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COMPUTER SIMULATION OF HARD ROCK TUNNELING. VOLUME I. ANALYSIS

R. R. Hibbard, et al

General Research Corporation

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July 1972

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COMPUTER SIMULATION OF HARD ROCK TUNNELING

Final Technical Report

Volume I - Analysis

by

R. R. Hibbard
L. M. Pietrzak

AD 763563

July 1972

Sponsored by

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13. ABSTRACT

A model of the hard-rock tunneling process is developed including a three-dimensional stratified geology, a modular representation of many excavation system possibilities, and a cost-accounting system to facilitate cost-benefit analysis of tunneling system performance.

The mathematical representations of rock fragmentation, materials handling, ground support, and environmental control are given, supported by empirical data and an analysis of the physics involved.

The model is implemented in FORTRAN 4 for CDC 6000 Series Computers. A computer source tape is available from NTIS.

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14 KEY WORDS	LINK A		LINK B		LINK C	
	ROLE	WT	ROLE	WT	ROLE	WT
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Excavation						
Tunneling						
Computer Simulation						

FOREWORD

This report on a systems study and computer simulation of rapid underground excavation, prepared by staff members of General Research Corporation (GRC), documents a contract which is part of the Advanced Research Projects Agency's (ARPA) program in rock mechanics and rapid excavation.

GRC's major tasks in this program include the following:

1. General investigation of the nature of the excavation process in itself, and as an element in the total underground construction setting
2. Functional separation of the excavation process into its basic elements of rock fragmentation, materials handling, ground support, and environmental control, and an analysis of these elements to establish mathematical representations of performance and cost
3. Identification of the major interrelationships among these basic elements of the excavation process which must be accounted for in any system evaluation
4. Development of a computer simulation to estimate the performance and cost of alternative excavation methods including conventional and some novel and advanced techniques

The overall objective of this research program is to identify specific excavation systems and methods which may be substantially faster and more economical for underground excavation of deep hard rock than those utilized in the past.

Volume I of this report documents the analytical approach we have taken to modeling excavation. Volume II presents additional information and provides a user's manual for the computer subroutines which we have produced.

The authors greatly appreciate the helpful cooperation and assistance received from the many persons we contacted in the course of this study. C. S. Robinson of Robinson and Associates and L. Heflin of the Washington Metropolitan Area Transit Authority provided insight into the geological exploration aspect of project planning. K. Fox and staff of the U. S. Army Corps of Engineers, Protective Structures Division, identified some of the major concerns of military facilities planners and estimators. The Bureau of Reclamation in general, and M. E. Kiplinger and A. S. D'Alessandro in particular, provided useful data from their files. C. Crane of the Robbins Co. and G. Wickham of Jacobs Associates provided a perspective from the equipment developer's and project manager's points of view. J. Watson of Physics International and D. Nixon of Westinghouse discussed pros and cons of some of the novel excavation techniques. R. Dick of Twin Cities Mining Research Center of the Bureau of Mines was particularly helpful in evaluating drill-and-blast excavation. Ray Moran of Moran Engineering Co. supplied information on rail systems for materials handling. Other staff members at each Mining Research Center of the Bureau of Mines who provided briefings on their projects were invaluable in assessing the state of the art and the prospects of advances to come in underground excavation. The opinions expressed in this report are, of course, solely the responsibility of the authors and General Research Corporation.

ABSTRACT

A model of the hard-rock tunneling process is developed including a three-dimensional stratified geology, a modular representation of many excavation system possibilities, and a cost-accounting system to facilitate cost-benefit analysis of tunneling system performance.

The mathematical representations of rock fragmentation, materials handling, ground support, and environmental control are given, supported by empirical data and an analysis of the physics involved.

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FIGURES (Cont.)

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TABLES (Cont.)

TABLES (Cont.)

SUMMARY

This report describes the development of a computer model which is intended to be used to aid in the analysis of the relative cost and performance characteristics of existing and technically advanced tunneling methods. The structure of the model has been kept as simple, straightforward, and general as possible so that alternative tunneling methods under consideration can be quickly, easily, and economically compared by varying selected characteristics of the tunneling method and environment while keeping other characteristics constant. A considerable amount of thought has been expended in designing the model so as to leave open every possible avenue for future modifications to include any novel or hypothetical excavation method one may wish to assess.

The tunneling model is designed as a time-step simulation program. Each step in the simulation is considered to occur from one well-defined time and state of existence to another well-defined time and state of existence. The data processing activities which are required in order to proceed from one time and state to another are performed by subroutines which correspond to the various activities which must be performed in the tunneling process which is being simulated. Interaction between the activities of the tunneling process is considered to be of major importance in the model design and is treated with particular care.

A separate part of the total model is the geology model, a convenient tool for building a file of data which represents the geological conditions found within a given three-dimensional region of rock. The geology model can be used to model a geological region having up to 25 different zones of rock; each zone may be of different shape and have rocks with differing properties. A three-dimensional region was modeled to aid in the eventual consideration of shafts, caverns, and alternative tunnel routes.

The tunneling process itself is divided into four distinct functional elements for the purpose of describing the analysis and modeling process which has been performed. These elements are rock fragmentation, materials handling, ground support, and environmental control. The greatest emphasis of the study has been on investigating rock fragmentation and materials handling. The following summary of activities which are included in the simulation program for rock fragmentation and materials handling will given an impression of the scope of the tunneling model:

<u>Drill and blast fragmentation</u>

Subroutine MOVEIN moves drill jumbo to face
Subroutine HOLBRN drills holes for burn cut
Subroutine SETCHG sets explosive charge
Subroutine MOVOUT moves drill jumbo from face
Subroutine DREPAR accounts for maintenance and repair

<u>Boring machine fragmentation</u>

Subroutine ASSEMBL sets up boring machine
Subroutine BORE bores rock by cutter head rotation
Subroutine CUTTER accounts for cutter change and wear
Subroutine DISASM disassembles machine
Subroutine REPAIR accounts for maintenance and repair

<u>High velocity water jet fragmentation</u>

Subroutine JETIMP fragments rock by continuous or intermittent water jet
Subroutine JETAGN repositions water jet equipment
Subroutine JETMNT accounts for maintenance and repair

<u>Projectile fragmentation</u>

Subroutine PROJTL fragments rock by projectile impact
Subroutine PRJBR accounts for barrel wear and replacement
Subroutine PRJAGN repositions projectile equipment
Subroutine PRJMNT accounts for maintenance and repair

<u>Rail transport</u>

Subroutine RAILHL updates train status accounting for loading, dynamics, and switching logistics
Subroutine RAILEX accounts for track laying, train and switch addition
Subroutine ADDTRN accounts for logistics of train leaving discharge or maintenance areas
Subroutine RAILDS unloads train
Subroutine RAILMT accounts for maintenance and repair

<u>Truck transport</u>

Subroutine TRUKHL updates truck status and accounts for loading and dynamics
Subroutine TRUKEX accounts for system extension, road ballast and truck addition
Subroutine TRUKDS unloads truck
Subroutine TRUKMT accounts for maintenance and repair

<u>Conveyor transport</u>

Subroutine TRNPRT accounts for muck transport
Subroutine EXTNSN extends system
Subroutine SURGE accumulates and discharges muck from surge bin
Subroutine BELT accounts for maintenance and repair

<u>Machine loaders and shovels</u>

Subroutine MUKLOD loads muck at face and unloads into main line system
Subroutine MUKIN moves loaders to face
Subroutine MUKOUT moves loader from face
Subroutine MUKMT accounts for maintenance and repair

<u>Integrated conveyor loader</u>

Subroutine CVLOAD loads main line muck transport system
Subroutine CVLMT accounts for maintenance and repair

We have confined our modeling of ground support to a comparatively simple approach which provides to the overall excavation model a representation of ground-support selection, design, and installation which will allow valid comparison between different excavation methods. The ground-support techniques which are modeled include the installation of steel rib sets, rock bolts, shotcrete, and combinations of these with blocking and lagging or additional support provided if needed.

The three primary aspects of environmental control included in the simulation are ventilation, mechanical cooling, and water removal. Each of these aspects is modeled giving consideration to the problems associated with deep tunnels, high rock temperatures and novel excavation techniques.

I. OVERVIEW AND EXCAVATION MODEL FORMULATION

A. INTRODUCTION

Current capabilities for excavation seem to be limited to 200-300 ft per day in soft rock by mechanical borer and 70 ft per day in hard rock by drill-and-blast technique. Future civilian and military requirements for excavation may demand rates two to three times more rapid than is presently possible and would require lower unit costs than are now attainable. This need for improved underground excavation techniques has been documented in recent reports of studies undertaken by the National Research Council,[1,2] the Organization for Economic Cooperation and Development (OECD),[3] the Underground Construction Research Council of ASCE,[4] and many other groups.[5]

The approach taken in this report is a systems approach to achieve

1. The identification of the performance and cost characteristics of current excavation systems
2. The development of analytical tools, particularly computer models and mathematical representation, to quantify the improvements (performance and cost) to be derived from

    New excavation techniques

    Balanced and compatible system design

    Improved equipment reliability

    Mechanization-automation

    Improved scheduling including planned maintenance

    Improved geological surveying, prediction, and in situ measurement

In our semiannual report[6] the preliminary approach to computer simulation development was documented. Briefly, the development proceeded as follows. Survey and data gathering was performed to assess the military requirements for underground excavation.[7] As a result it was concluded that tunneling should receive first attention, to be followed subsequently by investigation of deep shaft drilling and finally cavern excavation. During this data-gathering phase, information on present capabilities of various excavation systems was assembled and research in progress on excavation problems was identified.

In parallel with data gathering, an early conceptual formulation of a tunneling system model was begun and eventually led to the structure and logic of the model as it now exists. Three features of this model may be mentioned here to orient the reader to the general nature of the simulation. First, it is a time-based computer simulation for the change of status within the tunnel sequentially at equal steps in time (a 6-min interval is the order of magnitude normally considered). Second, the computer model is modular; therefore, the simulation of an entire excavation system consists of an assembly of many subroutines, each of which represents a separate and distinctly identifiable activity in the excavation process (e.g., the changing of cutters on a boring machine is modeled by one subroutine). Third, the advanced and novel excavation techniques are included in the model (in addition to conventionally utilized techniques) in order to estimate their impact on the ability to excavate hard rock.

As it became possible, as a result of the data gathering, conceptual formulation, and technical analysis, software was developed to model the most important aspects of the tunneling process. Mathematical representations of physical processes were derived, computer logic was constructed, and test cases were validated.

B. THE EXCAVATION PROCESS IN THE TOTAL CONSTRUCTION SETTING

This section presents a perspective of excavation from a broad point of view to review some of the many factors to the actual physical excavation process itself and not directly modeled by our computer simulation, but which may affect the cost and performance of a tunnel excavation system.

Figure 1 represents the sequence of milestones and activities that are typically followed in the total process of constructing a tunnel underground. The excavation simulation discussed subsequently in this report is focused directly on Activity No. 6 in Fig. 1. In this study, no attempt was made to model directly either the preconstruction stage activities or the secondary phase construction (Activity No. 7). Some of the general factors affecting the design, operation, and cost of excavation are listed in Table 1. While the simulation accounts for most of these factors, the analyst should be wary of applying the simulation results to a specific project without considering the impact of these factors in greater detail.

Figure 2 represents the general flow of activities associated with the primary construction phase of a typical tunneling project. This figure is a more detailed representation of Activity No. 6 of Fig. 1. Figure 2 shows further what is and is not incorporated into the model. The construction of access portals or shafts has not been modeled and is left for future development of the simulation. Major failure or redesign of the tunneling system is also not included. Simulation of minor system breakdown and repair (Activity No. 5) is accounted for by providing operational availability factors for the various equipment. The decision logic is built to allow the appropriate time delays and added costs associated with these failures to be incurred according to these user-specified factors.

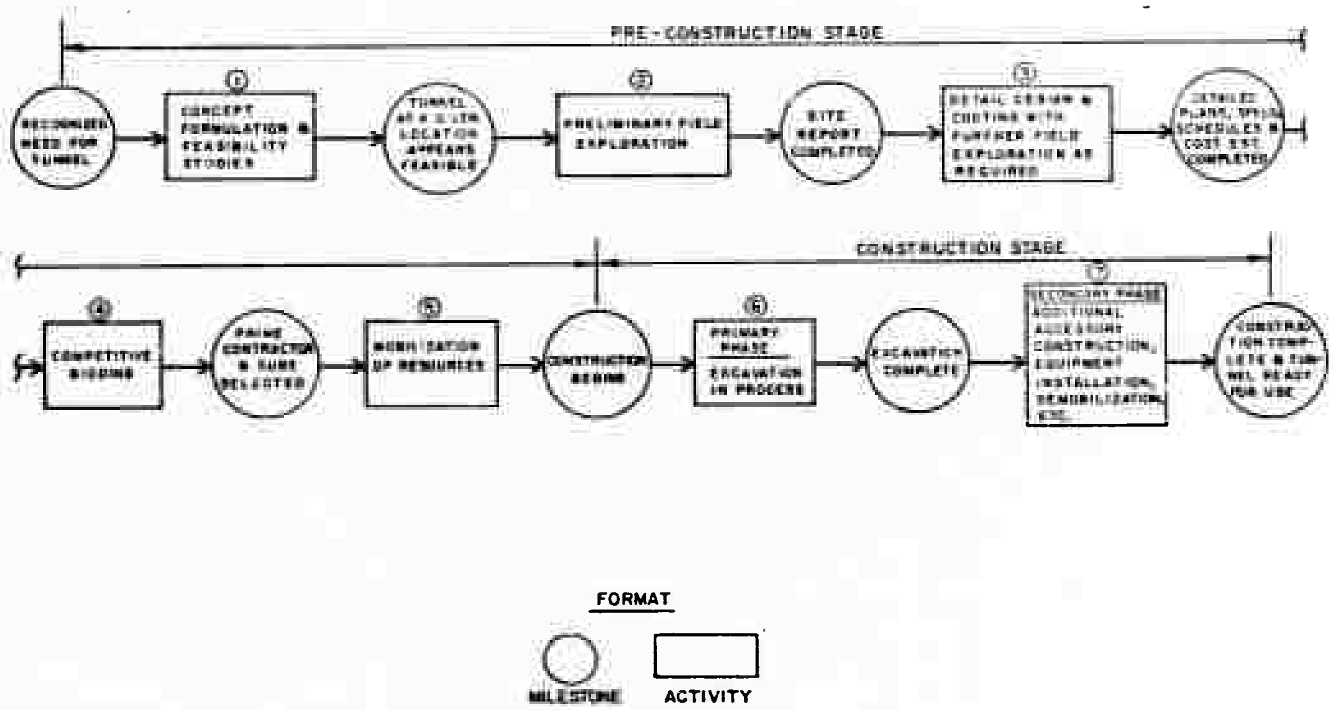

Figure 1. The Excavation Process as an Element in the Total Underground Construction Process

TABLE 1

GENERAL FACTORS AFFECTING THE DESIGN, OPERATION, AND COST OF EXCAVATION

Physical Factors

LOCATION & ACCESSIBILITY

- Urban
- Rural

GEOLOGY & HYDROLOGY

- Rock or soil type, structure, properties
- In situ stress conditions
- Subterranean temperature
- Location & variation of phreatic surface
- General flow conditions
- Geological surveying & prediction

GENERAL ENVIRONMENT

- Climate
- Altitude

OPERATIONAL REQUIREMENTS

- Intended use (military, water conveyance)
- Operational life (permanent, temporary)
- General configuration (no. of tunnels & proximity, geometry, etc.)
- Depth, alignment, grade requirements
- Environmental control requirements (ground water, air quality, etc.)

Economic-Political Factors

AVAILABILITY & COST OF RESOURCES IN PROJECT TIME FRAME

- Labor
- Material
- Equipment
- Financing

LEGAL & ORGANIZATIONAL

- Health & safety requirements
- Union demands
- Contractual
- Management & scheduling

FLEXIBILITY OF COMPLETION DATE

- Military threat
- Impact of delays

Technical Factors

- Geological surveying & prediction techniques
- Accepted design practices

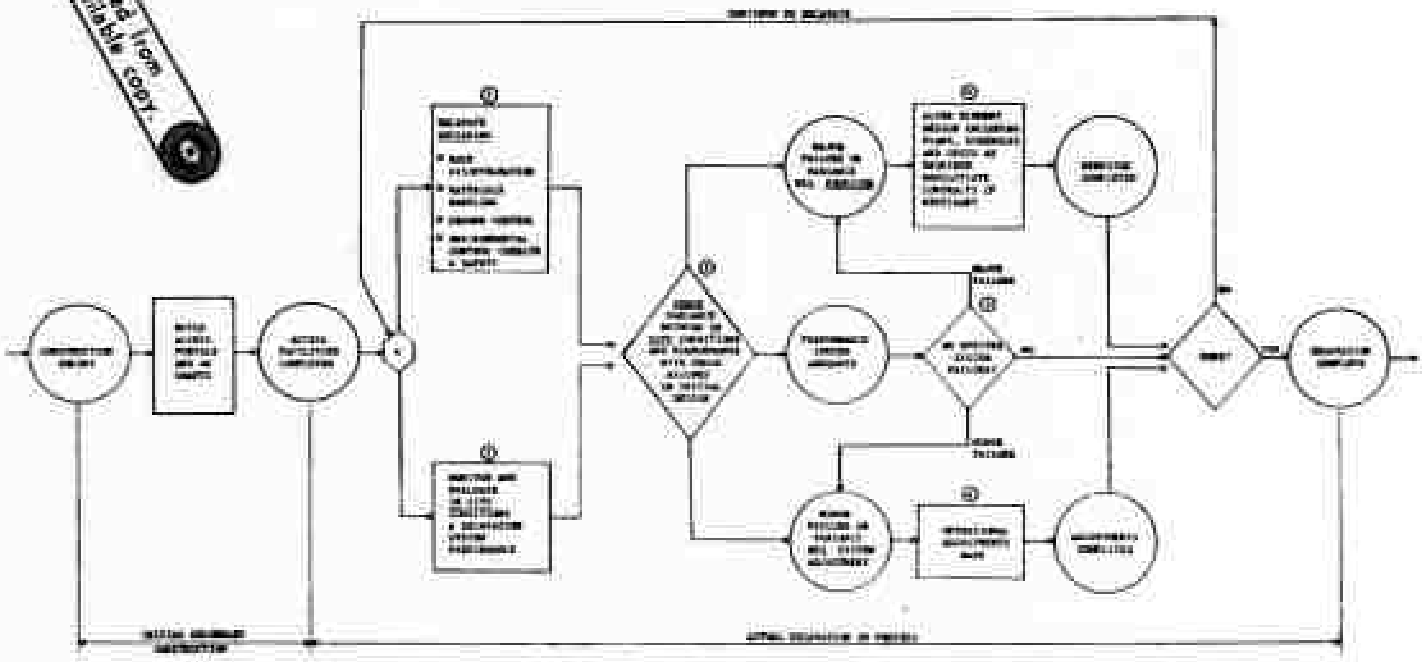

Figure 2. Primary Construction Phase

The major programming effort has been expended modeling those elements of rock disintegration, materials handling, ground control, and environmental control (Activity No. 1) of various excavation systems.

C. DESIGN CONCEPT OF THE EXCAVATION MODEL

The excavation model is intended to be used as a tool to aid in the analysis of the relative costs and performance characteristics of existing and technically advanced tunneling methods. The structure of the model has been kept as simple and straightforward as possible so that users can concentrate on solving problems associated with tunneling methods rather than trying to unravel software intricacies. Also, the model structure has been kept general to facilitate future modifications. Finally, in line with the objective of generality, the model design permits the tunneling method inputs to be quickly, easily, and economically changed. This design facilitates comparisons of alternative methods by varying selected characteristics of the tunneling method and environment while all other characteristics are kept constant.

The basic structure of the excavation model is shown in Fig. 3. The dashed lines indicate the flow of information; the solid lines indicate the sequence in which processing takes place. (This convention will apply throughout this section of the report.) The three separate parts of the excavation model are executed serially.

The geology model is used to produce detailed and consistent representations of realistically complex geologies, in a convenient manner. It is intended to be flexible enough so that it can be used to produce reasonable approximations of known geologies. It produces a geology file of a given region and miscellaneous output reports. The geology file is reusable.

The operation of the geology model involves specifying rock strata surfaces, specifying rock properties by stratum, forming the surfaces

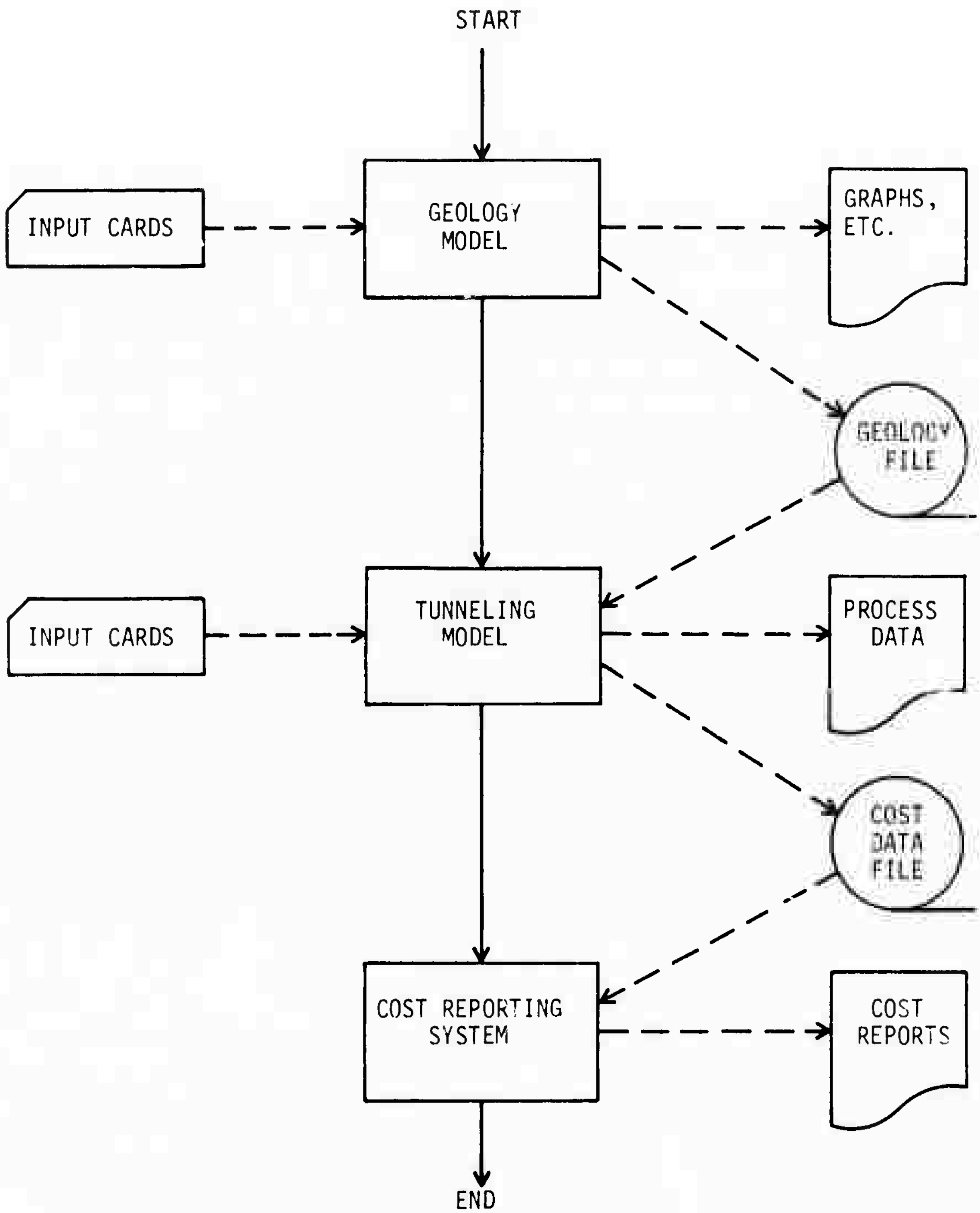

Figure 3. Excavation Model

required, and then reordering the information to produce a geology file which can be accessed by geographical coordinates. The object is to model realistically complex geologies in three dimensions, in a reasonable manner, without a great expenditure of computer time.

The tunneling model is used to simulate any one of many excavation methods, including interactions with the geology of the region and interactions among the various activities involved. It accepts information from both the geology file and from input cards, which specify the geometry and coordinates of the desired tunnel and control information required by the particular excavation method used. The output of the tunneling model consists of reports concerning the operation and progress of the tunneling simulation, as well as a file of cost information.

The tunneling model is designed as a time-step simulation program. Each step in the simulation is considered to occur from one well-defined time and state of existence to another well-defined time and state of existence. The data processing activities which are required in order to proceed from one time and state to another are performed by subroutines which correspond to the various activities which must be performed in the tunneling process which is being simulated. A control program is provided to coordinate the activities of the subroutines; considerable effort has been exerted to keep the logical design of this control program as simple as possible.

The functional separation of the tunneling process into a set of activities which are modeled by subroutines is shown in Fig. 4: A hierarchy is illustrated which classifies each activity by its association with a general process and element as described in detail in the earlier semiannual technical report.[6] Briefly, the following terminology and definitions review the basic classification scheme used:

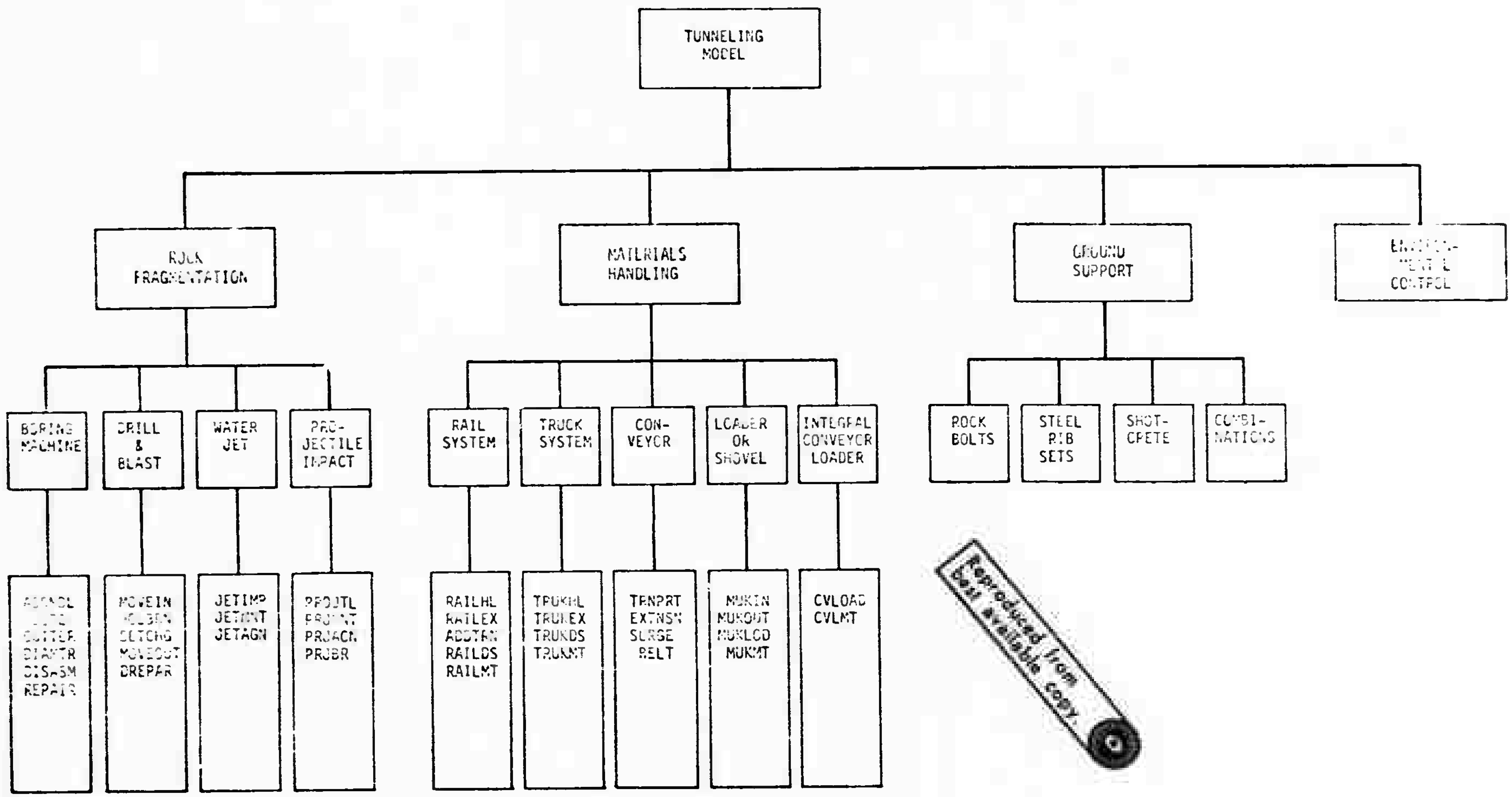

Figure 4. Tunneling Model Structure

1. EXCAVATION. That portion of the total effort of constructing a hard-rock tunnel which directly and physically contributes to the removal of the rock and the preparation of the resulting empty space for use as a tunnel.

2. ELEMENT. Functional breakdown of the overall excavation effort at the most general level. This normally includes:

   Rock Fragmentation. Breaking the rock at the tunnel face.

   Materials Handling. Carrying broken rock (muck) away from the face, or construction materials to the face.

