Representatives Conference Report to Accompany H.R. 25, 1975, Surface Mining Control and Reclamation Act of 1975, Report No. 94-101, U.S. 94th Congress, 1st Session, 91 p.

__________, Senate Conference Report to Accompany H.R. 25, 1975, Surface Mining Control and Reclamation Act of 1975, Report No. 94-101, U.S. 94th Congress, 1st Session, 91 p.

U.S. Dept. of the Interior, Bureau of Mines, 1974, Assessment of the Impact of Air Quality Requirements on Coal in 1975, 1977, and 1980: Mineral Industry Surveys, Washington, U.S. Printing Office, 234 p.

U.S. Library of Congress, Science Policy Research Division, 1973, Energy Facts, 93rd Congress, 1st Session, 539 p.

Ware, T.M., 1956, Operations Research in the Mining Industry: Ops. Res., Vol. 4, No. 1, pp. 119-120.

Wilke, F. Ludwig, 1972, Capacity Calculations Investment Allocation and Long-Range Production Scheduling in German Coal Mines: Tenth Internat. Symp. on the Application of Computer Methods in the Mineral Ind., Johannesburg, South African Inst. of Mining and Metall.

Woolsey, R.E.D. and Swanson, H.S., 1975, Operations Research for Immediate Application: New York, Harper and Row, 198 p.