   Ground Support. Reinforcing or supporting the ground around the excavation and installing permanent lining.

   Environmental Control. Control of undesirable gases, fumes, dust, water, heat, etc., within the excavation.

3. GENERAL PROCESS. A general process is a way in which the function of a particular element of the excavation process might be performed. It is the next level of detail within a given element. For example, the element rock disintegration might be accomplished by the general process of drill and blast, boring machine, water jet, or projectile impact as shown in Fig. 4.

4. ACTIVITY. Activities are those operations included within the performance of a specific general process. For example, the general-process boring machine includes the activities of maintaining, boring, cutter changing, etc. These activities, which are modeled by subroutines, are the basic building blocks of the computer model.

5. TECHNIQUE. A technique is a manner in which a specific activity might be accomplished. For example, the activity of boring might be accomplished by a rolling disc cutter boring

machine or a carbide insert cutter boring machine. A revision of one subroutine would be the primary change required to convert the system simulation from one technique to another.

Each of the activities involved is simulated by a family of alternative subroutines, which are coordinated in order to simulate the general processes used in a particular excavation system design. It is possible to simulate alternative excavation systems by exchanging subroutines or by coordinating them in different manners, without the need for extensive reprogramming.

Note that this logical arrangement of subroutines does not require that information be exchanged only "up and down the branches" of the tree structure. If a common information exchange area is provided, it is possible for one activity to influence another even though they are not in the same general process structure. For example, rock disintegration by projectile impact or water jet might lead to the presence of large amounts of heat or water in the tunnel, which would then have to be removed by the general processes used for environmental control. This interaction between the activity subroutines is depicted schematically in Fig. 5. The ability to model large numbers of interactions of this kind is considered to be one of the advantages of using a computer to simulate the excavation effort.

The kinds of interaction between elements of the tunneling model that are currently included in the tunneling model are shown in Fig. 6. Interactions which should be considered in addition to those modeled are parenthetically included. These would be considered for modeling as further development of the model proceeds. Figure 6 shows, for example, that the impact of rock fragmentation on materials handling is due to heading advance (the materials handling system must also advance) and rock volume (the materials handling system must have sufficient capacity

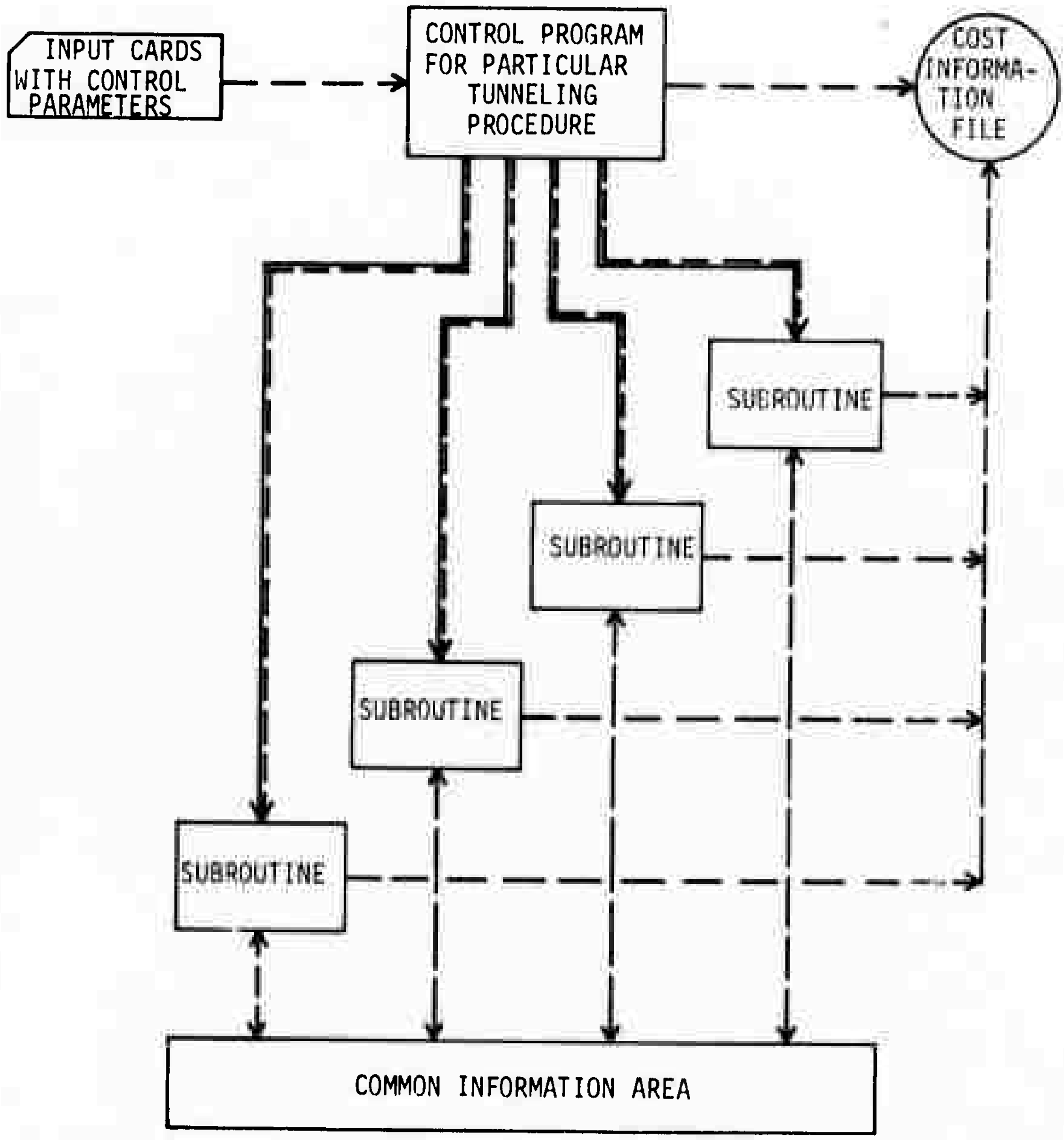

Figure 5. Activity Interaction

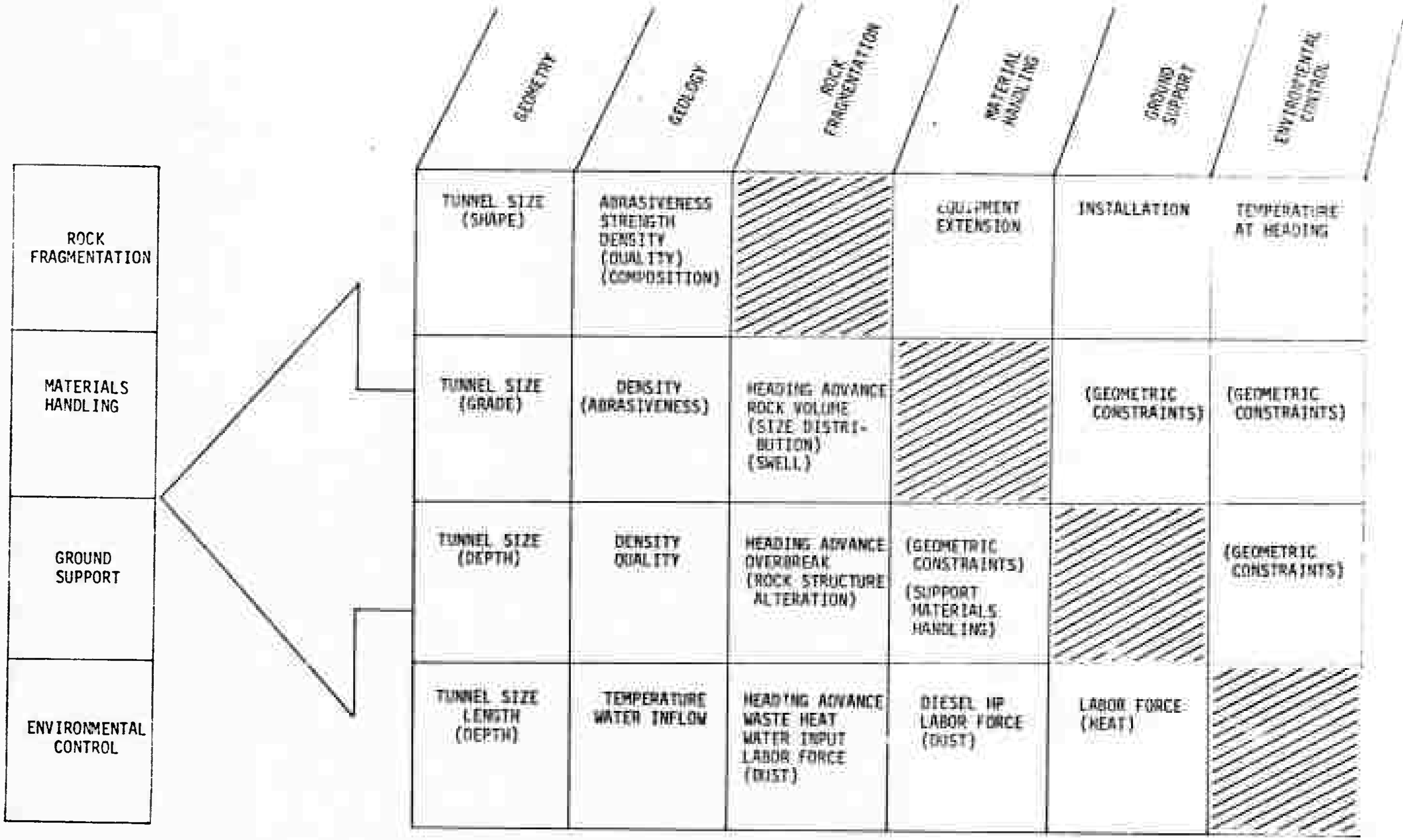

Figure 6. Element Interaction

to remove the rock). While the present model does not include consideration of the size distribution of rock fragments, this consideration should be kept in mind when ascertaining the suitability of a particular materials handling system to be used with a fragmentation system. For example, a hydraulic materials handling system may be suitable for transporting small-particle-size muck (less than 6-in. sizes) away from the tunnel face but it certainly could not bring the needed men and materials to the face as a conventional rail system could. It may be necessary in the event of using a materials handling system which is limited to handling a certain size distribution of material to augment this system with another for personnel and materials transport. Secondary crushing of the broken rock at the face may also be needed. The reader can no doubt identify other interactions which should be kept in mind.

The concept of the simulation proceeding from one well-defined time and state to another is implemented through the use of old and new working parameter common areas (Fig. 7). During any given time step, the old working parameter common area's contents describe the entire state of the tunneling system at the beginning of the time step; the new working parameter common area's contents describe the entire state of the tunneling system at the end of the current time step. Each subroutine receives its input information from the old common and places its output information in the new common. Each subroutine, therefore, determines the state of the system at the beginning of the new time step according to information received at the beginning of the current time step. In addition, as the subroutines are executed, cost information is generated. This information is placed in a cost information common area by the subroutines involved.

Note that the end result of this method of utilizing subroutines is that subroutines which are used for equivalent tunneling purposes, if called by the same FORTRAN name, may simply be exchanged for one another and the model will continue to operate and produce consistent operating

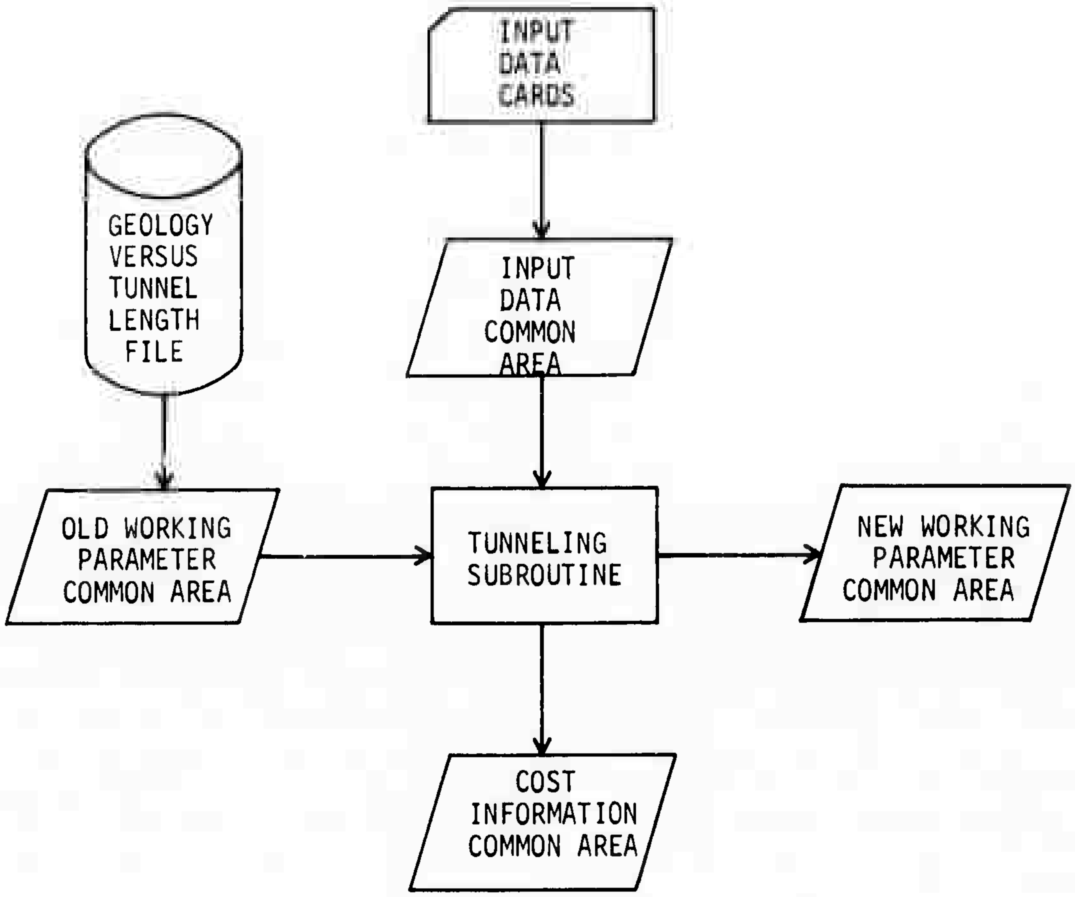

Figure 7. Information Flow for Individual Tunneling Model Subroutines

and cost reports as desired, even though the logic of the two subroutines, as well as their input information demands and output information results, might be radically different.

D. PERFORMANCE REPORTING

Figure 8 itemizes the information that will be provided periodically by the tunneling model during the simulation. In addition to the gross performance measures provided by advance, average advance rate, and maximum advance rate, two performance measures which reflect element reliability and compatibility will be printed out for each of the elements of the system. These performance measures are operational availability and subsystem (or element) utilization.

Operational availability may be defined as the probability that at a random point in time a subsystem can perform at or above some specified minimum level of operational capability to perform its primary function. It is a measure of the reliability of the subsystem. It may be expressed as

$$A = \frac{T_t - T_d}{T_t}$$

where A = operational availability

T_t = total shift time

T_d = total down time for planned maintenance, parts replacement and unscheduled repair time

Utilization is a measure of compatibility between elements of the system. It reflects inadequate matching of performance (e.g., a materials handling system inadequate to support boring machine advance rates) and interference between elements (e.g., ground support installation which

INTERIM PERFORMANCE REPORTS

DAYS SINCE COMMENCED EXCAVATING	59.121	
HEADING POSITION(FEET)	5092.339	
AVERAGE ADVANCE RATE(FEET/OPERATING HOUR)	11.833	
MAXIMUM DAILY ADVANCE		
DAYS SINCE COMMENCED EXCAVATING		32.000
HEADING POSITION-BEGINNING OF THAT DAY(FEET)		2553.534
HEADING POSITION-END OF THAT DAY(FEET)		2671.309
AVERAGE RATE FOR THAT DAY(FEET/OPERATING HOUR)		15.099

CUMULATIVE SUBSYSTEM PERFORMANCE..

ELEMENT/MEASURE	OPERATIONAL AVAILABILITY*	SUBSYSTEM UTILIZATION**
ROCK FRAGMENTATION	.376	.300
MATERIALS HANDLING	.955	.300
GROUND SUPPORT	.975	.447
ENVIRONMENTAL CONTROL	1.000	1.000

*OPERATIONAL AVAILABILITY=(TOTAL SHIFT TIME-DOWN TIME)/TOTAL SHIFT TIME

**SUBSYSTEM UTILIZATION=(TOTAL SHIFT TIME-DOWN TIME-IDLE TIME)/TOTAL SHIFT TIME

Figure 8. Interim Performance Report

requires a halt in boring operation). Utilization may be expressed as

$$U = \frac{T_t - T_d - T_i}{T_t}$$

where U = utilization

T_t = total shift time

T_d = total down time

T_i = total time during which an element is idle although available due to constraints imposed by external factors such as interference with other elements

In addition to this standard periodic performance report the user may select to print any additional information regarding the operation of the system. If he were interested in how the cutter changes for a boring machine affected its performance, for example, he may wish to print out cutter wear reports, cutter change records, and linear feet traveled by each cutter before replacement.

E. COST REPORTING

Cost reports, generated like periodic performance reports, reflect both overall cost and element cost by selected categories and are printed periodically. Figure 9 itemizes the information included in these reports.

The cost categories are described as follows:

a. Direct Labor

This is the cost of labor directly applicable to a specified activity, e.g., in the rock disintegration-boring machine option it will be the cost per hour of the crew required to operate the machine and any auxiliary activities included under this option. This figure is the

INTERIM COST REPORT

DAYS SINCE COMMENCED EXCAVATING	59.121
AVERAGE COST PER IN SITU CUBIC YARD EXCAVATED	31.92
AVERAGE COST PER FOOT OF TUNNEL	154.92

CUMULATIVE COST SUMMARY TO DATE..(DOLLARS)

ELEMENT/CATEGORY	DIRECT LABOR	PLANT + EQUIPMENT	JOB MATERIALS	PERMANENT MATERIALS	OVER HEAD	ELEMENT TOTAL
ROCK FRAGMENTATION	63666.04	191456.89	104006.16	0.00	35912.91	395042.00
MATERIALS HANDLING	138158.29	38928.80	7957.62	0.00	18504.47	203549.19
GROUND SUPPORT	99890.56	408.04	0.00	18304.21	11860.28	130463.09
ENVIRONMENTAL CONTROL	41332.56	5038.34	432.86	0.00	4680.38	51484.13
CATEGORY TOTAL	343047.45	235832.07	112396.64	18304.21	70958.04	780538.41

Figure 9. Interim Cost Report

product of manning requirements, prevailing wage rates, and elapsed shift time.

b. Plant and Equipment

This item represents the ownership cost of both capital equipment, i.e., depreciable equipment such as boring machines, conveyors, trucks, and fixed plant which is required (rail, power cables, etc.).

c. Job Materials

This is the cost of the consumable items used during a given activity. Examples of this would be the cost of power used, cutter costs, and explosive costs. The input to the program will be the unit costs, i.e., cost per kilowatt-hour for power. The individual activity subroutines will calculate from its internal functional relationship the cost of power consumed for a given advance in the specified tunnel. For the given element, the cost of all job materials will be accumulated and shown on the output records in a form shown.

d. Permanent Materials

This item represents the cost of materials used which form part of the permanent structure of the tunnel, i.e., the cost of rock bolts, steel ribs, or concrete.

e. Overhead and Profit

Overhead and profit expense is a fixed percentage charge to all the elements of the excavation process prorated by element subtotals to account for administration, supervisory personnel, unassigned labor, contingency, and profit.

The objective in providing this breakdown of total cost into a set of standard cost categories, divided among the excavation elements, is to facilitate a cost-benefit type analysis of new tunneling techniques. There has been a continuing conflict of opinions as to which aspects of tunneling are the most significant in determining the cost of any particular underground project. This is due in part to the lack of a basis upon which to compare the projected costs, actual costs, and potential savings in a systematic rational way. In terms of the actual costs incurred during conventional excavation of an underground project, it is expected that labor costs will continue to be the greatest single fraction, often running to 50-60% of the total project cost. This is due to the present reliance on excavation techniques having minimal emphasis on automation. There will be a trade-off, of course, between the cost of providing a reliable automated system and the cost of providing the laborers needed to do the same job. But clearly the trend will be toward more automation in the excavation systems which are developed for the future. One of the primary purposes for a systems study and model development is to make possible the identification of where the greatest savings may be achieved, and what it would take in terms of equipment and labor to realize these savings. The cost reporting structure shown in Fig. 9 provides one suitable basis of comparison.

F. APPLICATIONS

The simulation may be used to analyze on-going excavation projects (Fig. 10) or it may be used as a tool for system studies of excavation. Section VI of this report provides an example of the former. This section describes applications of the model to illustrate what kinds of system studies are facilitated by the simulation. A general methodology of systems analysis of excavation would consist of three steps:

1. Analysis of each process (current and novel) to identify performance characteristics, controlling parameters, significant trends, and inherent limitations

2. Analysis of element interaction to identify major incompatibilities, transfer of energy or material, and cost implications

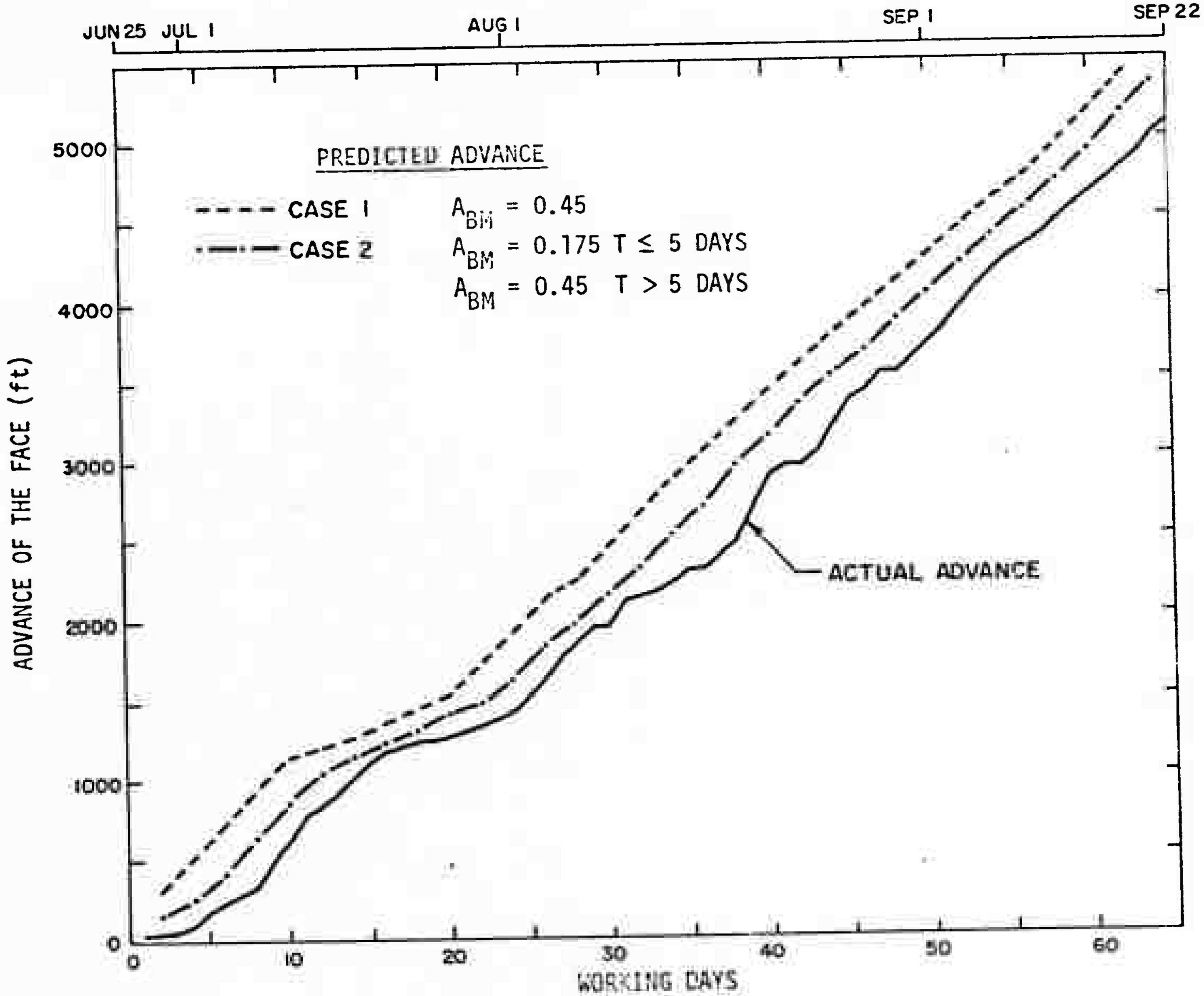

Figure 10. Comparison of Actual and Projected Heading Advance for the Layout Tunnel

3. Analysis of entire systems to identify optimum performance of current systems, major impact of novel systems, and areas to focus research and development to yield the greatest improvement in excavation cost and performance

In the comparative analysis phase of study it would be possible to perform parametric studies to derive curves which identify maximum rates of advance, cost, and other measures which reflect the estimated performance of selected (or hypothetical) excavation systems in various geologies. In any such analysis a list of the major independent parameters would be likely to include excavation system characteristics, and geology and geometry of the tunnel. Derived performance measures may include effectiveness (tons per hour excavated or the feet per hour advanced), cost (dollars per yard or dollars per foot), operational reliability to reflect element reliability; and subsystem utilization to quantify element compatibility and adequacy.

Presentation of results may be graphical, as shown generally by Fig. 11, or tabular as shown in Table 2.

TABLE 2

GENERALIZED TABULAR PRESENTATION OF RESULTS FOR PARAMETRIC ANALYSIS OF AN EXCAVATION SYSTEM

	Characteristic Performance	Effectiveness Measures	Cost Measures
System A	e.g., power, labor, material requirements	e.g., rate of advance	e.g., $/yd^3$
System B			
System C			

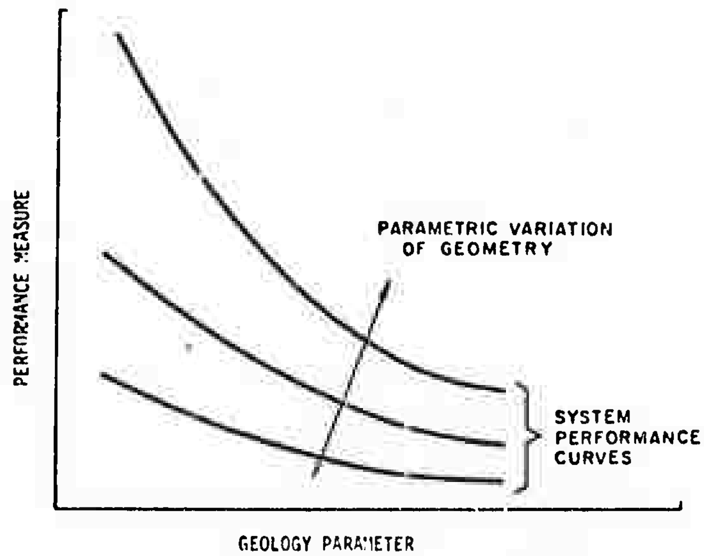

Figure 11. Generalized Graphical Presentation of Results for Parametric Analysis of an Excavation System

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Some simplified examples will now be given to illustrate the preceding applications. Figure 12 shows present boring machine capability in terms of hourly rate of advance for continuous operation for tunnel diameters between 6 and 20 ft. This curve is based on relationships derived in the course of this study to represent the actual performance of boring machines on past and current tunneling projects. Since this curve depicts rate of advance of a boring machine while it is operating continuously, the daily advance rate achievable would be the multiple of the daily operating hours (excluding down time, idle time, and cutter change time) and the hourly advance rate.

A second simplified example is illustrated by Fig. 13. The rock compressive strength is 20,000 psi. The estimated rate of advance of a boring machine can be calculated from the relationships derived in this study and is shown as the boring machine capability curve of Fig. 13 (solid line). Boring machines have apparently advanced faster in 16-20 ft

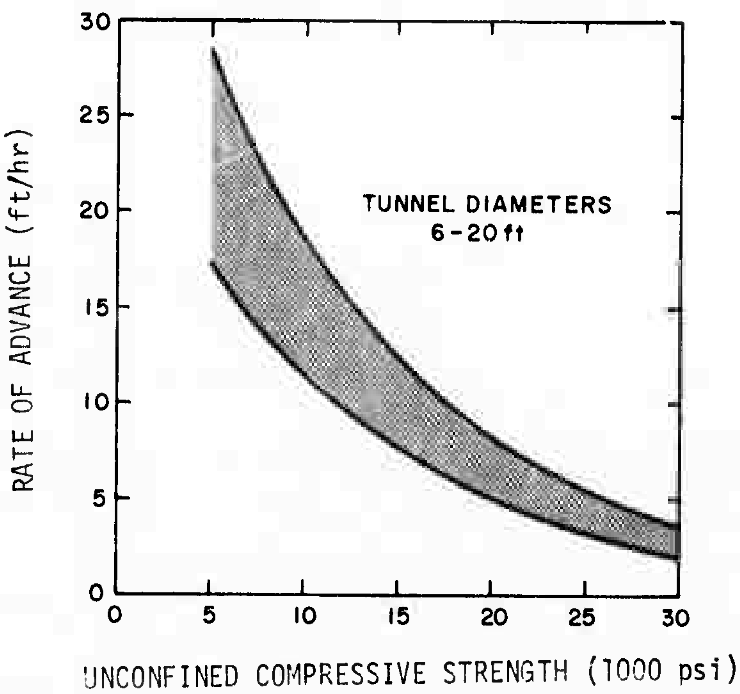

Figure 12. Boring Machine Capability Versus Rock Strength (Hourly Rate of Advance for Continuous Operation)

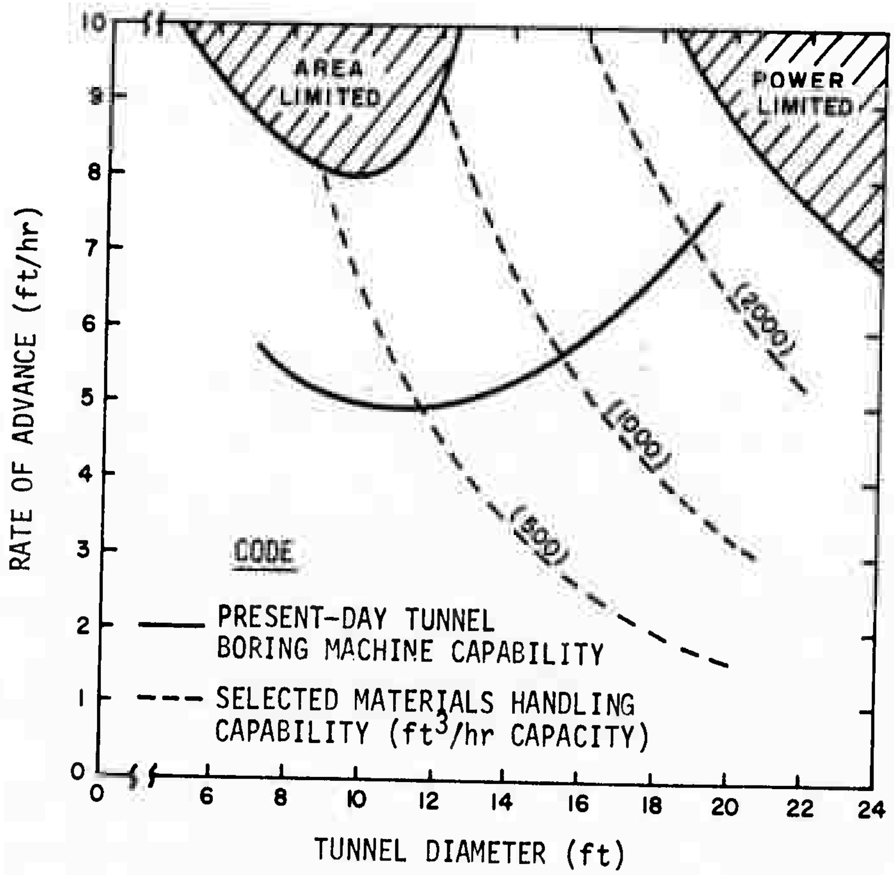

Figure 13. Parametric Analysis of Tunneling System (Simplified Example)

Rock Compressive Strength = 20,000 psi

diameter tunnels than in 8-14 ft tunnels (possibly due to machine limitations in the smaller tunnels or to some unidentified rock fracturing enhancement in larger tunnels) and this is shown in the shape of the curve. These faster rates of advance in the larger tunnels, however, have been achieved only by increasing exponentially the rated horsepower of a machine for a corresponding linear increase in diameter. A 20-ft-diameter machine, for example, requires 1800 hp. A conclusion may be drawn that there is probably a region of larger tunnels and higher advance rates that cannot be bored because of power limitations. This is shown as the "power-limited" region shaded in Fig. 13. Further study is needed to define the true boundary of this region. The region excluded in Fig. 13 is for a 2000-hp or greater power requirement.

Interaction between the boring machine and the materials handling system can be seen by considering the selected material handling capability curves (dashed lines) of Fig. 13. Each of these curves shows the rate of advance that could be supported by such a capability (no change in bulk density of the rock and constant bulk flow is assumed for simplicity). It can be seen that a 500 ft^3/hr materials handling system would be inadeuqate for a tunnel diameter greater than approximately 11.5 ft, 1000 ft^3/hr for 15.4 ft, and 2000 ft^3/hr for 18.6 ft, respectively.

One further limiting factor may also be identified. Investigation may show that the range of material handling capacities is limited by tunnel size. Large volume rate capacities perhaps cannot be achieved unless the tunnel size exceeds some necessary minimum area to accommodate the system. This would impose a tunnel area limitation on material handling capability and a corresponding region of Fig. 13 ("area limited") is not attainable because of this geometric constraint. Again, further study is needed to define the boundary of this region.*

Note that this simplified example presents only an approach to systems analysis. It does not specifically include consideration of many details (for example, boring machine, cutter life, materials handling system extension and logistics, ground support installation, environmental control) which may be important and can be studied using the computer simulation.

A third example, one illustrating comparative cost and performance analysis, is shown in Table 3. In this simplified example three different rock fragmentation processes are compared. The gross measures of performance, rate of advance in feet per hour, and job material unit cost in dollars per cubic yard are listed in the right-hand columns of this table. The particular tunnel being considered is 20 ft in diameter in 15,000-psi rock of 168 lb/ft^3 weight density.

An important feature to note in this table is the identification of different major cost items for the three different processes. For the boring machine, the major job material cost item is cutter wear; for the water jet it is power cost; for projectile impact it is the cost of projectiles. Additional investigation of other rock strengths would

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*The boundary of this region can be defined when a relationship between the maximum volume rate capacity of the materials handling system and the tunnel diameter can be established. For example, an assumption that the relationship is of the form

$$\dot{Q} = \sum_{n=1}^{N} C_n D^n$$

where $\dot{Q}$ = maximum volume rate capacity of material handling system
C_n = coefficients of the relationship
D = tunnel diameter

leads to a boundary in Fig. 13 defined by the equation

$$R = \frac{4\dot{Q}}{\pi D^2} = \frac{4}{\pi} \sum_{n=1}^{N} C_n D^{n-2} \qquad \text{where } R = \text{rate of advance}$$

TABLE 3

COMPARATIVE COSTS AND PERFORMANCE

ASSUMED TUNNEL DATA

DIAMETER = 20 ft

COMPR. STR. = 15,000 psi

WEIGHT = 168 lb/ft^3

ROCK FRAGMENTATION PROCESS	NUMBER OF UNITS	POWER REQUIRED, EACH UNIT, hp	COST OF POWER \$/hr	WASTE HEAT ADDED, 10^6 Btu	COST OF HEAT REMOVAL \$/hr	WATER ADDED ft^3/min	COST OF CUTTER WEAR \$/hr	COST OF PROJECTILES \$/hr	RATE OF ADVANCE ft/hr	JOB MATERIAL UNIT COST \$/yd^3
BORING MACHINE	1	1800	20	3	6	---	286	---	12.3	2.2
WATER JET	10	1430	160	34	68	100	---	---	1.0	18.9
PROJECTILE	2	1900	42	9	18	---	---	300	9.5	3.3

ASSUMED EQUIPMENT CHARACTERISTICS

BORING MACHINE

$\eta = 0.3$

WATER JET

d = 0.1575 in.

p = 10,000 psi

$\eta_t = 0.3$

$\eta_i = 0.2$

CONTINUOUS

PROJECTILE

vel = 5500 ft/sec

W = 4 lb

vol = 50 in.^{3}

cost = \$.25/projectile

$\eta_g = 0.3$

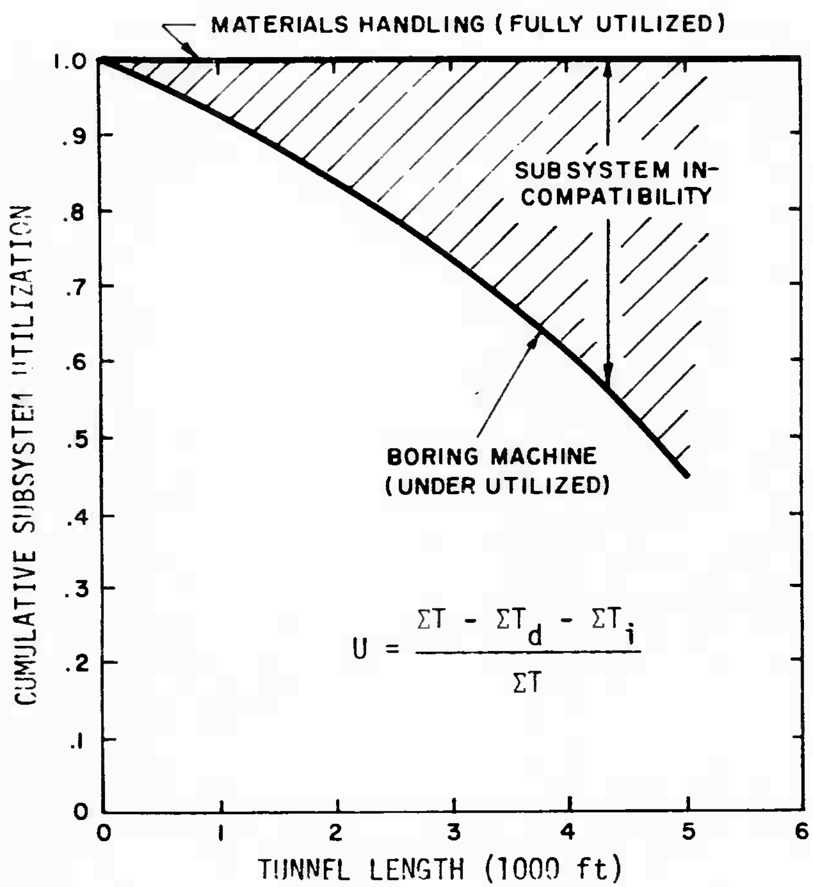
$\eta_i = 0.2$

rate = 10/min EACH GUN

identify trends in these costs; it would be expected, for example, that cutter wear cost would rise steeply as rock strength or abrasiveness increased, while water jet power or projectile cost may not increase as rapidly.

Again it should be stressed that this is a simplified example which illustrates the types of studies needed to show the major performance characteristics of different systems; it is not the result of any detailed study.

A final example of system analysis is shown in Fig. 14. This example illustrates the use of the utilization values that are calculated for each of the excavation elements to identify incompatibility between a boring machine and its materials handling system. Assume that the materials handling system capacity drops off as the tunnel heading advances. This could be due to increased cycle times, for example. If the materials handling system capacity matched the advance rate of the boring machine at the start of the tunnel, it would be less and less adequate as the boring machine advanced. Thus the boring machine would have to idle at times to allow the materials handling system to catch up, and this would be reflected in low utilization of the boring machine. The difference between the utilization of the materials handling system and the boring machine is a measure in this case of the incompatibility between them.

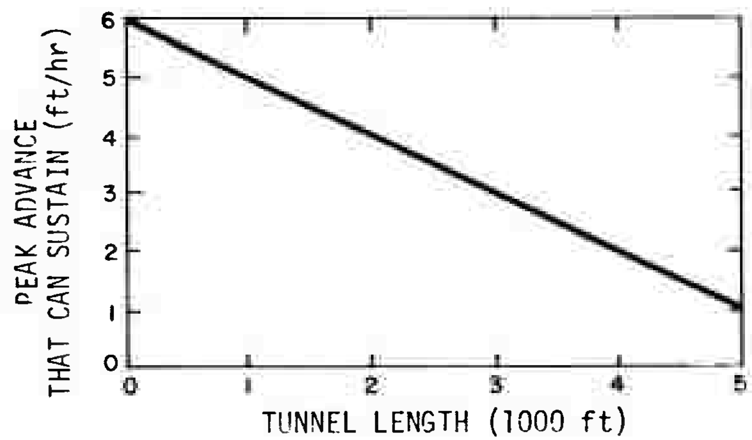


a) Subsystem Incompatibility

ASSUMED SYSTEM PERFORMANCE

(1) BORING MACHINE
PEAK ADVANCE RATE (6 ft/hr)
A = 100%

(2) MATERIALS HANDLING
A = 100%



b) Materials Handling Capability

Figure 14. Impact of Incompatible Element Performance (Simplified Example)

II. GEOLOGY

A. GEOLOGICAL SURVEYING AND PREDICTION

It is generally recognized that the geological (and hydrological) conditions more than any other factor determine the degree of difficulty and the cost of a given tunnel project. This is easy to see, since the tunneling system, support and liner design, and total system performance are a direct and strong function of the geologic medium to be tunneled through. In essence, the latter is truly a key variable in the total economic picture of a project. As a result, geologic exploration and prediction techniques have a very important influence on the planning, design, and performance of an excavation system. Although it is not yet possible to identify a return-on-investment relationship for geological exploration it is clear that a more accurate knowledge of geological conditions will permit considerable savings from the improved planning and design of a project.

At present, less than 2% of the total project cost is generally allocated to pre-excavation geological investigations.[8,9,10] This probably reflects the fact that the scope and extent of the geological survey is a compromise between technical desirability and economic feasibility. Moreover, the point of compromise may not be reached objectively in many instances. Budgetary considerations of sponsoring agencies, political considerations, etc., may also play a role in the decision process.

The results of a geological exploration program should consist of sufficient amounts of data concerning lithological, hydrological, and rock-mass properties to enable a designer and contractor to plan a construction project with confidence. This includes both the quantitative aspects of engineering and excavation system design, and scheduling plans and cost estimates. In other words, the contractor wants answers to the following key questions:

1. What would be the most suitable excavation method?

2. What are the ground support and tunnel liner requirements along the length of a proposed tunnel?

3. How much ground-water inflow can be expected along the tunnel length?

4. What is the location of potential geologic hazards?

The extent to which such questions can be answered with precision and reliability determines to a large extent the ultimate cost-performance success of the construction project.

The conclusion one draws from this and other more comprehensive discussions of geological conditions and their impact on excavation[11-19] is that non-homogeneous geology is a major consideration and any geology model intended to interact with an excavation simulation should be able to delineate geological discontinuities and inhomogeneities such as faults, joints, bedding planes, rock-soil interfaces, and ground-water concentrations.

B. GEOLOGY MODEL

1. Overview

The geology model is a part of the excavation simulation model which is illustrated schematically in Fig. 3. One of the basic requirements which was established for this simulation model by the Bureau of Mines was that it provide a realistic modeling of the geological conditions encountered during the process of tunneling; specifically, it was established that the rock which was encountered was not to be modeled as a homogeneous medium. It was also established that the excavation model was intended to be used as a research tool which would aid in evaluating the relative cost and performance characteristics of various techniques which might be used for tunneling through hard rock in the near future.

This latter consideration led to a decision to design the overall excavation model in such a way that the portion which was concerned with modeling geological conditions was a separate entity.

The geology model is intended to be a convenient tool for building a file of data which represents the geological conditions found within a given three-dimensional region of rock. The geology model is primarily intended for use in building files which represent hypothetical geologies. Reasonably accurate representations of existing geological regions may also be constructed, if desired. The sizes of the geological regions which are simulated, as well as the spacings of data points within the regions, are under the control of the user. A three-dimensional geology model was considered to be desirable because the eventual consideration of shafts, caverns and alternative tunnel routes would be aided by such a model. (A simple one-dimensional geology versus tunnel length program is also supplied as a utility tool if the user prefers. This will be mentioned again shortly in Sec. II-B-4).

Simulating inhomogeneous geologies in three dimensions seemed to be best approached by a deterministic model which would allow a user to include desired geologic features in any location or sequence he chose. Probabilistic variations of geology could subsequently be added if statistical studies provided the needed data. The geology model has, therefore, been developed as a deterministic model of appropriate three-dimensional geologic characteristics. The model is designed to simulate the geometry of strata having arbitrarily assigned quality, strength, abrasiveness, density, temperature, and water content.

Basically, we considered two methods of modeling the geology: serial and parallel. One can completely model the geology first, and then model the excavation process, or one can "make up" the geology as the excavation process takes place. The former approach was chosen. By modeling the geology first, we can completely separate the geology modeling logic from the excavation modeling logic; this simplifies both, and also simplifies the work involved in simulating alternative excavation methods. This approach also simplifies the work involved in evaluating the use of alternative excavation systems in the same geological conditions. In this case, the geology need only be simulated once. It can then be kept "on file" and be used repetitively by simulations of various excavation systems. The serial approach simplifies the work involved in simulating the actual geology in which one might be interested. It also simplifies the problems involved in ensuring that geological features are encountered in realistic sequences and contexts.

Using the serial approach, the geology is entirely determined before the excavation begins. This fact need not restrict the tunneling model. The tunneling model accesses the geology file to determine what the geology of a given location is, only when the excavation has proceeded to that point, and updates the previous knowledge of the geology at that time. From the point of view of the excavation simulation, the situation is exactly analogous to that found in actual practice; the geology is completely determined beforehand, but those who are excavating do not know for sure what the geology will be until they encounter it. The excavation simulation can therefore be made to respond to unexpected geologies in a realistic manner--involving alternative processes and techniques. time delays, and added costs.

The basic structure of the geology model is shown in Fig. 15. In the following paragraphs it will be convenient first to discuss the structure of the geology file which is generated; this gives some insight into the nature of the model. Then a discussion of the computer logic is given. Finally a description of how the geology model interfaces with the tunneling simulator is given. Further details documenting the inputs the methods of operation and the outputs of the model may be found in the accompanying appendices.[20]

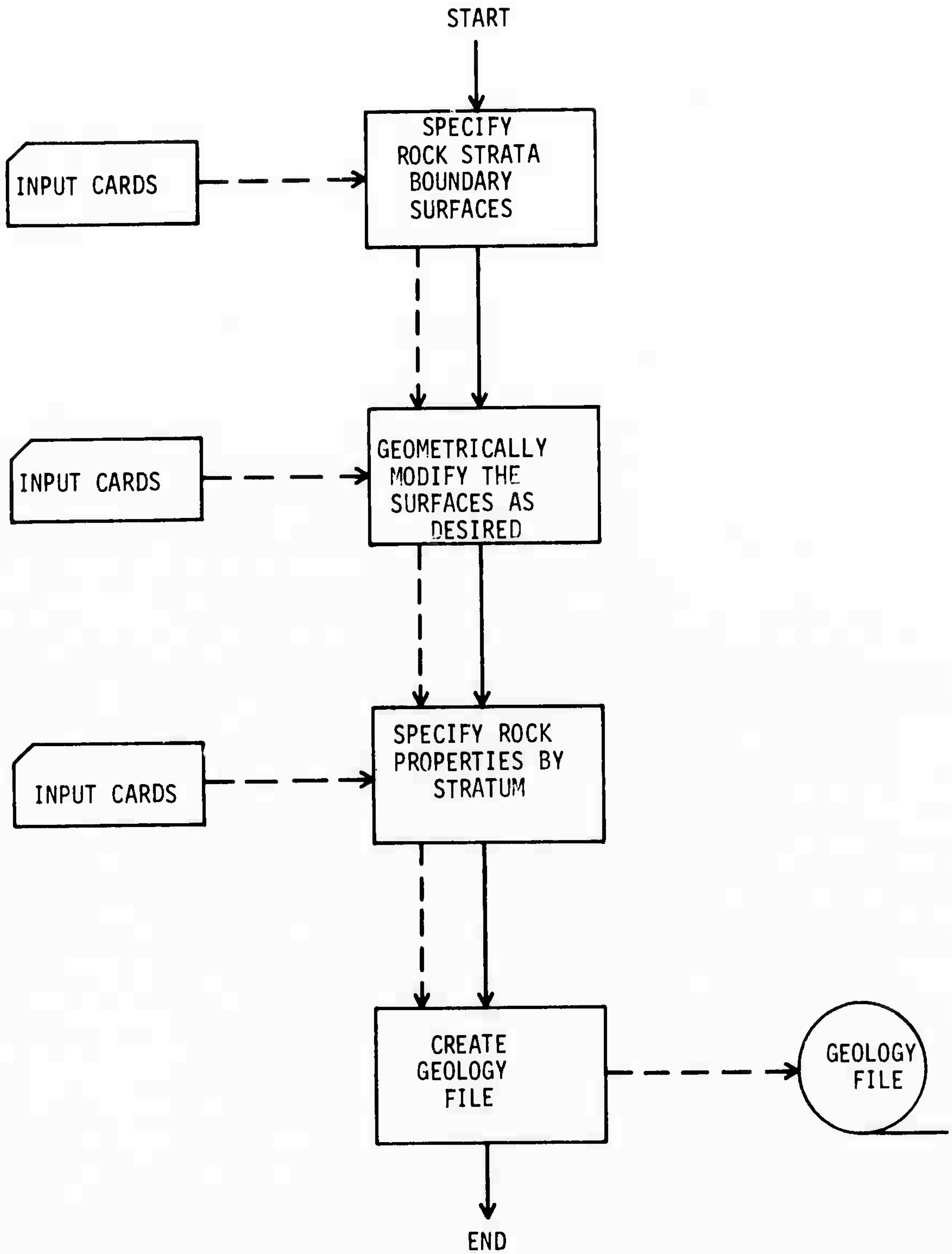


Figure 15. Three-Dimensional Geology Model

2. Output File Structure

The geology file produced by the geology model is a simple sequential (i.e., tape-like) file. This file consists of four parts, in order:

1. Dimensional Information
2. Rock Layer Information
3. Rock Property Information
4. Layer Position Information

Each of these four parts will be discussed in turn.

The dimensional information specifies the size of the region to be modeled and the horizontal spacing between data points. Implicitly, it also specifies the number of entries which will be found in the fourth part of the geology file--the layer position information. In order to see how this is done, imagine for a moment that you are well up in the air, looking down at a rectangular region of the earth's surface. This region is described, throughout the model, by means of the following directional notation conventions:

Y, NORTH, J, OR N DIRECTION

X, EAST, I, OR M DIRECTION

The compass-like notation is used for input convenience. The X and Y notation is used when it is convenient to refer to points within the region by means of floating-point coordinates; either the I and J or the M and N notation is used when it is convenient to refer to selected points within the region by means of integer coordinates. Vertical levels are always referred to by floating-point numbers; the variable name Z is used for this purpose. The positive vertical direction is upward.

Within the model, this region is divided up into an integral number of grid squares. There are NX squares from right to left and NY squares from top to bottom. NX need not equal NY. These grid squares need not actually be square. A scale factor can be associated with each of them. Thus, each square is SCALEX wide and SCALEY high. SCALEX need not equal SCALEY. The geology file contains geological layer position information for each point at which the grid lines cross.

Going back for a moment then, the first part of the geology file contains four entries of dimensional information which specify the size of the region to be modeled and the horizontal spacing of the data points. At the present time, the model can accommodate regions in which NX and NY are each less than or equal to 30. The values of SCALEX and SCALEY are not subject to any practical limitations.

The present version of the geology model allows the user to specify up to 25 different layers of rock. These layers of rock may be of up to 25 different kinds of rock.

The geology model allows the user to easily generate surfaces, which are interpreted as the upper boundaries of layers of specified kinds of rocks. The user is required to number these layers, for identification purposes, as he generates them. There is no requirement that the relative vertical positions of layers be in any way related to the identification numbers assigned to them. The basic coordinate system used is illustrated in Fig. 16. For the sake of clarity, only one surface has been drawn.

The second part of the geology file is simply a copy of the values which the user placed in the array named INDEX. The value of INDEX (ILAYR) is the number of the rock type which the user specified as being found directly beneath the surface which he gave the identification number ILAYR. Let us suppose that INDEX (ILAYR) = IROCK. Then, PROP (1 through 6, IROCK) contains the six parameters which the user specified as defining the kind of rock which he gave the rock identification number

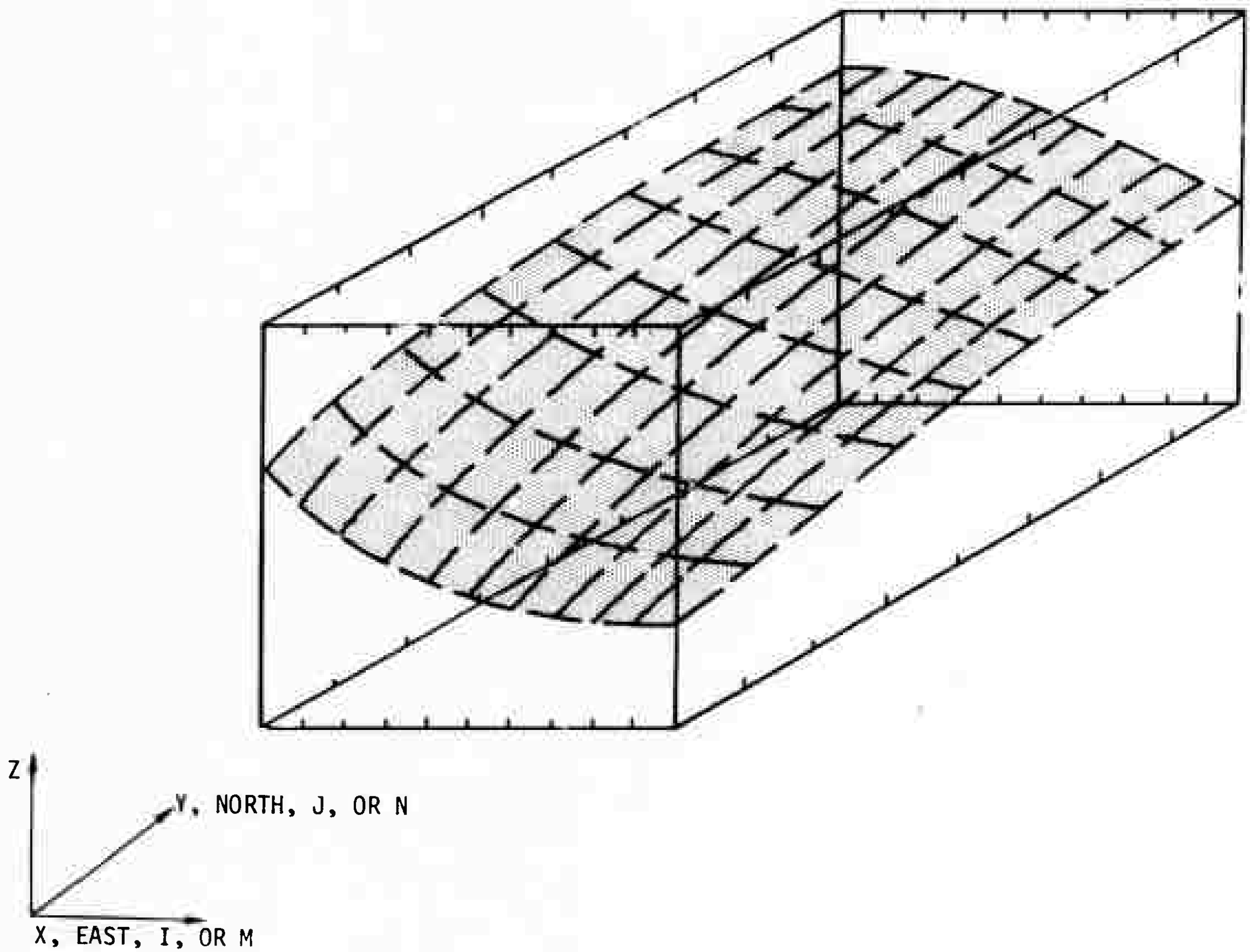


Figure 16. Representation of Geology

IROCK.* The third part of the geology file is simply a copy of the values which the user placed in the array named PROP.

The fourth part of the geology file consists of NX times NY entries. Each entry is for a single point at which the grid lines cross; the coordinates of that point are the integers (I, J). The Z_L for L = 1 to $ILAYR_m$ are the vertical positions of the top surfaces of the layers which the user has given the identification numbers 1 through $ILAYR_m$ respectively Z values for layers which he has not specified are set to the value $Z = -10^{51}$. In interpreting the entires at a given (I, J) location, it is understood that, in case of overlap, an arbitrary rule determines which layer numbers take precedence.

3. <u>Geology Model Logic</u>

There are two major phases of the geology model. During the first phase, the user is aided in manipulating surfaces on an NX by NY grid. These working surfaces may then be given layer number and rock type identification numbers, and be stored on auxiliary storage (usually disk storage) one surface at a time. The user is allowed to have up to seven working surfaces in existence at any one time. As was previously stated, the model as presently implemented will accommodate up to 25 layer surfaces.

Once the user has specified the surfaces which form the interface boundaries of the rock strata in the area of interest, he may geometrically alter or displace these surfaces by use of input data cards used to control the operation of the geology. The purpose of each of the different input data cards is explained in the appendices cited previously.[20] The main purpose of this operation is to allow the user

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*The properties presently considered are rock quality (RQD), compressive strength, abrasiveness, density, temperature, and water inflow. The geology model is equally capable of using any other six properties or combinations of these as desired.

greater flexibility in modeling geometric surfaces to represent geologic layers.

The result of the first phase of operation is a collection of individual files, each corresponding to the Z values of a single layer surface; each such file of Z values is in the customary FORTRAN ordering by I and J. What is desired is of course a single file, ordered by I and J, which contains the Z values for every surface at each value of I and J. This phase, which begins upon the receipt of a MAKEFILE card, is essentially a reordering phase. During this phase, the values of NX, NY, SCALEX and SCALEY, as well as the contents of the arrays named INDEX and PROP, are written out onto the new geology file. The rest of the file is constructed by stepping through all values of I and J and for each data point so defined, reading the corresponding Z value from each of the layer-surface files, consolidating these values into one entry, and writing this information onto the new geology file. When this process is completed, the operation of the geology model is terminated.

Figure 17 illustrates the two-phase nature of the geology model.

In summary, the operation of the geology model involves specifying rock strata surfaces, specifying rock properties by stratum, deforming the surfaces as required, and then reordering the information to produce a geology file which can be accessed by geographical coordinates. The object is to model realistically complex geologies in three dimensions, in a reasonable manner, without a great expenditure of computer time.

4. Geology Model-Tunneling Model Interface

The first step in the operation of the tunneling model is to read the specifications defining the coordinates of the tunnel to be excavated, and to use this information to access the complete geology file which was produced by the geology model, in order to produce a much smaller file of geological information along the length of the tunnel. This step is

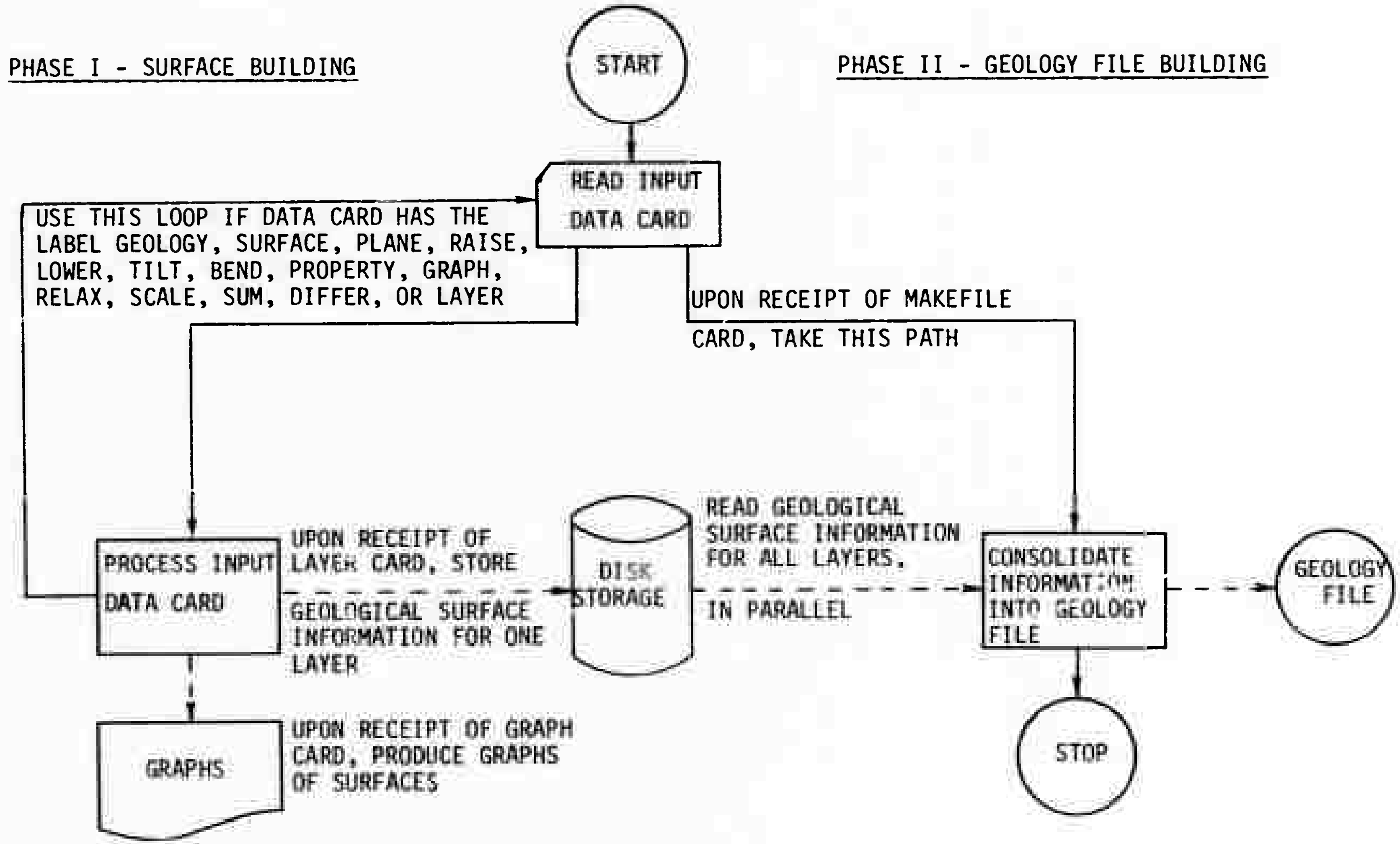
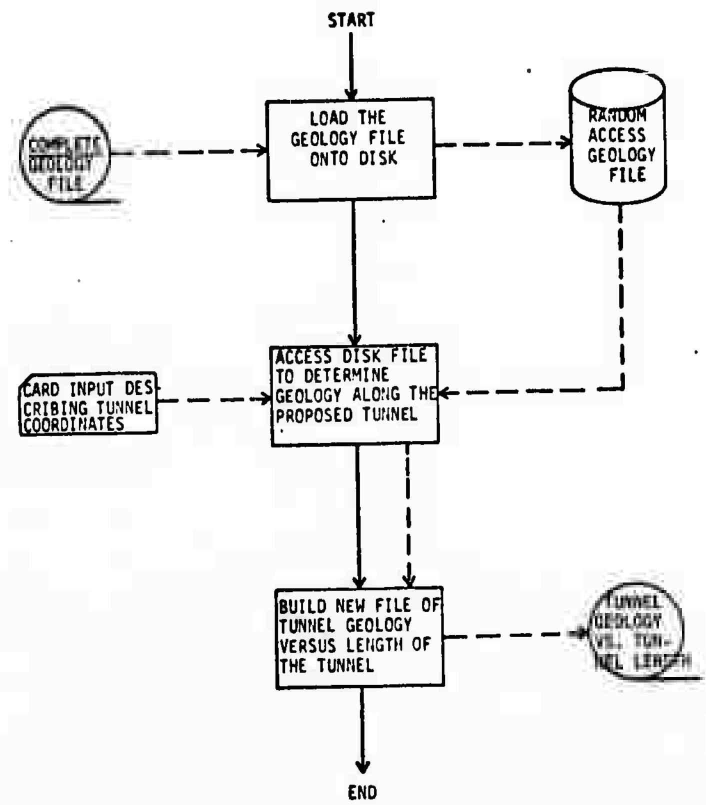


Figure 17. The Two Phases of Operation of the Geology Model

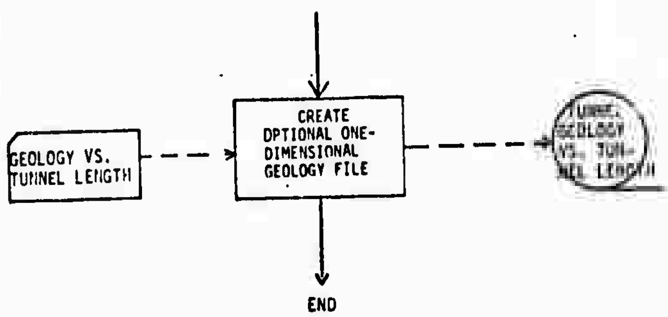
performed primarily as a matter of processing convenience; the resulting reduction in the volume of the geology file and the ordering of the data during this operation led to the simplification of the logic of the rest of the tunneling model. In addition, this smaller geology file provides a convenient starting point for the operation of the model during studies of the excavation of the same tunnel by alternative systems. In such studies, all processing up to this point need be performed only once. Figure 18a depicts the process of generating the file of tunnel geology versus tunnel length from the three-dimensional geology model.

If the user wishes to bypass the three-dimensional geology model entirely, a utility program is provided to create a one-dimensional geology versus tunnel length which includes the same six geologic properties as used above. In this case it is necessary to provide data cards which determine the location of change in geologic conditions along the length of the tunnel. This is illustrated in Fig. 18b.

The continued interaction between the resulting new geology file and the activities associated with the excavation process will become evident as we proceed into a discussion of the tunneling model itself, beginning with the first of the major elements of the process, rock fragmentation.



(a) Three-Dimensional Geology Conversion



(b) Utility Program for a One-Dimensional Geology Model

Figure 18. Creating a Tunnel Geology versus Tunnel Length File

III. ROCK FRAGMENTATION

A. INTRODUCTION

The function of the rock fragmentation element is to fracture rock into fragments suitable for removal by the materials handling system. Fracturing may be accomplished by mechanical forces, thermal stresses, impact, explosion, or any of a number of other techniques which can be used to break rock (Table 4).

TABLE 4
ROCK FRAGMENTATION TECHNIQUES

Rotary cutter	Flame jet
Drag cutter	Plasma
Spark	Electric arc
Explosive	Laser
Pellet	Electron beam
Projectile	Microwave
Impact hammer	Ultrosonic
Erosion	Radiant heat
Water jet	Nuclear

The drill-and-blast process of excavating is the standard and most often used process for hard rock. Although there are inherent cyclic delays (no rock can be loaded for removal during drilling and shooting, and all activity must be stopped after shooting to allow time for exhaust of explosive fumes) the high intermittent rate of breakage of rock is sufficient to counterbalance these delays, thus often making drill and blast the most rapid, economical, and sometimes the only practical, means of excavating hard rock today.

Improvements in the drill-and-blast process may be achieved by shorter cycle times, increased automation, and better operational planning. It may be unrealistic to expect great improvement, however, without substantially changing the cyclic nature of drill-and-blast excavation. Because conventional drill-and-blast excavation plays the major role in hard rock excavation today, it has been included in the excavation simulation to represent conventional excavation. It has been modeled in a manner which will also allow simulation of advanced drill-and-blast systems which are automated and have reduced cycle times.

The most promising approach to achieving rapid excavation of hard rock, the National Research Council concludes, lies along the path toward a more continuous method of rock fragmentation which eliminates wasteful cyclic delays. Placed at the top of its list of research priorities is development of new processes and equipment for boring tunnels and shafts in hard abrasive rock.[1]

A major thrust of research effort today is to improve the performance of tunneling machines and mechanical excavators so they will perform economically in hard rock. At the present time these recently developed "moles" and tunneling machines are most suitable for soft-to-medium-hard rock and fairly uniform geology. They do not yet attain economical performance in harder, more abrasive rock because their cutters wear too rapidly and do not break rock fast enough; the machines themselves may cause costly delays during fabrication and assembly, or through breakdown. Considerable research is now in progress, with some notable success, to eliminate equipment failure, reduce cutter wear, and improve rock breakage of tunneling machinery. Boring machine excavation is modeled to reflect the performance of the current rolling disc cutter type of machine such as that manufactured by the Robbins Company.

A second major thrust of research effort today is to identify new techniques to break rock without requiring physical contact between the rock and the machine, thereby eliminating cutter wear that periodically halts progress for replacement of cutters. The range of possible techniques to achieve this goal is wide; it includes impact techniques using projectiles or water jets, thermal techniques using electron beams, plasma arcs, lasers, and many others usually termed as a group the novel techniques of rock disintegration. Two of these novel techniques have been included in the simulation at this time: projectile impact and water jet (both pulses and continuous). Additional novel techniques may be modeled when sufficient field trial data is available to allow a valid estimate of performance to be made.

Before we proceed to a detailed description of the computer models of the various fragmentation techniques, we present a short discussion of rock failure and the derivation of an engineering parameter to measure the interaction between fragmentation device and the rock.

1. <u>Rock Failure</u>

Some of the failure modes of hard rock are:

1. Crushing in a region of high applied pressure which is greater than the compressive strength of the rock. This pressure may be applied by the thrust of a cutter rolling across the rock; it may result from the sudden expansion of exploding gases; or it may be induced by the energy transferred from a high-velocity jet of water or solid projectile striking the face.

2. Shearing of the rock from the forces induced by uneven comprehensive loading, the scraping of picks and non-rolling cutters or the erosive force of a fluid jetting across the rock surface at high speed.

3. Tensile stress failure caused by the interaction of stress waves produced in the rock by detonation or impact of a jet or projectile. This mode of failure is found especially in brittle rock.

4. Separation of faulted or kerfed fragments by stresses induced at terminal junctures of cracks, kerfs, and fissures in the rock. This mode of failure exploits preexisting weakness present in the rock or created by kerf cuts or previous jet or projectile impact.

Although it might intuitively be expected that the performance of any method of rock fragmentation would be strongly dependent on the mechanical properties (e.g., compressive strength) of the rock, no unifying relationships of this kind have been developed. This may be the result of an insufficient understanding of the different modes of failure of the rock. These may be related to rock properties not now being measured, or perhaps the rock properties which are being measured are not being correlated in proper combination.

An engineering approach to measuring the performance of any fragmentation technique is to assume a relationship between gross energy output of the device and the amount of rock that is fragmented as a result.

A common form for this relationship is:

$$E = \frac{Pt}{Q}$$

where E = gross energy required per unit volume of rock broken
P = power supplied by the fragmentation device
t = time interval in which fragmentation occurs
Q = volume of rock fragmented

The energy parameter E, as defined above is actually a measure of device-rock interaction. Frequently found in the literature referred to as "specific energy," it has occasionally been misconstrued as an inherent property of the rock or of the rock fragment size distribution. The value of the energy parameter measured for any rock-breaking process, however, will depend not only on the energy supplied, but also of the fraction that is absorbed by the rock, the modes of failure of the rock, and the size of rock fragments formed. In the following sections it will be made clear what basis for determining energy values has been used. The term "specific energy" is not used in this report because we wish to emphasize that the values of energy for two different techniques are comparable only if the bases for calculation and the fragment size distribution are similar.

TABLE 5

ROCK FRAGMENTATION PROCESSES MODELED

Drill and Blast
Boring Machine
Water Jet (pulsed or continuous)
Projectile Impact

Table 5 above lists those rock fragmentation processes which are presently included in the excavation model. Other promising fragmentation techniques are being investigated now for inclusion in a further development of the model.

Some concluding remarks concerning the interaction of rock fragmentation with the tunnel geometry, geology and other elements of excavation are appropriate at this time.

2. Tunnel Geometry

The size and shape of the desired tunnel places a major constraint on the type of excavation device that can be used. Most boring machines are currently designed to produce only circular tunnels and are generally limited to tunnels of 25-ft diameter or less (although the Mangla Dam project in Pakistan was successfully bored to a record diameter of 36.7 ft). Circular tunnels provide advantages in hydroelectric and water reclamation applications because their smooth uniform cross section reduces water flow resistance. For other applications such as transportation or mining, however, circular tunnels are usually not desired because a firm flat base is needed for mobility of equipment. Another limitation to the use of boring machines is due to the necessary large investment in the machine itself and the delay in mobilizing it at the tunneling site. Because of this investment in time and money, many shorter tunnels are not suited for economically boring by machine. Although there have been exceptions, most tunnels bored in the past have been a mile or more in length. The exceptions have generally involved reuse of a machine which had been paid for by an earlier project; as more tunneling machines become available in this manner it may be expected that a greater number of shorter tunnels will also be bored economically by machine.

The consideration of overbreak is important in deciding on an economical excavation technique. Overbreak is the amount beyond the design cross sectional area of the tunnel which must be excavated in order to achieve the satisfactory final design. In conventional drill and blast, where the surface produced by rock fragmentation is irregular and to some extent unpredictable, the overbreak required is substantial, frequently as much as 20%. This overbreak causes an increased load to be placed on the materials handling system because of the greater volume of rock that has to be removed. It also adds to the ground support and tunnel lining requirement for the tunnel sometimes increasing the concurrent costs considerably. Boring machines (and possibly many of the novel techniques considered) produce a more predictable, smoother tunnel surface, and 5% overbreak or less is commonly planned for.

Tunnel curvative and change in cross section are other aspects to consider. Tunnel boring machines, although their flexibility to respond to these geometric requirements is improving, are presently too large and cumbersome to cope with changes in alignment, grade, or diameter efficiently.

3. Geology

The impact of geology on drill-and-blast excavation lies in its relationship to drill pattern, amount of explosive needed, and depth of round suitable to maximize the tunnel advance without jeopardizing the stability of the tunnel or excessively overbreaking the tunnel walls. Compared to boring machine excavation, drill and blast responds more flexibly to changing geologic conditions and is not as limited by hard abrasive rock.

The strength, abrasiveness, and quality of the rock are the major factors affecting boring machine performance. High rock strength slows boring machine advance, high abrasiveness wears the cutters faster, and poor or blocky rock quality causes difficulties with the acquisition of the broken rock, sometimes jamming the cutters or muck buckets on the boring head.

The novel techniques such as water jet and projectile, which rely on impact for fragmentation of the rock, appear to be affected most by the rock density because it determines the energy coupling between the jet or projectile and the rock. The degradation of fragmentation rate with increasing rock density for these impact techniques is considerably less than would be the degradation of boring machine performance for the same harder rocks.

Thermal techniques still under investigation in the laboratory fracture rock in several ways which depend on the mineral characteristic of the rock. Spalling is enhanced with increasing quartz content of the rock. Thermal expansion of the rock in situ creates stresses which cause failure and are related to the application of the thermal energy, and the thernal properties of the rock.

4. Materials Handling

Any fragmentation technique affects materials handling because of the rock volume broken and the heading advance. The broken rock must be removed and the materials handling system must be advanced to keep up with excavation. Beyond these obvious interactions there are several of importance to consider generally. Size distribution of the rock fragments determines the suitability in many cases of the materials handling system. Rail car systems or truck haulage can be suitable for almost any size distribution of muck, while conveyors, hydraulic pipelines, or pneumatic transfer systems may require smaller fragments and a more uniform distribution of sizes.

The abrasiveness of the fragments affects the wear of the materials handling system. Erosion of hydraulic pipelines caused by abrasive rock particles remains a major problem with these systems.

5. Ground Support

The problem of overbreak has already been discussed as one of the impacts of the fragmentation technique on the ground support element of excavation. Two other impacts also come to mind. First, some excavation techniques minimize the damage done to the remaining rock, causing fewer fractures that extend into the walls of the tunnel, thereby preserving the integrity of the rock and reducing the ground support needed. Boring machine excavation clearly falls in this category as may some of the novel fragmentation techniques. Drill and blast, however, except for specially planned and monitored smooth wall blasting, does not. This may be of some greater consequence to tunnels designed for military applications where the response of the tunnel to dynamic loading caused by nuclear burst shock effects (e.g., block motions) may be more a consideration than for conventionally utilized tunnels.

A second interaction to consider between the excavator and ground support may occur during the tunnel excavation. An incompatible mating of rock fragmentation and ground support may lead, in the extreme, to a condition where neither may proceed while the other is in operation. For example, most ground support techniques used today were developed before the continuous excavator was developed and as a result are more suited to cyclic installation than continuous installation. When steel rib sets are used in conjunction with a boring machine and the rock quality is poor, requiring sets to be installed very near to the face, it is frequently necessary to halt the boring operation while sets are installed because of the interference between these subsystems. More continuous ground-support systems may be desirable in the future to match the performance of boring machines and other continuous excavation techniques. As novel techniques prove feasible for excavation, consideration should be given to what kind of ground support would be suitable used in conjunction with them.

6. Environmental Control

Dust, radiation, noise, water, thermal energy, toxic fumes, spewing rock fragments, fire hazard, and rotating machinery hazard are all environmental control considerations related to the fragmentation device.

The hazard of fumes associated with drill and blast is well known and planned for. The machinery hazards associated with machanical excavation can be minimized with proper design and safety procedures. The kinetic energy of rock fragments bursting from the tunnel face on impact of a high velocity water jet or projectile which may turn these fragments into hazardous misssiles should be considered. The noise of any high energy mechanical technique may require quieting countermeasures to be taken.

The thermal problem associated with high energy fragmentation techniques deserves special mention. In deep tunnels contemplated for military application the ambient rock temperature will probably exceed 100°F and require some form of cooling of the environment during the excavation process. Any additional load put on this cooling plant by thermal waste energy transferred into the tunnel environment as a by-product of the fragmentation process would add to the cost of cooling. This should be taken into account in the assessment of the suitability of any fragmentation technique for deep tunnel excavation.

Waste water released into a tunnel by a water jet may also pose particular problems. Recycling or removal would of course be necessary, but the problems of high humidity and low visibility due to moisture in the air would also have to be accounted for.

Table 6 summarizes the interaction of rock fragmentation with other aspects of excavation.

TABLE 6

SUMMARY OF FACTORS INTERACTING WITH ROCK FRAGMENTATION ELEMENT

<u>Geometry</u>

Size, shape, length, depth, grade, alignment, overbreak, tunnel surface desired

<u>Geology</u>

Strength, abrasiveness, quality, structure, water inflow, rock temperature, mineral characteristics

<u>Materials Handling</u>

Volume rate, heading advance, rock fragment size distribution, wear

<u>Ground Support</u>

Rock integrity, installation interference, overbreak

<u>Environmental Control</u>

Dust, radiation, noise, water, thermal energy, fumes, flying rock fragments, fire hazard, rotating machinery hazard

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B. DRILL AND BLAST

1. Introduction

The drill-and-blast process of rock fragmentation is the standard and most adaptable process for hard rock or mixed hard and broken rock. Blast holes are commonly drilled with a battery of hydraulically positioned drifter drills or manually operated air-leg drills mounted on a multileveled jumbo. The jumbo is a mobile steel structure which supports the drills and serves as a working platform for setting the charge, installing roof supports, and in some cases providing secondary ventilation at the face. Common drill patterns include the wedge, pyramid, and burn-cut. The rock is blasted with explosives and the blast fumes are cleared from the face area before the operations proceed.

The computer simulation of drill-and-blast fragmentation, while not intended to account for the great variety of blasting techniques currently utilized in the field, does provide a reasonable portrayal of the major events occurring in full-face blasting using a standard burn-cut pattern and is flexible enough to interact with different geological conditions and tunnel areas. It has been assumed throughout the modeling process that any tunnel modeled could be considered essentially equivalent to a round tunnel of some given diameter in order to allow comparison between drill-and-blast effectiveness and boring machine or novel process effectiveness.

Table 7, which has been kindly supplied by R. A. Dick of the Bureau of Mines, Twin Cities Research Center, provides a guide to the representative performance for drill-and-blast excavation. Optimum blast rounds vary from one project to the next. One reason is that rock properties and structure vary considerably from one tunnel to another and a pattern that achieves excellent results at one operation may be a failure at another. Another factor is the difficulty in drilling boreholes precisely. Even if the best pattern for a given heading were known, good blasting practice would dictate overdesigning the road to allow for drilling precise blastholes at a high rate of speed.

Generally, because ANFO is extremely cheap, adaptable to mechanized loading, gives ideal coupling, and seems to do as good a job in breaking rock as higher cost products, it is the preferred explosive unless conditions are wet.

It is generally felt that burn cuts with large-diameter relief holes offer the best opportunity for improving blasting efficiency underground. Even here, the primary restriction to the depth of advance is the accuracy of the boreholes.

TABLE 7

SELECTED DRILL-AND-BLAST PROJECTS

No.	Area of Tunnel ft²	Shape	Rock Description	Type of Cut	Large Diameter Cut Relief Holes No./Size	Size of Holes, in.	Depth of Round, ft.	Explosives/ft Advance	Holes per Round	Powder Factor lb/yd³	No. of Drills	Drilling Time, hr	Mucking Time, hr	Cycle Time, hr	Rounds per 24 hrs	Explosives
1	40		Basalt	B		1 1/2	6	18	34	12.2	2	2	2	5 1/2	6	Gelex 1
2	44		Basalt	B	3/3"	1 1/2	6	21	28	12.3	2	1 3/4	2	5	6+	Sp.Gel 60
3	44		Limestone	B		1 5/8	6	23	31	8.1	3	1 1/4	1 1/4	4	6+	Sp.Gel 40
4	53		Diabase	B	2/3"	1 3/4	6	17	26	7.8	2	1	1 1/2	3 1/4	7	Gelex 2
5	55		Gneiss	B		1 1/2	6.5	16	36	8.2	3	1 1/2	1 1/4	7	7	Tro. TL5532
6	70		Limestone	B		1 7/8	10	37	49	4.6	3					
7	81		Vol. Ash	B		1 3/4	10	21	36	4.7	3	1/2	1 1/2	4	7	Gelex 2
8	88		Limestone	V		1 5/8	6	11	26	3.3	2	2	1 1/4	6	5	Sp. Gel 40
9	90		Basalt	B	2/5"	1 5/8	8	12	30	3.8	5	3/4	1	2	9	Gelex 2
10	90		Granite	B	2/5"	1 5/8	11	22	34	5.8	5	1 1/2	1 1/4	4	6	Gelex 2
11	90		Limestone	Py		1 5/8	7	15	33	4.3	2	1 3/4	1	4 1/2	5	Sp. Gel 40
12	98		Limestone	B		1 5/8	6.8	28	40	5.3	3	2 1/2	2 1/2	6 1/2	3 1/2	Sp. Gel 60
13	113		Granite, Gneiss	Py		1 5/8	8	25	43	7	4	1 1/2	1 1/4	3 1/2	6	Gelex 2
14	113		Slate	B		1 1/4	6	7.6	41	1.8	4	1 1/2	1	3 3/4	6	Sp. Gel 40
15	120		Limestone	B		1 5/8	10	24	44	5	3	3	3 1/2	7 1/2	3	Atl. Gelodyn
16	122		Granite	V		1 3/4	6.8	31	57	4.2	6	1 1/8	2	4	5	Atl. Gelodyn
17	128		Shale	B		1 3/8	8	14	33	2.5	3	2	2	5 1/2	4+	Gelex 2
18	130		Vol. Ash	V		1 5/8	11	24	56	4.3	4	1/2	1 1/2	6	6+	Herc Gelamite
19	154		Basalt	V		1 3/4	6	7.9	37	2.1	2	2	2	7 1/2	2+	Atl. Gelodyn
20	182		Granite	B	2/5"	1 3/4	11	30	55	6.2	7			3	8	Herc ANFO
21	190		Shale	B		1 1/2	8	17	44	2.4	2	4	8	14 1/4	2	
22	[illegible]		Granite	B	2/5"	1 3/4	12	55	49	6.7	9	1	1 1/2	3	8	Herc ANFO
23	[illegible]		Basalt	Py		1 3/4	5	17	45	2.9	4	1 1/2	1	3	7	Gelex 2
24	234		Limestone	V		1 3/4	10	33	46	6	2	4	2	7 1/4	2+	Gelex 2
25	253		Shale & Vol.	V		1 5/8	2	25	32	2.8	4	1/2	1/2	2	12	Herc HF 50
26	281		Limestone	B	1/10"	1 5/8	11.5	52	65	5	7	2 1/2	2	6	4	Herc Gelamite
27	298		Limestone	B	1/10"	1 5/8	11.5	44	60	8.8	5	1 1/4	1 1/2	4	5	Atl. Gelodyn
28	294		Schist	V		1 5/8	12	55	85	5.1	7	2	2 1/2	6	3 1/2	Gelex 1
29	333		Granite	V		1 5/8	11	38	95	2.7	10	1 1/2	2 1/2	5 1/4	3	Gelex 2
30	331		Granite	Slab	Pilot	1 5/8	12	23	45	1.8	4	3	1 1/2	5 2/3	4 1/2	Gelex 2
31	400		Shale	V		1 7/8	6	66	60	3	6	2	1	8	3	Atl. Amodyn
32	452		Basalt	V		1 3/4	10	41	71	2.4	8	2 1/2	2 1/2	6	4	
33	496		Limestone & Sh.	V		1 7/8	6	50	84	3	9	1	3/4	3 3/4	6	Gelex 2
34	365		Shale	Py		1 3/4	8	50	75	2.4	8	1	2	2	3	Gelex 1
35	566		Granite	V		1 3/4	10	80	150	3.8	16	1 3/4	2 1/4	4 1/2	5	Atl. R 2/9
36	729		Granite & Schist	V		1 5/8	7	57	122	2	8	3	3	7	3	HiCap1

The activities that are modeled by the simulation, along with their subroutine names, are given in Table 8. The portrayal of these activities by mathematical representations will be considered in sequence in the following paragraphs.

Table 9 provides a list of the parameters and input specifications which drive the simulation.

TABLE 8
DRILL-AND-BLAST ACTIVITIES

- Subroutine MOVEIN
  Moves drill jumbo to face
- Subroutine HOLBRN
  Drills holes for burn-cut pattern
- Subroutine SETCHG
  Sets charge
- Subroutine MOVOUT
  Moves drill jumbo away from face
- Subroutine DREPAR
  Maintenance and repair

2. Subroutine MOVEIN

This subroutine accounts for the time delay in repositioning the drill jumbo at the tunnel face at the start of a new cycle. The user may specify the anticipated time required for this repositioning and alignment check. If no time value is specified, the program assumes a 15-min delay time, which agrees with common practice.[21,22]

TABLE 9

DRILL-AND-BLAST PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

EQUIPMENT INFORMATION

(1) Drill Jumbo

(a) Time to move to face (hr)* ____________________

(b) Time to move away from face (hr)* ____________

(2) Drifter Drill

(a) Number of drills* ____________________

(b) Area of holes (in^2) ____________________

(c) Power output (hp) ____________________

(3) Burn-Cut Drill

(a) Number of drills* ____________________

(b) Area of holes (in^2) ____________________

(c) Power output (hp) ____________________

PATTERN INFORMATION

(1) Number of drifter holes* ____________________

(2) Number of burn-cut holes* ____________________

(3) Depth of holes per round (ft)* ________________

(4) Number of holes charged simultaneously* __________

EXPLOSIVES INFORMATION

(1) Powder factor (lb/yd^3) ____________________

(2) Amount of primer (lb/hole) ____________________

(3) Time to set one charge (hr) ____________________

(4) In lieu of items 1-3 (select one)

(a) Dynamite option

(b) ANFO option

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*Default values are programmed into the simulation for these parameters. The user may override these internal values by specifying an alternative value on the appropriate data card.

TABLE 9 (cont.)
DRILL-AND-BLAST PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

COST INFORMATION

1. **Plant & Equipment**

Item	Ownership (Rental) Cost Per	Hr (check one)	Ft	Unusual Development Cost Per	Hr (check one)	Ft
Major Items:						
Drill Jumbo						
Burn Cut Drill						
Drifter Drill						
Drill Positioner						
Jib						
Additional Items:						

2. **Job Materials**

Item	Cost Value	Cost Unit	Lifetime Value	Lifetime Unit
Major Items:				
Drifter Bit*		$		ft
Burn Bit*		$		ft
Steel*		$		ft
Primer*		$/lb	—	—
Explosives*		$/lb	—	—
Firing Cap*		$/cap	—	—
Miscellaneous*		$/hole	—	—
Additional Items:				

3. **Direct Labor**

Labor Type	Number Required/Shift	Rate $/hr
Major Types:		
Shifter		
Miner		
Nipper		
Chucktender		
Powdermen		
Maintenance		
- Foreman		
- Mechanic		
- Electrician		
Additional Types:		

4. **Permanent Materials**

Item	Cost Value	Cost Unit
(None required)		

3. Subroutine HOLBRN

This subroutine models the drilling of the holes into which the explosives are placed, and the drilling of the relief holes which are utilized in a burn-cut pattern.

Although there is a wide range of drill types and methods of mounting drills, most drilling in hard rock tunneling is done with a percussion drill having either rifle-bar rotation or some separate positive method of drill rotation.

Sinkers and jackhammers, designed to be hand held, vary from a light (30-lb) drill to a heavy (70-lb) drill. Feed legs and jacklegs, which are sinker drills mounted on an air-feed leg, are used generally for both lateral and overhead drilling. Drifters are self-rotating drills which are screw or chain fed. Burn-hole drills are drifters used to drill the large holes on a burn-cut pattern of shooting.

Drifter drills are suspended from jibs mounted on drill jumbos which serve as working platforms and house all facilities required for drilling a round: pumps, air and water connections, lights, and ventilation. The jumbo may also be used for loading the holes, placing supports, and in some cases handling muck cars.[23]

If the parameter E is defined as the gross drill energy expended per rock volume removed (which includes an energy transfer coefficient, generally 0.6 to 0.7, for the loss in energy that occurs between the drill and the drill bit), then drilling rate in feet per hour, R_D, can be expressed as

$$R_D = 1.98 \times 10^6 \, P/A_h E$$

where P = power output of the drill (hp)

A_h = hole cross-section area (in^2)

E = drill energy expended per rock volume removed $[(in.-lb/in^3) \times 10^3]$

Some representative data for energy per volume relationship of percussive drills in hard rock is shown graphically in Fig. 19. Also shown in this figure is the relationship incorporated in the computer subroutine which is derived from a least squares fit of the data for rock strengths below 50,000 psi. For rock strength above 50,000 psi the observed data are inconclusive but suggest an energy per volume range of approximately 50 to 75 $[(in.-lb/in^3) \times 10^3]$.

The relationships included in the computer model are:

$$E = 15\, e^{0.031\sigma}$$

where E = energy per rock volume removed, $[(in.-lb/in^3) \times 10^3]$
σ = rock compressive strength, 10^3 psi between 5000 and 50,000 psi

and

$$E = 60\ [(in.-lb/in^3) \times 10^3]$$

for $\sigma > 50{,}000$ psi.

The power output of a drill may be calculated to be the number of piston blows per minute times the energy in each blow. The following formulas have been shown by Hustrulid[26] to have general applicability to percussive drills in hard rock:

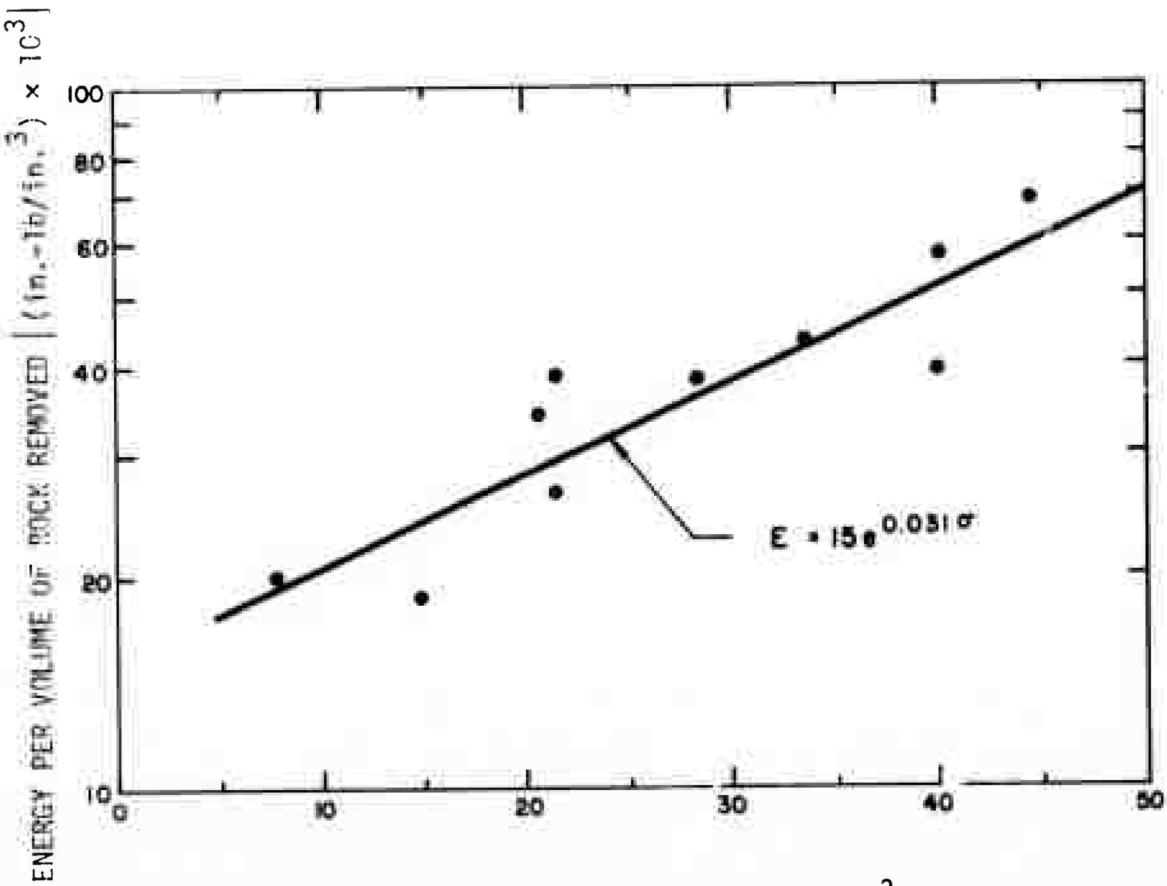

Figure 19. Percussive Drill Performance[24,25]

NOTE: To convert in.-lb/in^3 × 10^3 to J/cm^3, multiply by 6.9. This energy parameter is commonly found in published literature in units of J/cm^3.

$$P = fE_p$$

$$f = 22\sqrt{\frac{6pAg}{Sw}}$$

$$V_S = 0.68\sqrt{\frac{SpAg}{6w}}$$

$$E_p = \frac{1}{2}\frac{W}{g}V_S^2$$

where A = area of piston head (in^2)

E_p = piston energy (ft-lb/blow)

f = blow frequency (blows/min)

g = acceleration of gravity (fps^2)

P = power output (ft-lb/min)

p = applied air pressure (psi)

S = piston stroke (in.)

V_S = piston striking velocity (fps)

w = weight of piston assembly (lb)

It is intended that a required input to the computer subroutine which calculates drill performance will be the power output of the drill. This can be estimated by the user of the model from the above formulas and information provided by drill manufacturers.

A simple mathematical representation which approximates the number of shotholes per round which need to be drilled is:

$$N_h = (D + 0.1\ D^2)\psi$$

where N_h = number of drifter holes required

D = tunnel diameter (ft)

ψ = factor to account for rock quality, strength, tunnel geometry, diameter of shotholes, and diameter of burnholes

At the present time in the model, ψ is set equal to unity, and this gives an adequate representation for average rock conditions (Fig. 20). In massive, intact hard rock a more representative value of ψ may be 1.1 to 1.5. For large tunnels a substantial reduction in the number of holes required can be realized if the shothole diameter is increased. For minor areas the same saving is not evident and rarely justifies the employment of bigger drills.[27]

The number of drifter drills employed is assumed, if not otherwise specified, to be one-eighth the number of holes drilled each round so that each drill drills eight holes.[23] The user may specify an alternative number of drills if he chooses.

The time to drill the shotholes is calculated as

$$\Delta t_D = \frac{\ell_D}{R_D}\left(\frac{N_h}{N_D}\right)$$

where Δt_D = drill time (hr)

ℓ_D = depth of the holes (estimated to be 1 ft greater than the advance per round)

N_h = number of shotholes

N_D = number of drills

R_D = rate of drill penetration (ft/hr)

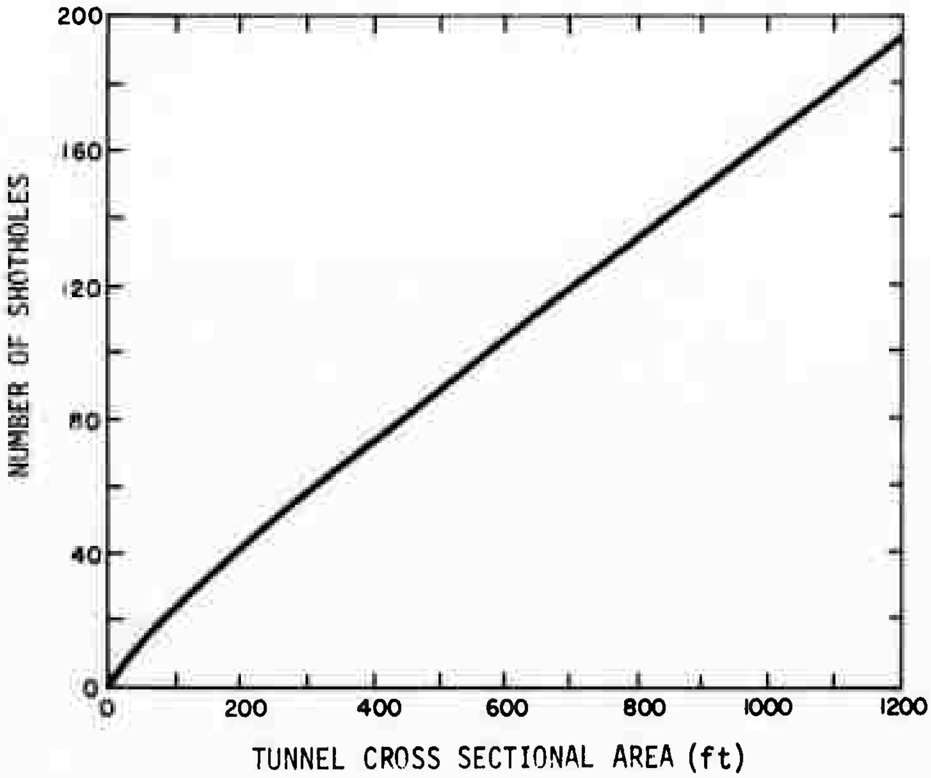

Figure 20. Shotholes Required for Average Rock Conditions (ψ = 1)

To estimate the length of the rounds, it is first necessary to determine the spacing of the supports required from the geological information available. This is discussed in Sec. V concerning ground support. In a supported tunnel, the round length is limited to the distance between the supports (or the allowable unsupported tunnel length if some form of continuous support is installed). In an unsupported tunnel, the length of a round is determined by the type of cut used and the diameter of the tunnel. The burn-cut pattern modeled by the simulation could theoretically be of any length round. In practice, production has seldom increased if rounds over 10 ft long have been used.[23] The model therefore is based on the assumption that the length of a round will be either that allowable by ground support constraints or 10 ft, whichever is appropriate. If a V or diamond cut were to be modeled, the round length would be limited to not more than two-thirds of the diameter of the tunnel. If longer rounds were to be attempted, a large amount of drilled but unbroken rock would be left at the end of the tunnel.

The time to drill the burn-cut holes is similarly calculated as

$$\Delta t_{Db} = \frac{\ell_{Db}}{R_{Db}} \left(\frac{N_{hb}}{N_{db}} \right)$$

where Δt_{Db} = drill time (hr)

ℓ_{Db} = depth of holes (estimated to be one foot greater than the advance per round)

N_{hb} = number of burn holes

= 2 for tunnel diameter $\leq$ 15 ft

= 3 for tunnel diameter > 15 ft

N_{db} = number of burn-cut drills

R_{Db} = rate of drill penetration (ft/hr)

In both instances the user supplies as input to the program the cross-sectional area of holes to be drilled and the estimated power output of the drill.

The job material costs associated with this hole drilling activity are drill bits and steel costs. If the program user supplies the necessary information, the following formula is used to calculate these job material costs for the drifter drills.

$$\text{Job material cost (\$)/round} = \frac{C_B}{L_B}(n_D \ell_D) + \frac{C_S}{L_S}(n_D \ell_D)$$

where C_B = drifter bit cost ($)

L_B = average bit life (ft)

n_D = number of drifter drills

ℓ_D = average depth of holes (ft)

C_S = cost of steel for one drill ($)

L_S = life of steel (ft)

Drill bits will typically have a life of 200 linear feet in granite. A standard bit costs approximately $20. These values are programmed as optional internal values if the user does not specify alternative ones.

Drill-bit life is longer in less abrasive rock, and this would reduce bit and steel costs proportionally. The expected life in other rock has not been identified in this study. The relationship for granite, however, may be used as a conservative formula for drill bit costs.

Compared to mechanical excavation by tunnel boring machine, drill and blast is a more labor-intensive process. A representative set of labor crews for different diameter tunnels is provided as an example in Table 10a. The crew size for any given project, however, will vary considerably depending on labor availability, customary contractor practice, labor regulations, and other factors. As can be seen, the number of men to be used for drilling will increase with the rock face area. These data are subject to interpretation since electricians and mechanics will have duties other than maintaining the drills. For example, they may also be concerned with the maintenance of the equipment for materials handling.

The amount of plant and equipment used will also vary with the tunnel diameter (Table 10b). The variable quantities are the number of drifter drills, jibs, and positioners. The number increases with tunnel diameter in such a way that the time for drilling in cycle time will remain approximately the same with varying diameter for a given compressive strength of rock.

4. Subroutine SETCHG

This subroutine calculates the powder requirement for blasting, and accounts for the associated time and cost of the activity of setting the charge. The user has the option of specifying either ANFO or dynamite as the type of explosive and letting the program compute the powder factor.

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*Powder factor is a common measure of powder requirements; it is the number of pounds of powder that is required per cubic yard of rock broken.

TABLE 10

REPRESENTATIVE DRILL-AND-BLAST COSTS

(a) Manpower per Shift

Skill	Tunnel Diameter 12'	15'	25'	Washington, D. C., Area Hourly Costs*
Foreman	1	1	1	$8.32
Miners	3	4	10	7.20
Mechanic	1	1	2	8.10
Electrician	1	1	1	8.92

*Includes 25% fringe benefits, FICA, etc.

(b) Plant and Equipment

Plant and Equipment Used	Number Required for Tunnel Diameter 12'	15'	25'	Cost/Unit $
Jumbo	1	1	1	30,000
Burn-Cut Drill	1	1	1	11,500
Drifter Drill	3	4	10	5,800
Jib	3	4	10	5,200
Drill Positioner	3	4	10	2,800

In the case of a user-specified powder factor, the weight of powder per round is

$$W_{EX} = \frac{P.F.}{27} A_t \Delta X$$

where W_{EX} = powder per round (lb)

$P.F.$ = powder factor (lb/yd^3)

A_t = tunnel cross-sectional area (ft^2)

ΔX = estimated advance per round (ft)

If the powder factor has not been specified, it is approximated by

$$P.F. = \left(\frac{27}{144}\right) \frac{A_h \rho_e}{A_t} n_h$$

where $P.F.$ = powder factor (lb/yd^3)

A_h = area of shothole (in^2)

A_t = tunnel cross-sectional area (ft^2)

n_h = number of shotholes

ρ_e = weight density of the explosive (lb/ft^3)

= 50 lb/ft^3 for ANFO

= 70 lb/ft^3 for dynamite (Gelex #2)

The above equation assumes that each shothole is fully charged with explosive.

The time required to charge the holes and set the charge is assumed to be

$$\Delta t_s = \tau \frac{n_h}{n}$$

where Δt_s = time to charge and set charge per round (hr)

τ = time to charge and set one hole (hr)

= 4/60 if not otherwise specified

n_h = number of shotholes

n = number of holes charged simultaneously, either manually or automatically

= $n_h/8$ if not otherwise specified

The job material costs expended by this activity may be calculated as

$$\text{Job material cost (\$)} = n_h(W_p C_p + C_{cap}) + C_m \, \Delta X + W_{EX}(C_{EX})$$

where n_h = number of shotholes per round

W_p = amount of primer (lb/hole)

= 0.5 lb/hole ANFO

= 0 dynamite

C_p = cost of primer (\$/lb)

= 0.2 ANFO

= 0 dynamite

C_{cap} = cost per cap (\$/cap)

= 0.3 ANFO or dynamite

C_m = miscellaneous cost per hole for stemming, wire, etc. (\$)

= \$1 per foot of advance

ΔX = estimated advance per round

W_{EX} = powder per round (lb)

C_{EX} = powder cost ($/lb)

= 0.66 ANFO

= 0.3 dynamite (Gelex #2)

5. Subroutine MOVOUT

This utility subroutine accounts for the time delay in moving the drill jumbo back from the face following the hole drilling and setting of the charges, plus the blasting of the round. The user may specify the anticipated time required for this activity. This should include the estimated smoke time to remove blast fumes from the tunnel following blasting.

6. Subroutine DREPAR

Subroutine DREPAR accounts for drill-and-blast maintenance and repair periods. In terms of performance parameters it accounts for either the drill-and-blast system availability or the down time per maintenance period and the average time between maintenance periods. The effect of breakdowns may also be accounted for by the operational availability factor assigned to the drill-and-blast system. During drill-and-blast maintenance and repair, none of the other activities (MOVEIN, HOLBRN, SETCHG, or MOVOUT) proceed.

Costs associated with subroutine DREPAR include maintenance costs for minor servicing and repairs, prorated per maintenance period and labor cost for the drill-and-blast crew as provided for in Table 9. Bit and steel cost have previously been calculated in subroutine HOLBRN and are not included as maintenance costs in DREPAR.

C. BORING MACHINE

1. Introduction

An increasing percentage of rock tunnels are being bored by a tunnel boring machine every year. Although conventional blasting continues to be the most economical method for excavating large tunnels in hard abrasive rock, important strides are being made in most of the critical areas of tunnel borer design and these improved designs are making tunnel boring by machine competitive with conventionally excavated tunnels under many circumstances. A summary history of boring machine characteristics and performance was compiled during the early part of this study and was included as Appendix II of the earlier semiannual report.[6]

In the past, the factors which have had the most pronounced adverse effect on the overall average advance rate were:

- Unexpected large variation in tunneling conditions (e.g., major fault zones, squeezing plastic clay, large water inflow)
- Short life of bits, cutters and bearings in very hard rock
- Lack of compatibility between the boring machine and conventional ground control and materials handling systems (a need for an integrated system)
- Major equipment breakdowns resulting from manufacturing problems or operating techniques

Significant advancement of the art has come about, mostly as a result of attempts to design each machine to match the set of geologic conditions expected in each application. As a result, economical use of boring machines in both very hard rock (30,000-45,000 psi) and difficult geology may be foreseen in the next decade.

A representative example of a boring machine project in hard rock is the 19,970-ft-long, 12-ft-diameter River Mountains tunnel on the Southern Nevada Water Project (1968-1970). Rhyolite, rhyodacite, and volcanic lava flows were the principal rock types encountered. The maximum unconfined compressive strength was approximately 16,000 psi.

A Jarva Mark 11-12 tunnel boring machine advanced by a 2-ft stroke at a rate which varied from 0.5 to 6 in./min. Repositioning time was 1 min. On a 7 1/2 hr, 3-shift per day, 5-day work week basis, the average advance attained was 36 ft per shift. Maintenance on the machine was 25% of the available excavation time, much of which was used changing cutters. Each cutter required 30 min to replace. The drive-motor pinion and ring gears, the hydraulic system, and the conveyor drive motor required most of the repair work.

Undoubtedly one of the most significant tunnel boring machine projects in progress at this time is the 30-mi undersea high-speed railway tunnel between Honshu and Hokkaido under the Tsugaru Strait in northern Japan. Three versions of a Swiss-made boring machine, designed by Habegger, Ltd., and now produced by Atlas Copco, Inc., are being used; the first two models are 11.9 ft in diameter for the pilot bore and the third is 13.2 ft in diameter for boring a parallel service tunnel.

In this project, rock quality and strength variations are extreme, ranging from dry volcanic ash of about 4400 psi compressive strength on the Hokkaido side to andesite with numerous water-laden faults and average strength of 40,000 psi on the Honshu side.

Under ideal geological conditions, the second 11.9-ft machine can bore 13 ft/hr, but the adverse conditions under the strait have cut the advance rate to 5 ft/hr with the best one-month advance under 300 ft.

Those activities associated with tunnel boring machine operation, including subroutine names used in the model, are summarized in Table 11.

TABLE 11

BORING MACHINE ACTIVITIES

Subroutine ASSMBL

Boring machine assembly and setup

Subroutine BORE

Rock fragmentation by boring machine head rotation

Subroutine CUTTER

Cutter wear and cutter changing

Subroutine DISASM

Boring machine disassembly

Subroutine REPAIR

Boring machine maintenance and repair

These subroutines will be discussed in sequence in the following paragraphs. Also discussed will be the mathematical representations of boring machine performance as related to the input specifications appearing in Table 12.

2. Subroutine ASSMBL

This subroutine accounts for the time necessary to assemble and check out the boring machine at the tunnel heading prior to excavation. At the present time 160 hr is allocated to this task during which no further advance of the tunnel may be achieved. The user may select a value different from 160 hr if he desires.

TABLE 12

BORING MACHINE PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

Boring Machine Information

1. Rated rotational power (hp)* ____________________
2. Energy required per rock volume broken (in.-lb/in^3 × 10^3)* ____________________
3. Rotational speed of boring head (rpm)* __________
4. Time to assemble boring machine (hr)* __________
5. Time to change bore diameter (hr)* __________
6. Time to disassemble machine (hr)* __________
7. Maintenance parameters (separate from cutter change) ____________________
8. (a) availability of machine (%) __________

   (b) in lieu of (a) average down time per maintenance period (hr)* __________

   (c) average time between maintenance periods (hr)* ____________________

Cutter Information (rolling cutters)

1. Total number of cutters* ____________________
2. Radial location of cutters*

Cutter No.	R (in.)

3. Estimated cutter life as a function of rock abrasiveness* ____________________

	Abrasiveness Index	Travel (ft)
(Least abrasive)	1	
(Moderately abrasive)	2	
(Most abrasive)	3	

4. Time required to replace one cutter (hr)* __________
5. Minimum observed fractional wear of cutter to cause replacement during any one cutter change period* ____________________

TABLE 12 (cont.)

BORING MACHINE PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

1. Plant and Equipment

Item	Ownership (Rental) Cost Per	Hr (check one)	Ft	Unusual Developmental Cost Per	Hr (check one)	Ft
MAJOR ITEMS:						
Boring machine unit						
ADDITIONAL ITEMS:						

2. Job Materials

ITEM	COST		LIFETIME	
	VALUE	UNIT	VALUE	UNIT
MAJOR ITEMS:				
Power: Electric*		$/kw·hr	-	-
Cutters*		$/cutter		
Cutter Bearings (prorated)*		$/cutter chg.		
Minor Servicing & Repairs (per maintenance)		$		
ADDITIONAL ITEMS:				
		$/ft		

3. Direct Labor

Labor Type	Number Required/Shift	Rate $/hr
MAJOR TYPES:		
Machine Operator		
Miners		
Electrician		
Mechanics		
ADDITIONAL TYPES:		

4. Permanent Materials

Item	Cost Value	Unit
(none required)		

*Representative values are programmed into the model. A user may substitute his own values if he desires. In some instances certain constraints must be satisfied to use the internal values. In the case of boring machine horsepower, the tunnel diameter must be between 6 and 20 ft. In the case of energy required, rock strength must be between 5,000 and 30,000 psi.

3. Subroutine BORE

The subroutine BORE represents the breaking of rock at the tunnel face by the thrust and rotation of rolling disc cutters mounted on a boring machine head.

Present state-of-the-art performance of boring machines, for the purpose of the simulation, is derived from curves fit to empirical data. An empirical approach was selected because the mechanism of rock fracture by rolling cutter, carbide inserts, and drag bits is not sufficiently understood at this time to allow physical modeling.*

The rate of advance R of a boring machine can be expressed as

$$R = \frac{P}{AE}$$

where P = power output of the machine

A = tunnel cross section area

E = energy per unit volume of rock broken

For each time increment Δt of the simulation, the incremental advance of the tunnel face, ΔX, and the incremental volume of rock broken from the face, ΔV (unmodified for change in bulk density), may be calculated as

$$\Delta X = R\Delta t$$

$$\Delta V = RA\Delta t$$

---

*Work in progress at this time by W. A. Hustrulid and others to investigate this mechanism of rock fracture may lead to a revised code in the near future.

No accurate information of the actual power output of a boring machine under varying circumstances has been found. As a consequence rated rotational horsepower of the individual machines has been interpreted as power output. The energy per volume has been calculated according to the volume of rock broken off for this amount of rated horsepower available. The machines generally operate at some undetermined fraction of rated horsepower. Yet for our purposes, this simplification which yields a consistent set of data which allows prediction of rates of advance from machine rated horsepower is desirable. It might be noted that the added horsepower used to drive the hydraulic system, which is separate from the rotational power, is not included in rated horsepower. It is generally less than 10% of the rotational power.

There is a fairly consistent trend to greater machine horsepower with greater tunnel diameter. Figure 21 shows a horsepower curve which is an approximate fit to Robbins Boring Machine data and represents Robbins rolling disc cutter type machines. Other data points shown in Fig. 21 are included to show the scatter of different types of machines--Habegger uses drag bits rather than rolling cutters--and manufacturer experience (some data are first generation machines). This horsepower trend represents no more than historical information and it may not necessarily represent the correct machine horsepower for a given situation; it nevertheless reasonably represents state-of-the-art machine characteristics and as such has been included in the simulation as a relationship to provide the power value which is used if the user does not specify one. This relationship is restricted to be used to represent machine horsepower for tunnel diameters between 6 and 20 ft. There are not sufficient data to insure the relationship is valid for very small or very large tunnels. The relationship is

$$hp = 40e^{0.19d}$$

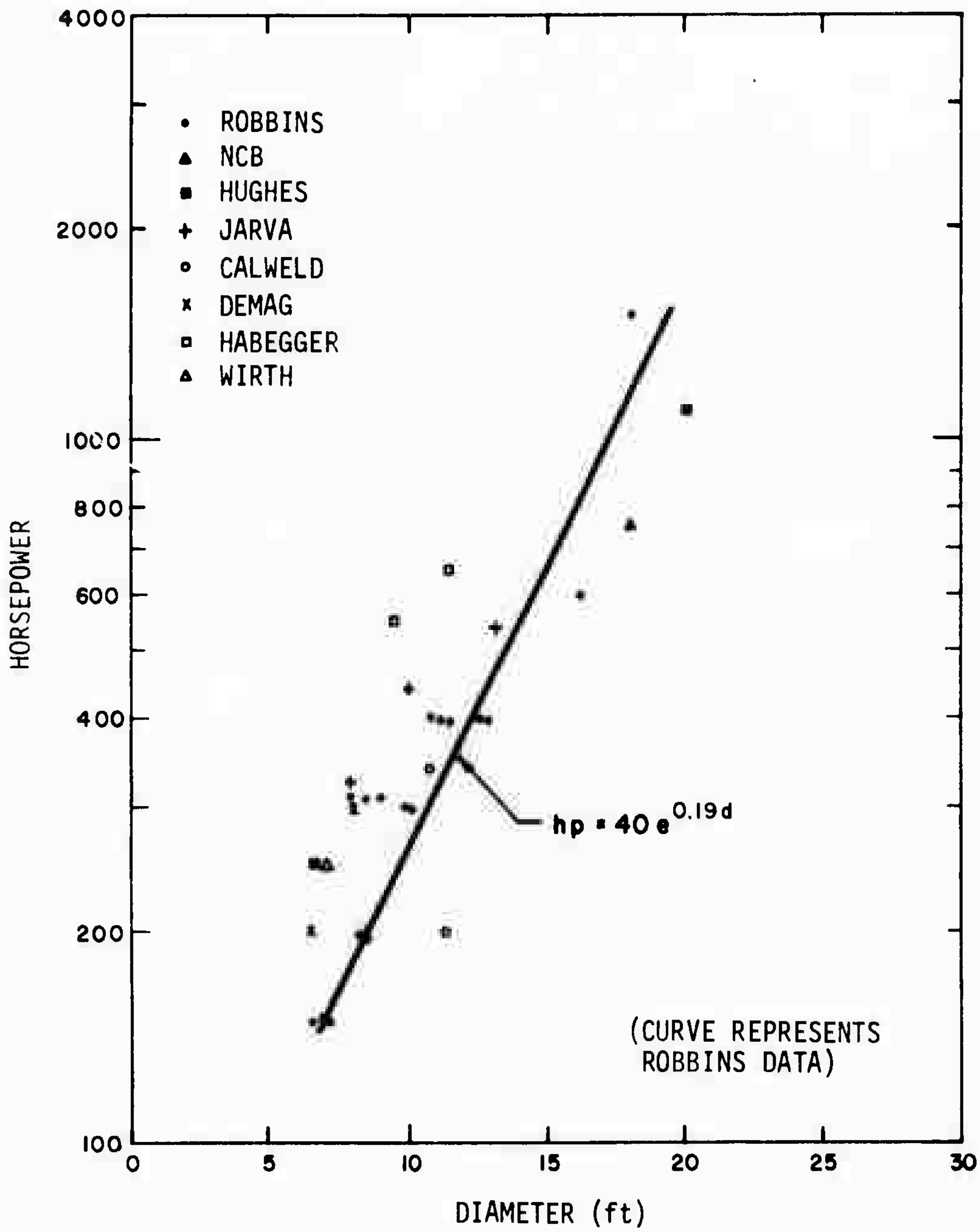

Figure 21. Horsepower versus Tunnel Diameter Trend for Tunnel Boring Machines

where $\quad hp$ = rated rotational horsepower

d = tunnel diameter, ft

Some representative data for the energy per unit volume of rock broken by boring machines is plotted in Fig. 22. The data of Muirhead and Glossop[28] shown in this figure was used to derive the relationship which is presently incorporated into the computer subroutine BORE to represent state-of-the-art performance of boring machines. Subsequently, representative data for numerous separate tunneling projects compiled by the authors was added to show the scatter of boring machine performance from one project to the next. Note that the minimum value of this energy value reflects the best attainable performance from a boring machine. Therefore, the derived relationship

$$E = 1.8e^{0.084\sigma}$$

where $\quad E$ = energy per volume rock broken, $\left|(\text{in.-lb/in}^3) \times 10^3\right|$

σ = rock compressive strength, 10^3 psi

which represents the performance of a boring machine for rock strengths between 5000 and 30,000 psi fitted to the Muirhead and Glossop data, appears to be an adequate "best performance" curve for tunneling machines. Use of this relationship for rock strengths beyond these limits should be done with reservation because of the lack of data. The subroutine BORE is terminated and an error message results if the user has not specified an energy value and the rock strength value falls outside these limits. Systematic acquisition of data from several boring machine projects may identify what factors, in addition to compressive strength, control the energy required to break the rock. Cutter spacing, thrust and rotation, rock quality, and boring head rotational speed may each be significant. It has not yet been possible to ascertain the degree of significance of these other factors.

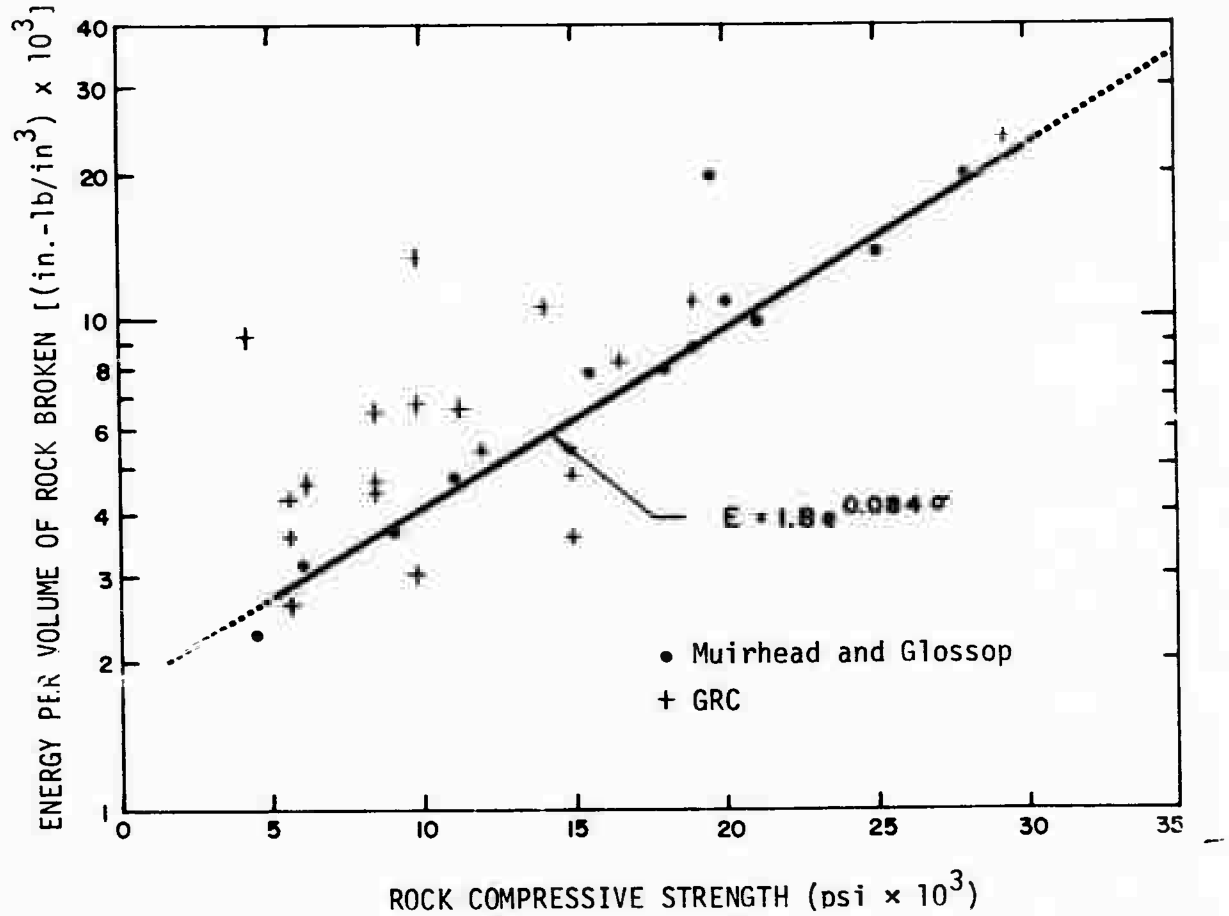

Figure 22. Representative Boring Machine Performance

For reference, Fig. 23 gives approximate ranges of compressive strengths for some common rock types.

A review of the usage of boring machines in a series of projects for the Bureau of Reclamation,[29] along with discussions with manufacturers, reveal that typical manpower requirements per shift associated with a boring machine are as given in Table 13. It should be emphasized that the numbers given as labor requirements can and will vary according to the efficiency of the contractor. However, Table 13 represents realistic average manpower figures and may be used as a preliminary guide when providing input to the model (Table 12).

TABLE 13

BORING MACHINE (DIRECT LABOR)
MANPOWER PER SHIFT

Tunnel Diameter	8'-14'	14'-20'	20'-30'	Washington, D.C., Area Hourly Costs*
Machine Operator	1	1	1	$8.32
Miners	2	3	4	$7.20
Electrician	1	1	1	$8.92
Mechanic	1	1	1	$8.10

*Includes 25% fringe benefits, FICA, etc.

The job material (or consumable item) cost associated with the activity BORE is that for electrical power to operate the machine. (Cutter costs are treated separately in the next section.)

Power cost is estimated from the rated horsepower of the machine motors, the time that the machine is in operation boring the rock face, and the input unit of electricity cost per kilowatt-hour.

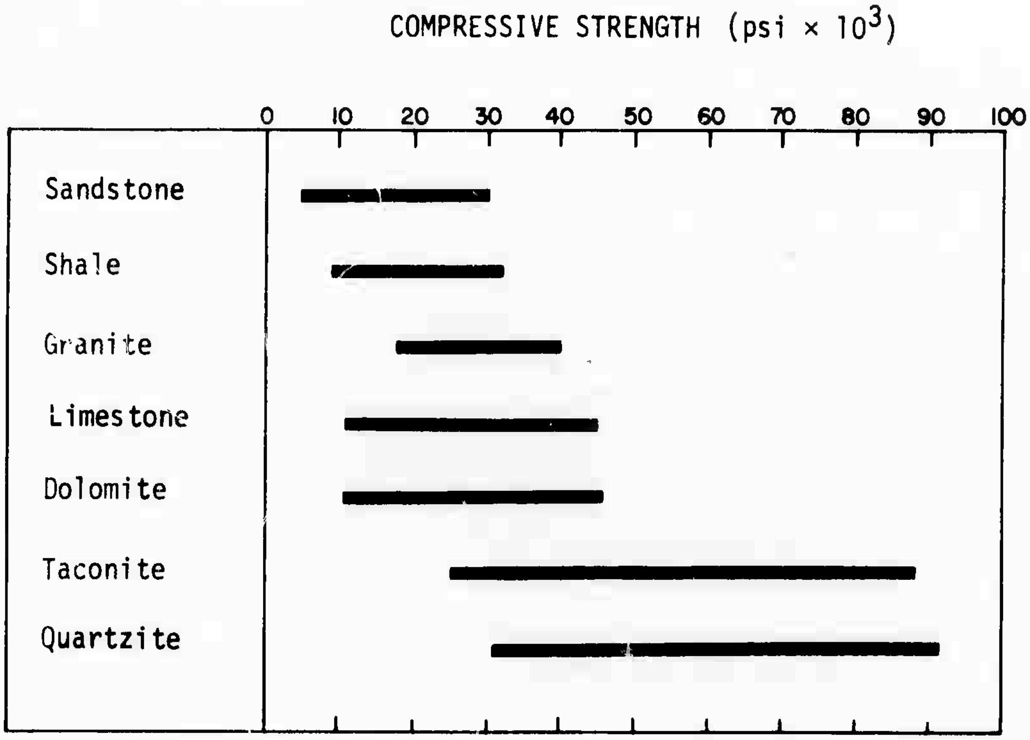

Figure 23. Range of Compressive Strength for Some Common Rock Types

$$(\$)\ \text{Power Cost} = (\$/\text{kW-hr})\ \frac{\text{hp}}{1.34}\ \Delta t$$

where Δt = time in hours. If the user has not assumed an alternative unit cost for electricity is assumed to be \$0.015 per kW-hr.

Plant and equipment costs for the boring machine can be estimated from the cost of boring machine plus the additional cost for the power transmission system.

Representative cost of a boring machine can be seen from the plot in Fig. 24 roughly to be a function of the installed horsepower,

$$\text{Machine Cost} = \$1000 \times \text{hp}$$

These results, which are derived from actual costs, give a guide to the capital costs involved. The lifetime of machines will vary according to the conditions of use and the maintenance provided. However, a formula which may be used to approximate machine cost per linear foot of tunnel driven, if the machine is not to be depreciated to zero over the project, is

$$\frac{\text{Machine Cost } (\$)}{(10{,}000\ \text{hr}) \times (\text{Estimated Penetration Rate})\ (\text{ft/hr})} = \$/\text{ft}$$

If the machine is depreciated over the total length of the tunnel this would be

$$\frac{\text{Machine Cost } (\$)}{\text{Total Tunnel Length (ft)}} = \$/\text{ft}$$

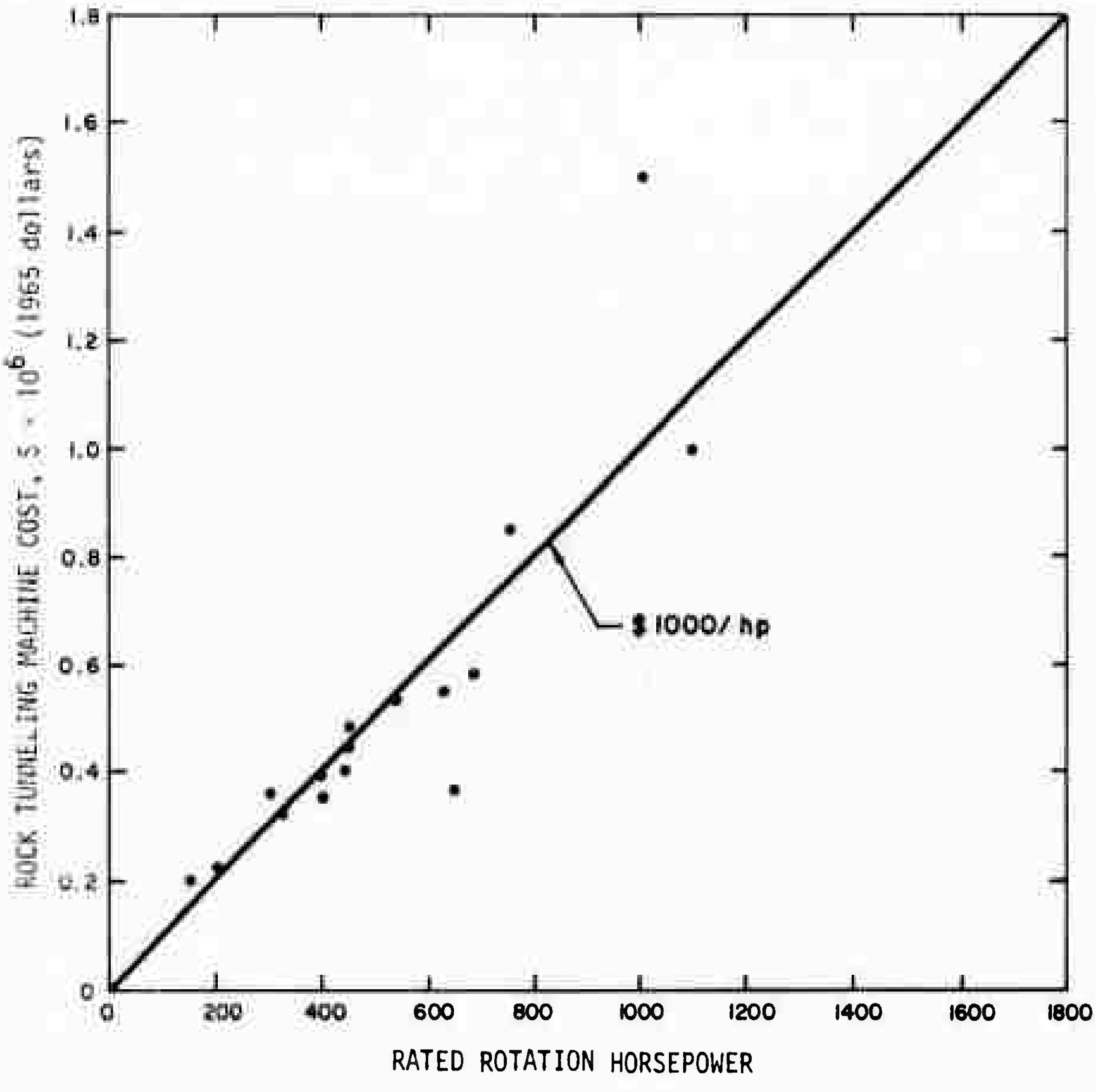

Figure 24. Rock Tunneling Machine Costs Related to Machine Rated Horsepower

If the machine is to be depreciated by operational hour,

$$\frac{\text{Machine Cost (\$)}}{\text{Operating Lifetime (hr)}} = \$/\text{hr}$$

An operating lifetime of 10,000 hr is representative. The time is calculated as that in which the machine is in actual operation; down time is not included. There is an additional cost for the power transmission system:

$$\text{Cost of Transmission System} = \$3.40 \times \text{Length (ft)}$$

4. Subroutine CUTTER

A major limitation of boring machine performance in hard rock is due to the frequent changes of worn cutters required. Subroutine CUTTER accounts for this cutter wear and replacement. An example of a boring machine cutter head showing cutter spacing is illustrated in Fig. 25. The machine shown is a Robbins Co. Model No. 142-139 which bores a 14-ft-diameter tunnel.

Frequency of cutter change depends on the rock strength and abrasiveness, the number and spacing of the cutters, the tunnel diameter, the boring machine head rotational speed, the thrust per cutter, cutter hardness, and a number of other factors. While it is current practice to try to schedule cutter replacement during general maintenance and repair of the machine (usually performed on a weekend or night shift), the harder and more abrasive rocks cause sufficiently rapid cutter wear to require cutter replacement several times a shift thereby delaying progress.

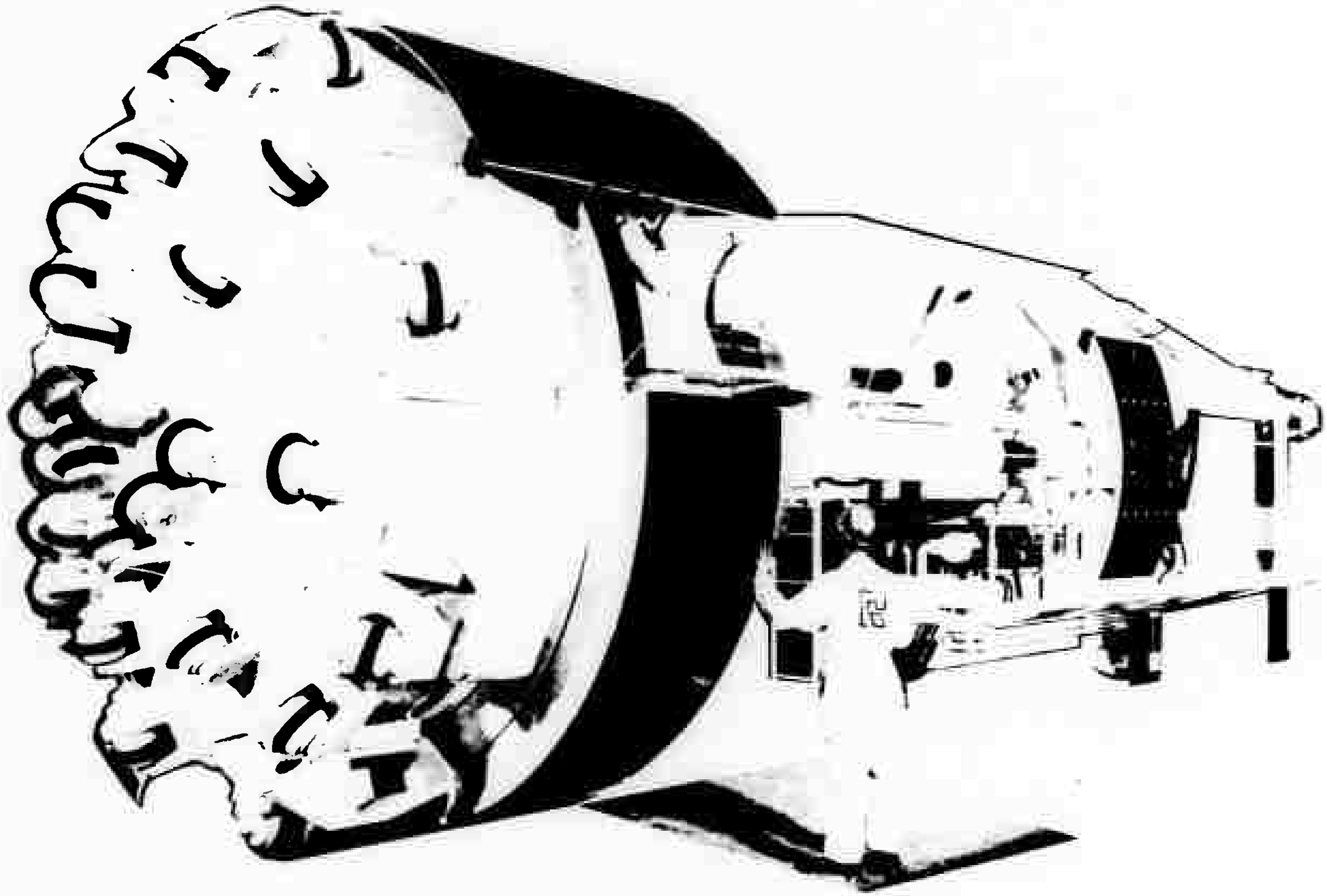

Figure 25. Tunnel Boring Machine Showing the Cutter Head
(Courtesy of Robbins Co.)

Many manufacturers estimate cutter costs by first assuming a cutter layout and taking the sum of radii of all cutters to find an average radius and corresponding average cutter circumference traveled during one revolution. The cutters are assumed to be able to travel a given number of linear feet while rolling against the rock face before wearing out. Typical figures are 400,000 linear feet for a sandstone and 700,000 to 1,000,000 linear feet for shales.[30] The figure is primarily dependent on the relative abrasiveness of the rock. In estimating the abrasiveness, one can use a variety of tests to determine the mineral content and grain size to produce a weighted Mohs' hardness for the rock (Fig. 26).

Figure 27 shows the estimated cutter costs in dollars per cubic yard of material removed as a function of the rock hardness. The curves for the three different types of rocks cover the range of expected abrasiveness as reported by J. P. Carstens et al.[31] and weighted to reflect current capabilities. The design of cutters is a constantly and rapidly evolving technology, and with this evolution the cutter costs are going down with experience. Caution must be applied to the use of these curves since the experience on which they are based was severely limited above 25,000 psi hardness. There may exist some limiting maximum rock hardness through which present-day cutter materials will not penetrate. Further research is necessary to identify what this limiting value might be.

Polynomial expressions that approximate the curves of Fig. 27 are (σ = compressive strength $\times 10^3$ psi):

Limestone (least abrasive)

$$(\$/yd^3)\ \text{Cutter Cost} = .216 + .844\sigma + .997\sigma^2/10^3$$

for $10 < \sigma < 45$

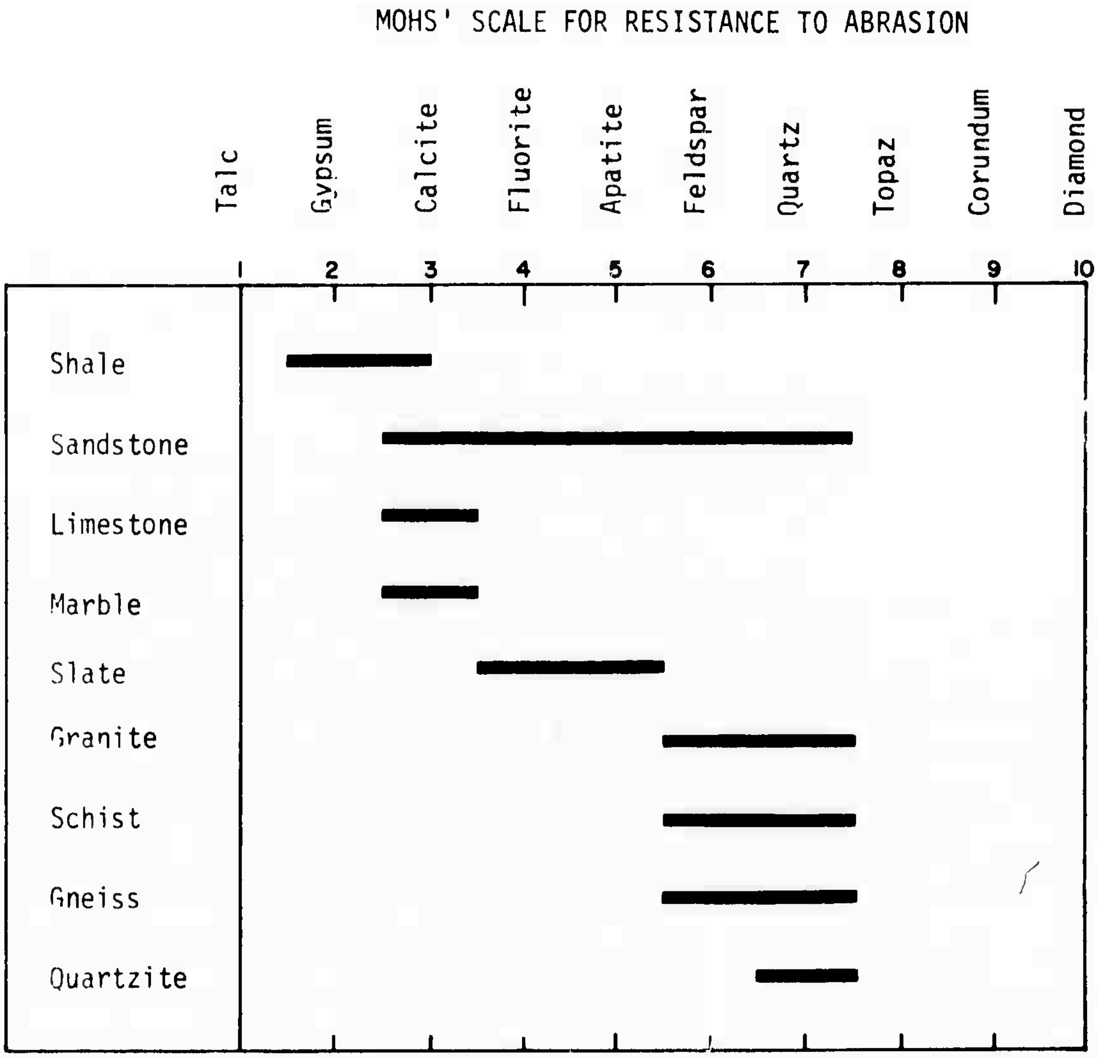

Figure 26. Approximate Mohs' Ratings for Some Common Rock Types

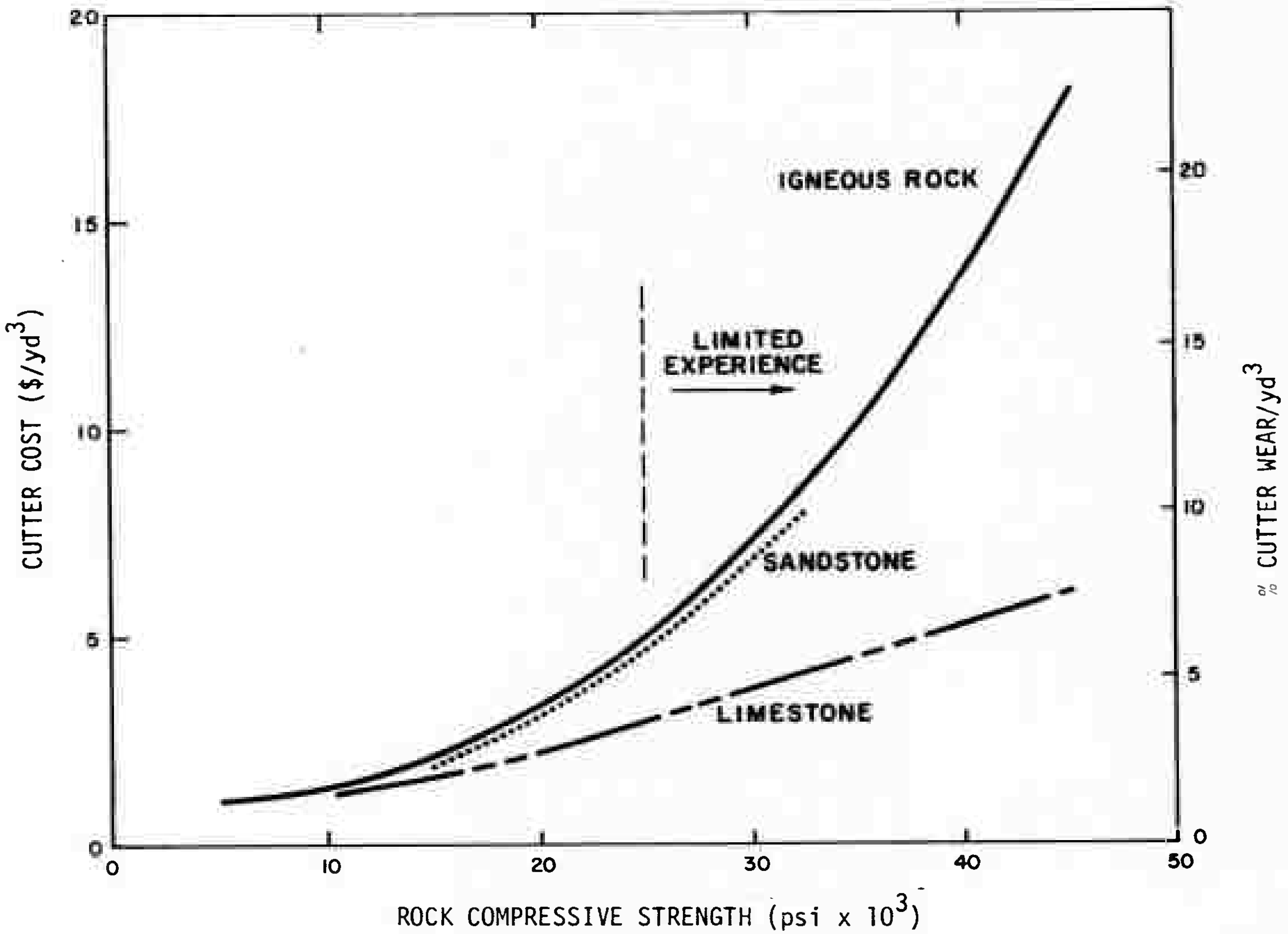

Figure 27. Hard Rock Cutter Bit Costs--Present Day Best Experience

Sandstone (medium abrasiveness)

$$(\$/yd^3) \text{ Cutter Cost} = .7 + .257\sigma/10^2 + .442\sigma^2/10^2 + .815\sigma^3/10^4$$

for $15 < \sigma < 33$

Igneous Rock (most abrasive)

$$(\$/yd^3) \text{ Cutter Cost} = .883 + .257\sigma/10^2 + .442\sigma^2/10^2 + .815\sigma^3/10^4$$

for $10 < \sigma < 45$

For the purpose of modeling cutter wear and replacement as part of a system simulation, however, this averaging method was judged unsatisfactory to account for the rapid cutter wear and more frequent replacement of the outer cutters relative to the inner ones. Particularly it fails to provide a measure of improvement which could be made possible by the replacement of gauge cutters (at the periphery) and other outer cutters with some novel device which could kerf and break rock in this region without concomitant rapid wear and frequent need for replacement (say, for example, an electron beam gun). It has been stated that this rapid wear of the gauge cutters* is a major factor causing the overall low rate of advance of a boring machine through very hard rock, 30,000 psi and up.[32]

The approach taken in the model to cutter wear and replacement is therefore to consider each cutter separately rather than by averaging. As seen in Table 12, the required input to the simulation specifies each cutter by its radial position on the cutter head. The expected lifetime

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*It is commonly believed that this rapid wear is due to the more severe tangential stresses applied to these cutters and to their repeated travel through the broken rock in the invert.

of a cutter in terms of linear feet of travel before replacement is necessary is specified for three degrees of rock abrasiveness: least abrasive, moderately abrasive, and most abrasive. This simplification of rating abrasiveness on a three-level scale is a first step toward relating cutter wear in greater detail to geologic parameters as greater understanding of this relationship is achieved. For the present model it is assumed that shale, limestone, and marble would be rated as least abrasive; medium sandstone and slate moderately abrasive; and the harder sandstone, granite, schist, gneiss, and quartzite would be most abrasive.

The fractional wear of any cutter incurred during boring time Δt is calculated as

$$WEAR_k = \frac{2\pi(rpm)\ (r_k/12)\ (\Delta t \times 60)}{CL_i}$$

$$= \frac{10\pi(rpm)\ (r_k)\ (\Delta t)}{CL_i}$$

where $WEAR_k$ = fractional wear of cutter k

rpm = boring head rotational speed (rpm)

r_k = radial distance of cutter k (in.)

Δt = time increment (hr)

CL_i = cutter life in rock of abrasiveness index i through which the cutter is traveling (ft)

When any cutter on the boring machine is completely worn (cumulative wear $\geq$ 1), the boring operation is delayed and the cutter is changed. The model provides that, when one cutter is being replaced because of 100% wear, other cutters which are worn beyond some wear criteria (e.g., 75%) are also replaced. This feature is incorporated into the model to allow the user the opportunity of measuring the improvement, if any, of boring machine rate of advance by changing more cutters at a time.

If the time required to replace a cutter is not specified by the user, it is assumed to be 30 min.

The cost of a rebuilt disc cutter is assumed to be \$80 if not otherwise specified. To these cutter costs, which are accumulated as job material expenses, is added the cost of cutter bearing and housing replacements. This is an event that on the average must be performed every six changes in cutter bit, and its cost has been prorated over each cutter replacement as follows:

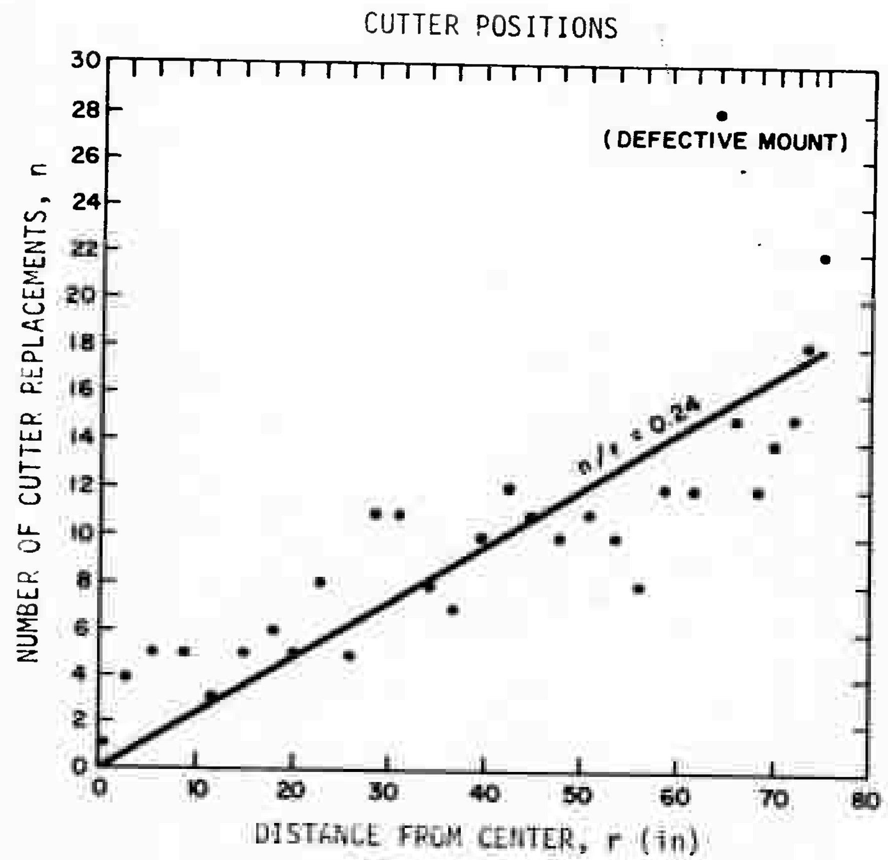
Prorated Bearing and Housing Cost = \$310/cutter change

An example showing how one determines the average cutter life from field data obtained during an actual boring operation is shown in Fig. 28. The tunneling project being considered in this case is the Layout Tunnel described in greater detail in Sec. VI. In this figure are plotted the number of changes for each of the 29 disc and center tricone cutters that were made during the project, in which 5038 ft of tunnel was bored during 438.2 machine operating hours.

Assuming, as above, that the cutter wear is proportional to the distance traveled, the best fit to the data produces a straight line having a slope of 0.24 and intercepting the origin. The average cutter life may be calculated as:

$$\text{LIFE} = \frac{2\pi(\text{rpm})\ (\text{Time} \times 60)}{12\ (n/r)}$$

$$= \frac{10\pi(\text{rpm})\ (\text{Time})}{(n/r)}$$



OPERATING DATA

BORED LENGTH	5038 ft
DIAMETER	12 ft 11 in.
TIME	438.2 hr
RPM	6
GEOLOGY	SANDSTONE, SHALE, AND VERY HARD CONGLOMERATE

AVERAGE CUTTER LIFE

$$\text{LIFE} = \frac{2\pi(\text{rpm})(\text{TIME} \times 60)}{12\,(n/r)} = \frac{10\pi(\text{rpm})(\text{TIME})}{(n/r)}$$

$$= 344{,}000 \text{ ft}$$

Figure 28. Cutter Life Calculated from Cutter Replacement Data (Layout Tunnel Example)

where LIFE = average cutter life (ft)

rpm = boring head rotational speed (rpm)

Time = total boring time (hr)

n/r = slope of straight line with zero intercept which best fits field data (n is the number of cutter replacements, r is radial distance in inches)

In this example, a cutter life of 344,000 ft was calculated for boring a geology consisting primarily of medium-to-hard sandstone intermixed with very hard conglomerate. Boring head rotational speed is assumed to be 6 rpm, the designed speed produced by the six 100-hp drive motors.

5. Subroutine DIAMTR

Subroutine DIAMTR is included to account for the time spent changing the boring machine to a marginally larger or smaller diameter by adding or removing peripheral cutters. Unless otherwise specified, the time required to perform this activity is 24 hr.

6. Subroutine DISASM

Subroutine DISASM accounts for the time spent disassembling the boring machine at the end of a project or at a point in the project where use of a boring machine is no longer feasible and some other means of fragmentation must be used. Appropriate instructions in the user-provided control program would specify when this subroutine would be used. The time period, if not otherwise specified, would be identical to that used for the subroutine ASSMBL: 160 hr.

7. Subroutine REPAIR

Subroutine REPAIR accounts for boring machine maintenance and repair periods, exclusive of cutter changing which has been accounted for separately. In terms of performance parameters, the user provides either a machine availability or down time per maintenance and the time between maintenance periods.

The information on characteristic periods of time for maintenance which was included in Appendix II of the semiannual report[6] may serve as a preliminary guide to scheduling these activities in the model. Further information on a specific project, the Layout tunnel, is provided in Volume II of this report.

D. WATER JET

1. Mathematical Discussion

High-velocity water jets, both steady and pulsed, are of interest for rapid excavation in hard rock because such jets have been shown capable of fracturing the hardest rock by high impact pressure and fluid shear forces. Rock disintegration by jet impact, utilizing the dynamic and static mechanical stresses induced in the rock, appears adaptable to a wide variety of geologic conditions, rock types, and environments, and may be particularly suitable for arbitrary geometries of excavation as well. As potential rock disintegration devices in a rapid excavation system, water jets offer the attractive advantages of minimal cutting tool wear and flexible response to a wide variety of conditions.

The performance of water jets is determined in part by the physical laws governing jet impact, which are described in terms of the dynamic equations of water jet motion. The following summary of the fundamentals of water jet dynamics is intended to provide a basis for understanding the relationship between pressure, nozzle size, jet velocity, impact cross-sectional area, and power required. Several recent thorough reviews of water-jet performance are available and can provide details that are omitted here for brevity.[33,34]

For a frictionless nonconducting fluid in motion with a steady pressure distribution, the quantity h defined by

$$h = \frac{1}{2}V^2 + e + \frac{p}{\rho} + \psi$$

has the same value at all points along the path of an element of the fluid. In terms of energy per unit mass:

h = total energy

$\frac{1}{2}V^2$ = kinetic energy

e = internal energy

$\frac{p}{\rho}$ = potential energy associated with the pressure field

ψ = potential energy associated with the external body-force field

If fluid flows steadily from a large reservoir through a small nozzle to form a high-velocity jet, the process is one of converting the potential energy associated with the reservoir pressure field into kinetic energy of the jet. If changes in internal energy and body-force field are negligible, and further if $(p/\rho)_{reservoir} >> (p/\rho)_{jet}$, both of which are true for the water jets of interest, then the following relationship provides a good approximation of the jet velocity:

$$V = \left(2\frac{p}{\rho}\right)^{1/2}$$

where V = jet velocity

p = reservoir pressure

ρ = reservoir density

The stagnation pressure at impact, that maximum of pressure attained by isentropic conversion of all kinetic energy into potential energy associated with the pressure field, is the reverse of the jet formation process:

$$p = \frac{\rho V^2}{2}$$

where p = stagnation pressure

ρ = stagnation fluid density

V = jet velocity

If the velocity across the nozzle cross section is regarded uniform, the mass flow rate through the nozzle is given by

$$w = \left(\frac{\pi d^2}{4}\right) \rho V$$

where w = mass flow rate

d = nozzle diameter

ρ = fluid density

V = jet velocity

The jet power (kinetic energy per unit time) is given by:

$$P = w\frac{V^2}{2} = \frac{\pi d^2 \rho V^3}{8}$$

The jet power may also be expressed in terms of reservoir pressure,

$$P = \frac{\pi\sqrt{2}\, d^2 p^{3/2}}{4\sqrt{\rho}}$$

or, for water, simply

$$P = 0.0174\, d^2 p^{1.5}$$

where P = horsepower

d = nozzle diameter (in)

p = nozzle reservoir pressure (psi)

A jet striking a flat surface is transformed into a sheet of fluid which flows radially outward from the stagnation point. Physically what occurs in the region of impact is a conversion of kinetic energy to the potential energy associated with a high pressure in the region of impact, pressure which serves to deflect the jet streamlines from the incoming direction to lines parallel to the impact plane spreading out radially. Momentum is conserved and the stream velocity regains its original value, the jet velocity, as it departs radially from the impact region. The sheet thickness, t, will be inversely proportional to the radial distance from the stagnation point in this region where the velocity has regained its original value:

$$t = \frac{d^2}{8r}$$

where t = sheet thickness

d = nozzle diameter

r = radial distance from stagnation point

From momentum considerations, the total force, F, on the flat surface is given by:

$$F = \rho V^2 \frac{\pi d^2}{4}$$

This is also equal to the integral of the pressure over the entire surface:

$$F = 2\pi \int_0^\infty pr\ dr$$

No complete description of the pressure distribution p(r) has been given for any three dimensional case. Leach and Walker[35] assumed, for an approximate solution,

$$\frac{p}{\frac{1}{2}\rho V^2} = 1 - 3\left(\frac{r}{R}\right)^2 + 2\left(\frac{r}{R}\right)^3$$

where R is the finite radial distance at which the pressure is regarded essentially equal to ambient pressure.

It can be shown that for the assumed pressure distribution given by the above equation,

$$\frac{R}{d} = \sqrt{\frac{5}{3}} \simeq 1.3$$

from which it appears that the pressure effects of the jet on the surface are confined to within a radius of approximately 2.6 jet radii. This agrees with test results obtained by Leach and Walker for jets of 315 and 560 atmospheres, at standoff distances of 330 and 76 nozzle diameters (Fig. 29).

For standoff distances less than about 500 nozzle diameters, the dispersion of the jet and degradation of impact pressure is apparently not significant. As distance is increased further, however, the total force of impact of the jet on the surface begins dropping rapidly. The results of Semerchan et al.[36] for jets of 50, 1000, and 1500 kg/cm^2 indicate at least a 50% reduction in the momentum of the jet at a standoff distance of 1500 nozzle diameters.

The total power required to operate a fluid jetting device goes up rapidly with increases in jet velocity or nozzle diameter. To produce a jet at 200,000 psi continuously through a 0.4 in. nozzle requires approximately 250,000 hp.

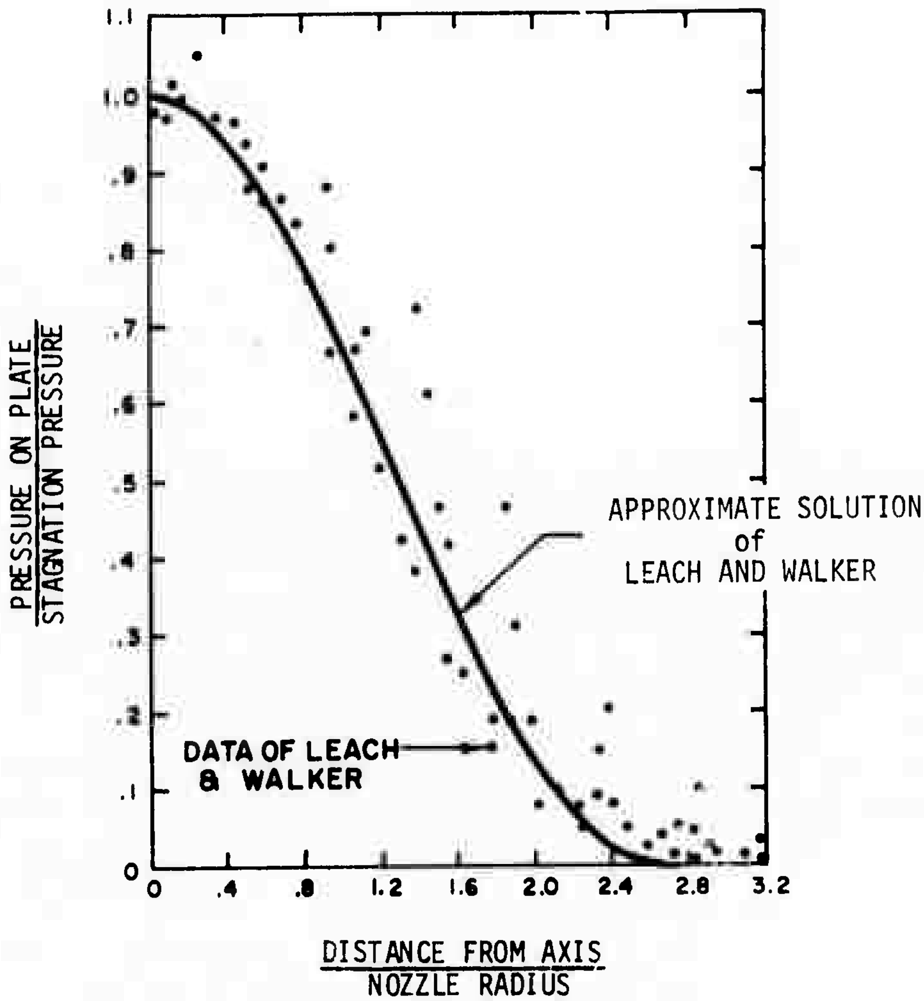


Figure 29. Pressure Distribution on a Surface at Right Angles to a Jet.[35]

To avoid this high power requirement of a large-nozzle, high-pressure, continuously jetting system, Singh and Huck,[37,38] Cooley et al.,[33,39] and others have selected the mode of intermittent jet pulses for investigation.*

2. Jet Use Effects

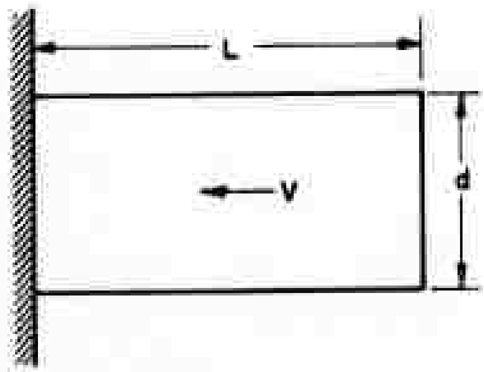
Any analysis of the effects of periodic, short-duration, high-pressure pulses of water must include consideration of the effects of unsteady hydrodynamics, shock-wave propagation, and compressibility of the fluid and solid media. Adequate coverage of these factors is available elsewhere,[33] and only a brief summary will be presented here.

Neglecting any cushioning of impact due to containment of air between the impacting liquid jet and the solid surface, both being regarded as flat, the force of impact between a cylindrical slug of liquid and the surface at any instant will be equal to the change in momentum occurring at that instant.

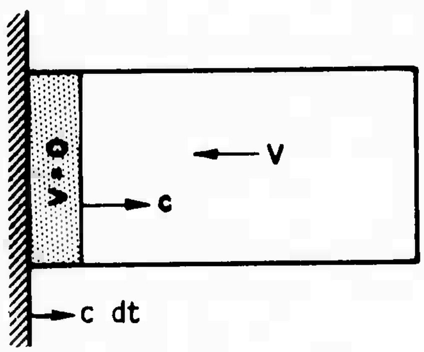
Immediately before impact, the cylinder of fluid has a linear momentum equal to $\rho\pi d^2 LV/4$ (Fig. 30a), where L equals the length of the fluid. After time dt has elapsed from the time of initial impact, a shock wave has propagated into the liquid a distance of c dt, c being the shock propagation speed. Within this distance of the impact interface, the speed of the liquid is essentially zero (Fig. 30b). The solid surface is regarded as completely rigid.

---

*One recently completed prototype design for a pulsed water jet for rock tunneling experiments calls for a jet at pressures of 300,000 to 1,000,000 psi, frequency of one pulse every 5 min (or modified to fire 20 pulses/min), and energy per pulse of 93,500 ft-lb. The jet diameter is 0.27 in. Prototype fabrication will be funded by the U.S. Department of Transportation, Office of High Speed Ground Transportation.



(a) Instant of Impact; Momentum = $\frac{\rho \pi d^2 L V}{4}$



(b) Time dt after Impact: Momentum = $\frac{\rho \pi d^2}{4}(L - c\ dt)V$

Figure 30. Impact of a Fluid Cylinder on a Rigid Surface

It should be noted that the speed of propagation of the shock into the oncoming jet, c, is not independent of the impact pressure, but becomes more rapid for higher pressures. This speed has been measured experimentally by several investigators and summarized by Cook, Keyes, and Ursenbach (Figs. 31, 32).[40]

The change in momentum occurring at the instant of first impact thus determines the impact force, F, on the solid surface:

$$F = \frac{\rho \pi d^2}{4} cV$$

or expressed as pressure over the impact surface,

$$p = \rho c V$$

The elastic response of the solid surface, the irregular shape of the impact surfaces, and any cushioning effect of air trapped between the fluid slug and the surface will reduce observed pressures below this maximum. Brunton[41] states, for example, that taking the elastic deformation of the solid into account reduces the pressure according to the following relationship:

$$p = \frac{\rho_1 c_1 \rho_2 c_2 V}{\rho_1 c_1 + \rho_2 c_2}$$

where ρ_1, ρ_2 and c_1, c_2 are the respective densities and shock propagation speeds in the liquid and solid.

The duration of this peak transient shock pressure is governed by the time it takes for the release of high pressure in fluid downstream of the shock wave to occur. This release is produced by an outward expansion of the walls of the fluid slug in this high pressure region by expansion waves. Such an expansion propagates into the fluid at a speed

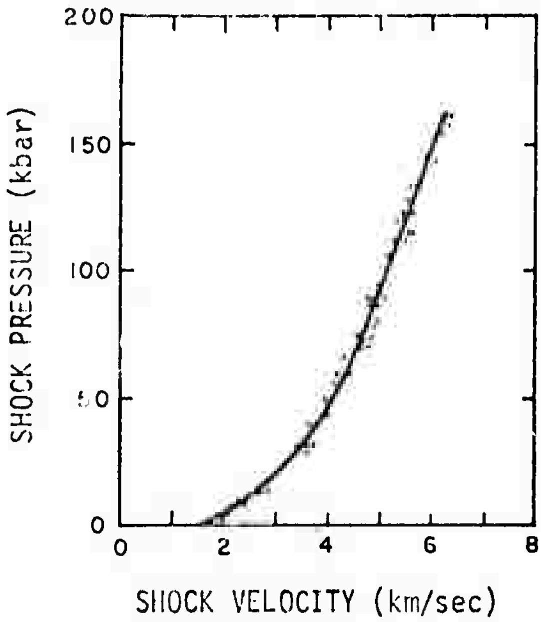


Figure 31. Experimental Shock Velocity vs. Pressure Data for Water[40]

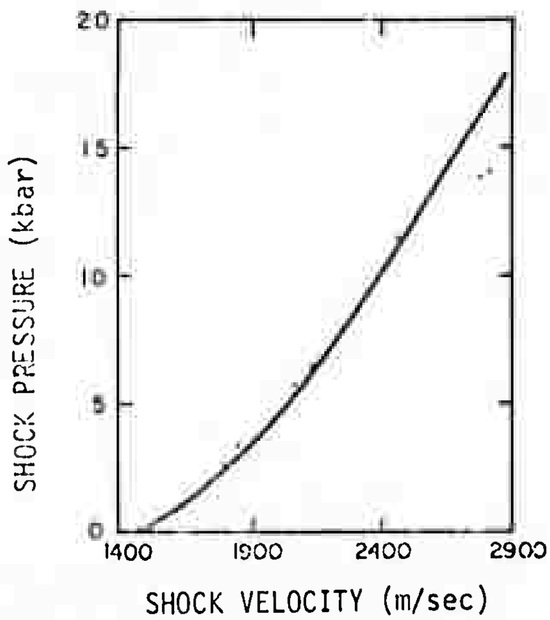


Figure 32. Experimental Shock Velocity vs. Pressure Data for Water (Low Pressures)[40]

of sound and indicates that this peak pressure should last only on the order of 1 μsec for a 3-mm-diameter jet slug.

Although its duration is short, the peak impact pressure is at least 4 times as great as the subsequent steady-flow stagnation pressure (Fig. 33) and may produce a considerable effect on the rock surface. In fact, it appears from a consideration of rock fracture modes that the high peak impact pressure which produces shattering and stress wave failure in brittle rock may be a more efficient means of rock disintegration than the crushing and erosion caused by steady flow.

One further consideration should be mentioned. Preliminary results of studies comparing effectiveness of different sizes of nozzles are inconclusive. There is some indication that larger nozzles are more efficient,[39,42] but for the range of pressures needed to fracture hard rock, the data are not sufficient to draw any conclusion.

The amount of energy expended for any rock breaking process will depend not only on the energy supplied, but also on the fraction that is absorbed by the rock, the size of rock fragments formed, and the mode of failure of the rock. A variety of water jet devices, both pulsed and continuous, have been tested in the laboratory and have given a wide range of energy values.

Figures 34, 35, and 36 summarize for three types of rock the results of Oak Ridge National Laboratory studies of rock fragmentation by a continuous jet of water.[42]

For pulsed jets, the scatter of results obtained for different jet devices is given in Fig. 37 for some of the harder (greater than 10,000 psi) rock samples tested in separate laboratory studies over the past six years. The data of Leach and Walker[35] included in this figure have been calculated from their reported depths of penetration, assuming

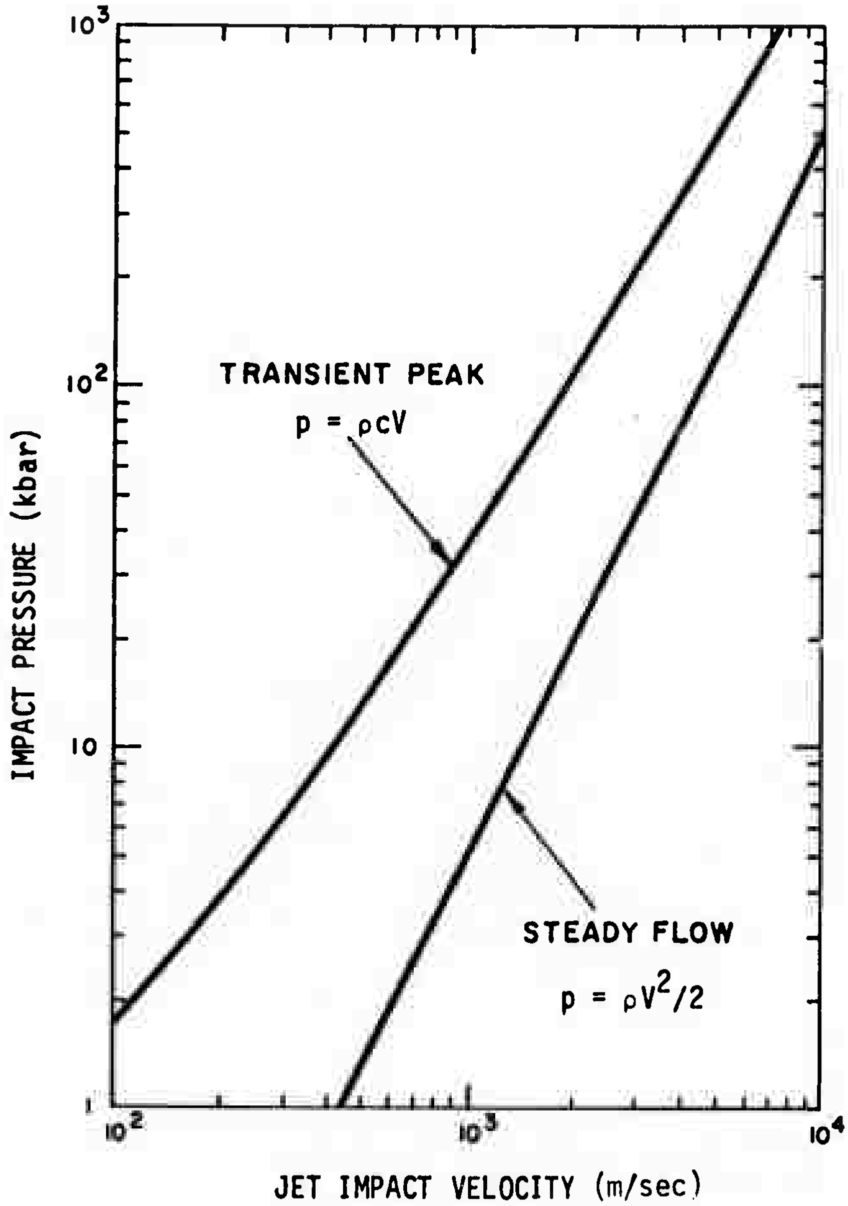


Figure 33. Transient Peak and Steady Flow Impact Pressures versus Jet Impact Velocity for Water (20°C)

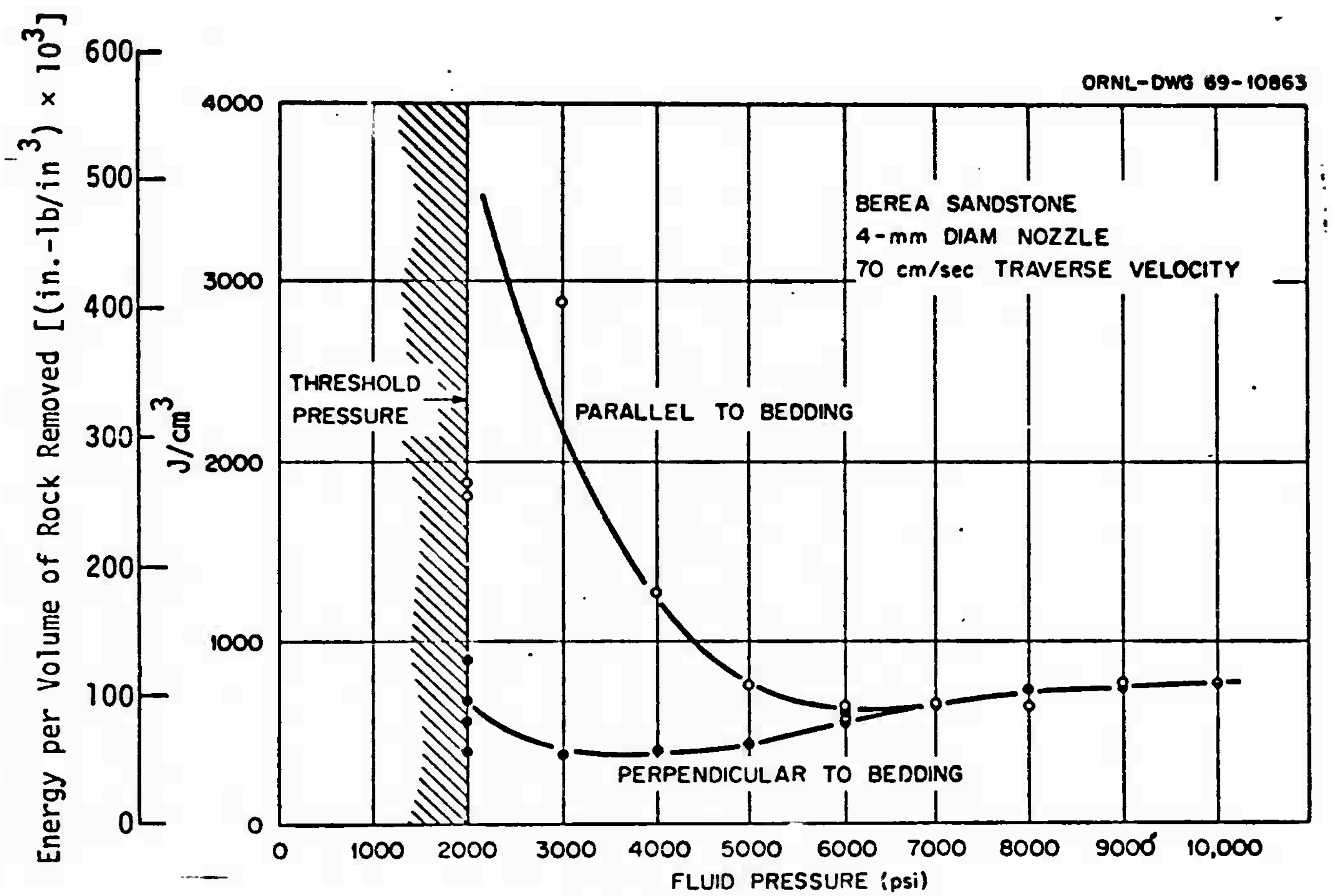


Figure 34. Continuous Water Jet Performance in Berea Sandstone[42]

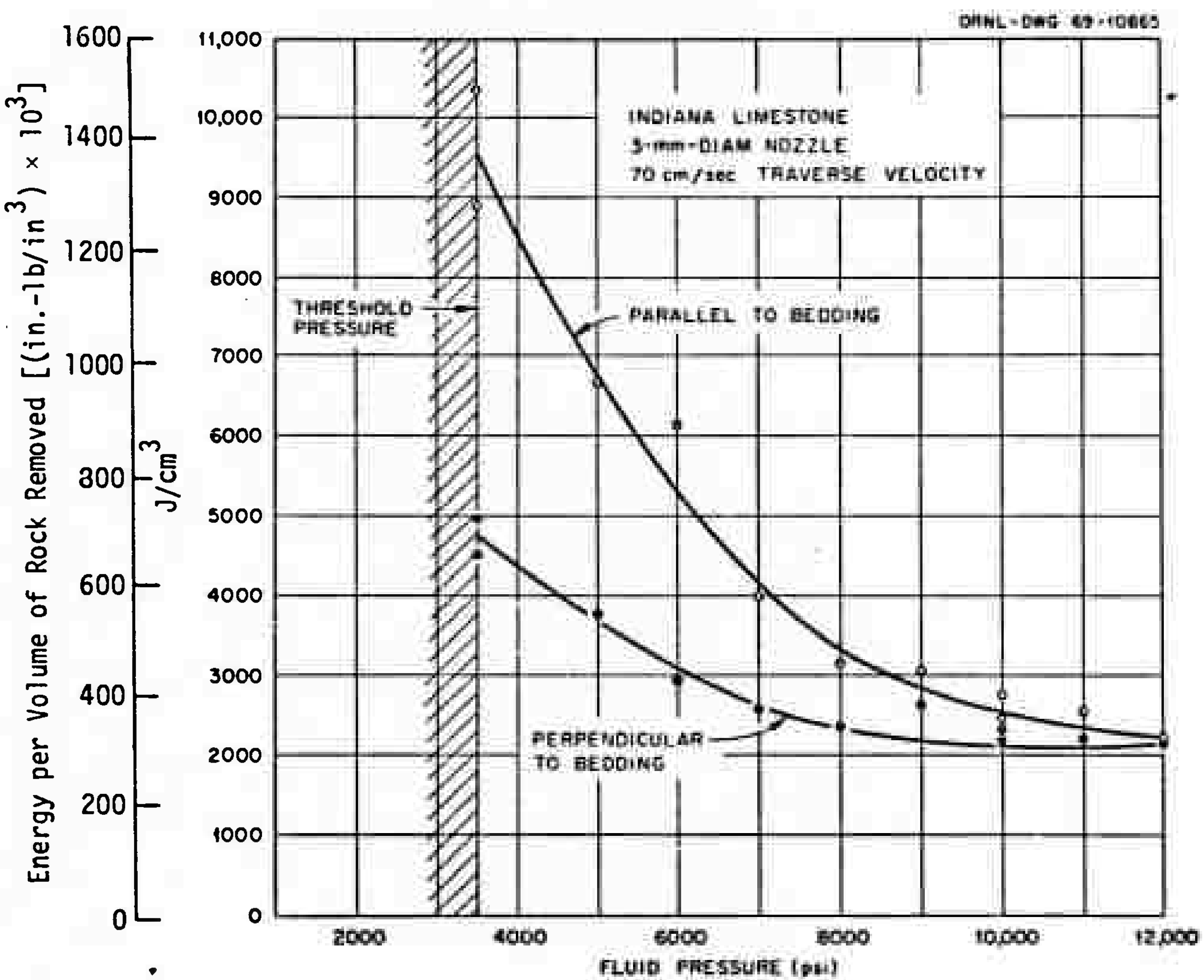


Figure 35. Continuous Water Jet Performance in Indiana Limestone[42]

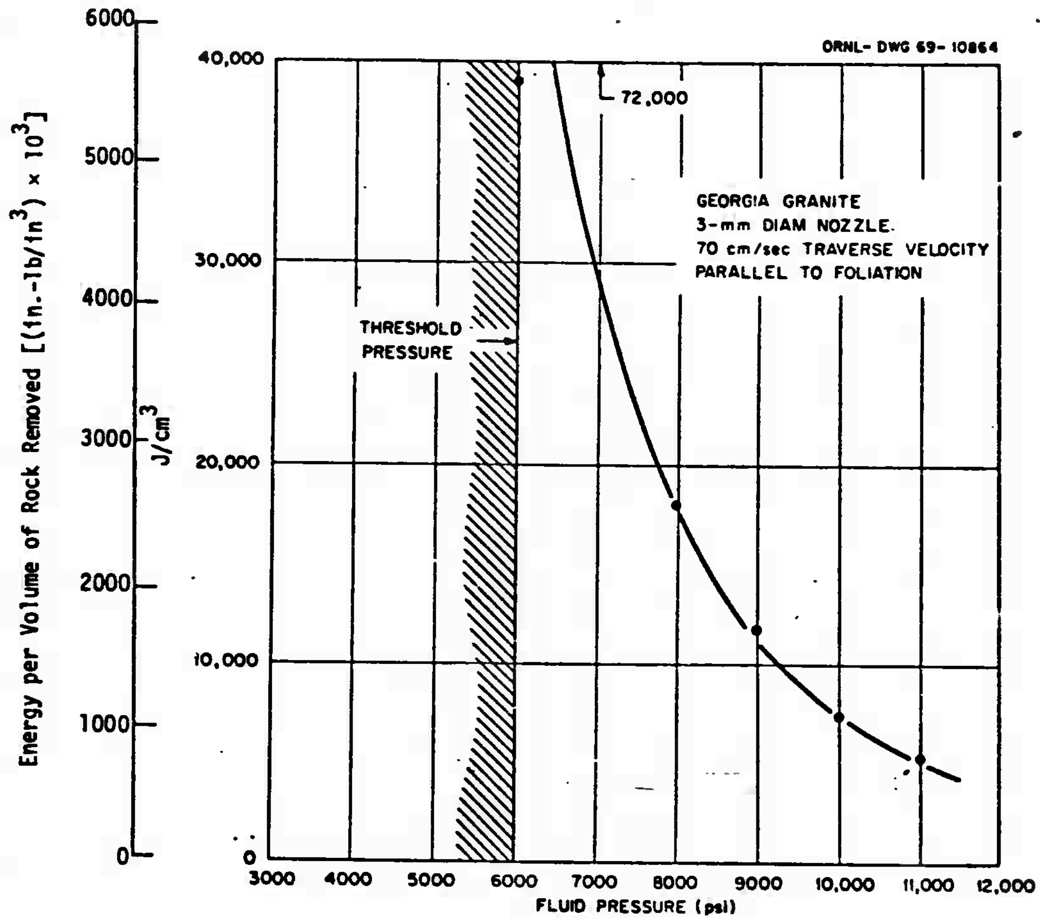
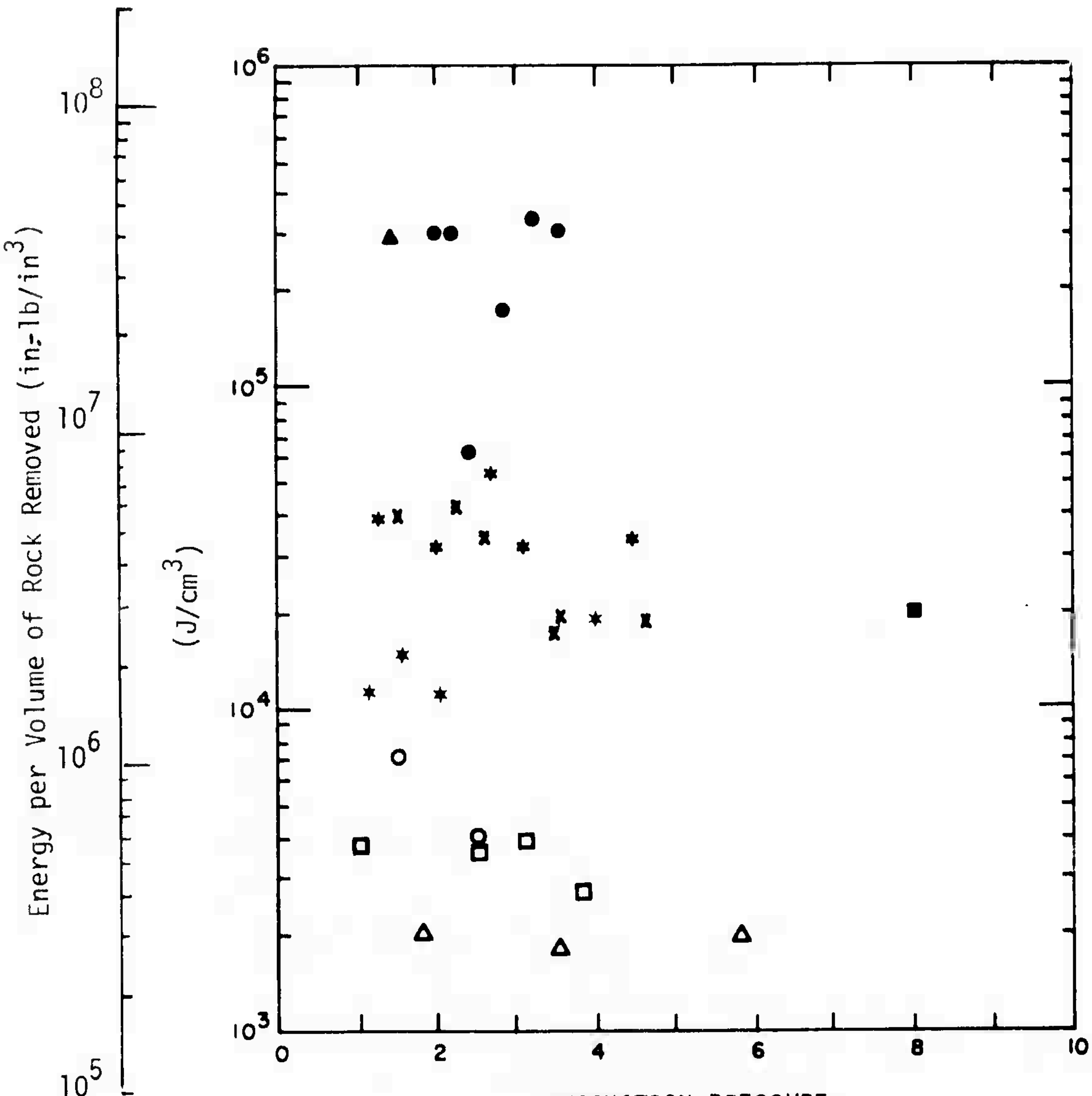


Figure 36. Continuous Water Jet Performance in Georgia Granite[42]



STAGNATION PRESSURE / COMPRESSIVE STRENGTH

DATA OF LEACH AND WALKER[35]

- x - 21,000 psi SANDSTONE
- * - 20,000 psi GRANITE
- * - 16,000 psi MARBLE

DATA OF SINGH AND HUCK[37]

- ▲ - 57,000 psi GABBRO
- ● - 32,000 psi GRANITE
- ■ - 10,500 psi BROWNSTONE

DATA OF CLIPP AND COOLEY[39]

- ○ - 30,000 psi GRANITE
- □ - 20,000 psi LIMESTONE
- △ - 15,000 psi SANDSTONE

Figure 37. Single Pulse Water Jet Test Results

penetration cavities to be cylindrical holes of 5 mm diameter as stated in their paper. Other data are from the researcher's own volume and energy calculations.

Clipp and Cooley[39] report a steady decrease in specific energy with higher pressure rates (Fig. 38). Also shown in this figure is an effect observed by other experimenters as well: multiple shots at the same target, particularly when directed close to an exposed edge or other free surface, significantly reduce the energy requirement to fragment the rock.

The scatter of data indicates that water jet performance depends largely on device design and efficiency. Any estimation of performance should be based on field trials of a particular device. Although an equation of the performance of water jet fragmentation has been developed for the computer simulations based on the available data, the resulting fragmentation rates calculated by this equation provide only a first order estimate and include considerable uncertainty.

The volume of water that is added to the tunnel by the jet device may be calculated by the previous equations. The removal of this water may not be a major problem with the very high pressure jets considered for hard rock excavation because of the low volumes of water added. Low pressure, continuous water jets, which add considerably greater volumes of water, may require some additional water recycling or removal equipment. High humidity and low visibility may be problems with both types of jets.

No value for the high noise level which is present during water jet operation has been found in the literature surveyed, but it has been discussed by researchers and observers of water jet operation as a possible drawback to the use of high-velocity jet devices in a tunnel environment. Similarly, there is no information published of which the authors are aware that identifies the partitioning of the kinetic energy of the

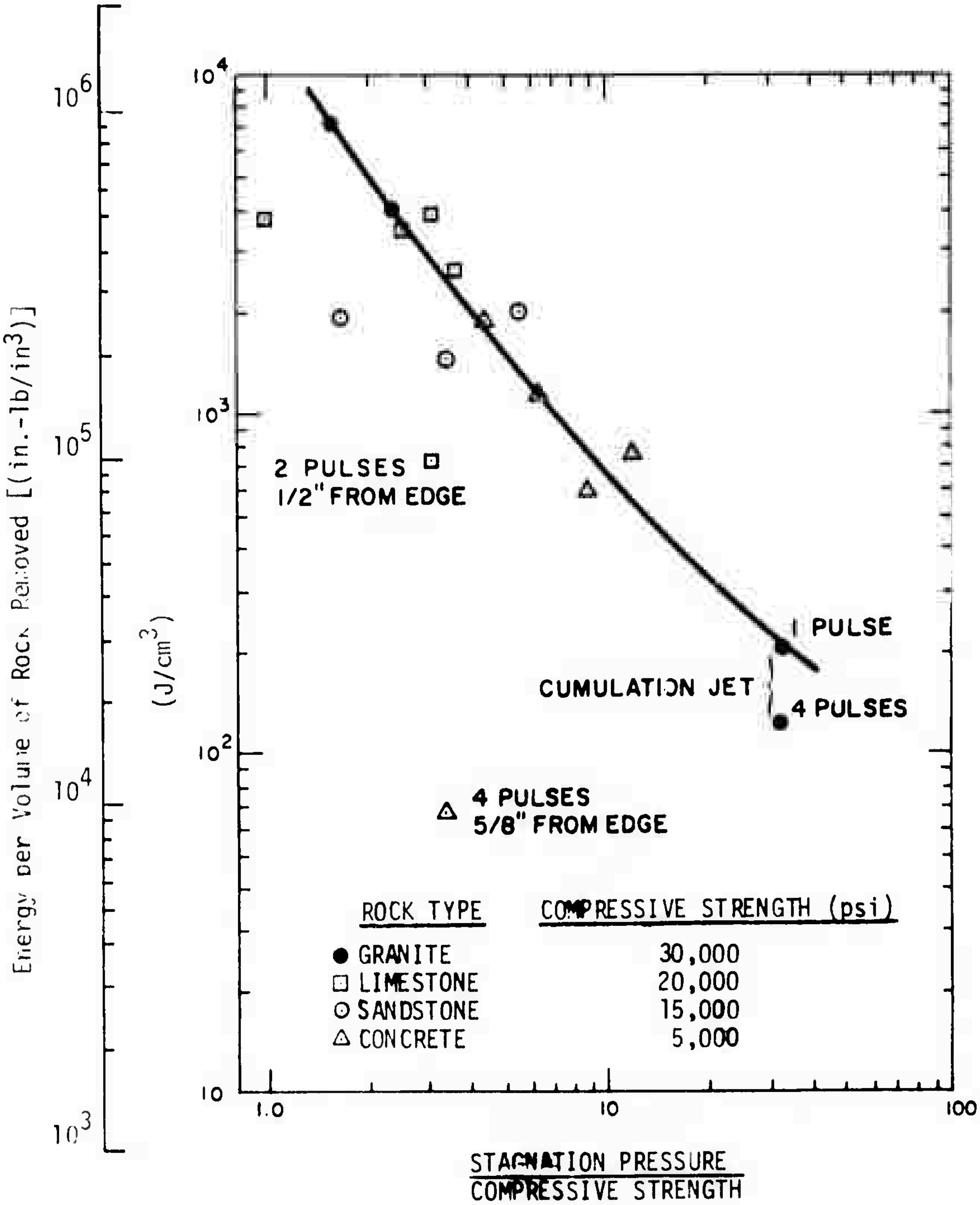


Figure 38. Effect of Jet Pressure on Energy Required to Break Rock[39]

jet when it impacts on the rock. It is unlikely that more than 25% of the jet energy would be transferred to the rock undergoing fragmentation. Analogous considerations of solid pellet impact have shown that this transfer of energy may be as low as 10 to 15%. The waste energy would enter the tunnel environment as kinetic energy of rock fragments which may present some hazard to the workers, and as thermal energy which would impose an added load on the environmental control system.

3. The Model

Water jet activities which have been modeled for the excavation simulation are given by Table 14. Table 15 provides the input parameters and specifications which drive the simulation of water jet performance.

TABLE 14
WATER JET ACTIVITIES

Subroutine JETIMP

Fragmentation of rock by continuous or intermittent water jetting, as specified

Subroutine JETAGN

Repositioning and alignment of the water jet device

Subroutine JETMNT

Maintenance and repair of water jet device

TABLE 15

WATER JET PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

(A) Equipment Information

Equipment Information

1. Number of jets ____________
2. Nozzle diameter (in.) ____________
3. Jet pressure (psi) ____________
4. Jet device efficiency ____________
5. Jet impact efficiency ____________
6. Energy required per rock volume broken (in·lb/in.^{3} × 10^3)* ____________
7. Mode of operation Continuous ( )
   Pulsed ( )
8. If pulsed mode,
   (a) Pulse rate (pulses/min) ____________
   (b) Pulse duration (sec) ____________

(B) Water Jet Cost Information and Input Specifications

1. Plant and Equipment

Item	Ownership (Rental) Cost Per	Hr (check one)	Ft	Unusual Developmental Cost Per	Hr (check one)	Ft
MAJOR ITEMS:						
Water Jet						
ADDITIONAL ITEMS:						

2. Job Materials

Item	Cost Value	Cost Unit	Lifetime Value	Lifetime Unit
MAJOR ITEMS:				
Power: Electric*		\$/kW·hr	-	-
Minor Servicing & Repairs (per maintenance)		\$		
ADDITIONAL ITEMS:				
		\$/ft		

*A representative value is programmed into the model. A user may substitute his own value if he desires.

3. Direct Labor

Labor Type	Number Required/Shift	Rate \$/hr
MAJOR TYPES:		
Machine Operator		
Miner		
Mechanic		
Electrician		
ADDITIONAL TYPES:		

4. Permanent Materials

Item	Cost Value	Cost Unit
(none required)		

4. Subroutine JETIMP

Subroutine JETIMP calculates the heading advance and volume of rock broken during the time the water jet device is in operation. It also calculates the volume of water added to the tunnel from the jetting system and the heat added to the environment.

The rate of penetration, R, of a water jet device can be expressed in general as

$$R = \frac{P}{AE}$$

where P = jet power
A = tunnel cross section area
E = energy per volume of rock broken

Penetration by a multiple jet device is assumed to be approximated by multiplying by the number of jets. Proper spacing of jets would be likely to improve performance over that for a set of jets each having an independent effect on the rock. The degree of possible improvement by multiple, simultaneous, properly spaced water jets has not yet been investigated. By analogy to boring machine cutter spacing, one concludes that significant improvement may be possible.

a. Continuous Jet

The following equations are used to calculate the performance of the continuous jet:

$$P_j = 0.0174\, D_n^{\,2}\, p_n^{\,1.5}$$

where P_j = jet power (hp)

D_n = nozzle diameter (in.)

p_n = jet nozzle pressure (psi)

Total jet power is NP_j where N is the number of jets in operation. The supply power required is

$$P_s = 0.745 \frac{NP_j}{n_j}$$

where P_s = total supply power (kW)

n_j = device efficiency

= 0.3 if not otherwise specified

Therefore the job material cost incurred during time interval Δt(hr) for this power is

$$\text{Cost } (\$) = P_s \, \Delta t \, (C_p)$$

where C_p = power cost (\$/kW-hr)

= \$0.015 if not otherwise specified

It is assumed that all waste energy from the fragmentation process adds heat to the tunnel environment which must be removed by the environmental control system. The amount of heat added during time interval Δt (hr) is:

$$H = (1 - n_j n_i) \, (P_s) \, (0.707) \, (\Delta t) \, (3600)$$

$$= 2545 \, (1 - n_j n_i) \, P_s \, \Delta t$$

where H = thermal energy added (Btu)

n_j = device efficiency

= 0.3 if not otherwise specified

n_i = impact efficiency*

= 0.2 if not otherwise specified

The volume rate of water which is added to the tunnel may be calculated by multiplying jet velocity times its cross section. The jet velocity in feet per second is

$$V_j = \sqrt{2 \times \frac{32.2}{62.4} \times 144 \times p_n}$$

$$= 12.2 \sqrt{p_n}$$

The volume of water (ft^3) added during time Δt (hr) is

$$Q = NV_j \left(\frac{\pi D_n^2}{4} \right) \frac{3600\ \Delta t}{144}$$

$$= 6.25\ \pi N V_j D_n^2\ \Delta t$$

The assumed relationship for the jet energy required per unit volume of rock broken, which approximates the results shown in Figs. 34-36, is shown plotted in Fig. 39. For the continuous jet the equation is

$$E = 47\ e^{.09\sigma_c}$$

*Impact efficiency is the fraction of jet energy which is transmitted into the rock for fragmentation. The remaining fraction of jet energy is expended as thermal waste or fragment energy which becomes thermal waste.

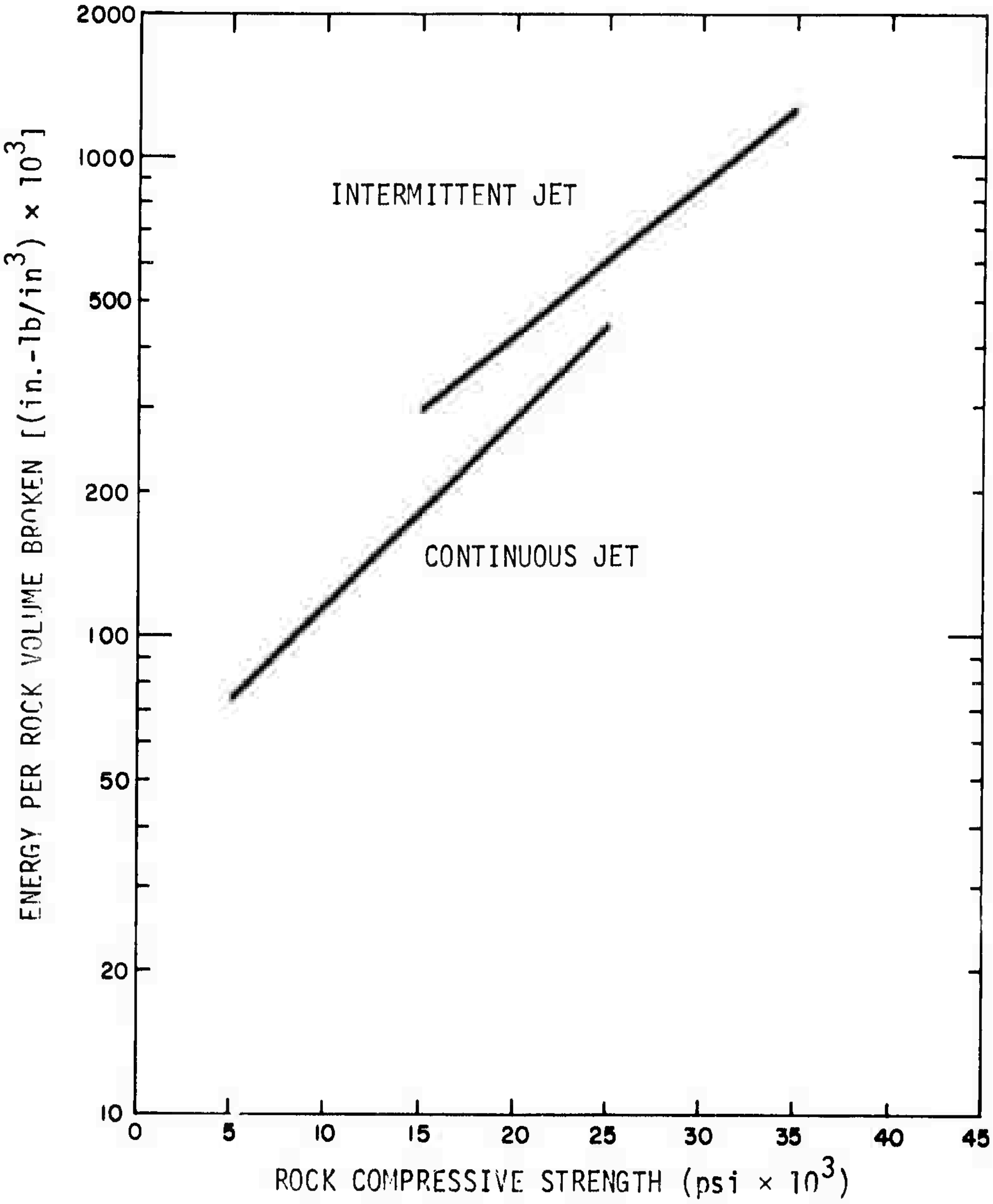


Figure 39. Performance Assumed for Modeling Pulsed and Continuous Water Jet Fragmentation

where E = jet energy per volume of rock broken [(in.-lb/in^3) × 10^3]

σ_c = rock unconfined compressive strength (psi × 10^3)

Use of this equation is permissible for rock compressive strength between 5000 and 25,000 psi. An error message results otherwise.

The advance of the tunnel heading during time Δt (hr) may thus be calculated as

$$\Delta X = 13.75 \frac{NP_j \Delta t}{\pi\left(\frac{D^2}{4}\right)E}$$

where ΔX = tunnel advance (ft)

D = tunnel diameter (ft)

During this time, the following rock volume will be fragmented:

$$\text{Volume} = \Delta X \pi (D^2/4)$$

b. Intermittent Jet

The calculations for a pulsed jet are similar to those for the continuous jet. The following modifications to the above equations are made to account for the intermittent nature of the jet.

The average power of the jet is

$$\overline{P_j} = P_j \text{ (PD)(PPM/60)}$$

where $\overline{P_j}$ = average power of the intermittent jet (hp)

PD = pulse duration (sec)

PPM = jet firing rate (pulses/min)

P_j = jet power of a continuous jet

The value calculated for average power of an intermittent jet then replaces the power value in the continuous jet performance equations to calculate power cost, heat added, and tunnel advance. The water volume calculation is similarly multiplied by the factor

$$PD \times \frac{PPM}{60}$$

to account for the intermittent nature of the jet

The assumed relationship for the jet energy required per unit volume of rock broken is shown in Fig. 39. The relationship is:

$$E = 100\ e^{.072\sigma_c}$$

valid between 15,000 and 35,000 psi.

5. Subroutine JETAGN

Subroutine JETAGN accounts for the time delay for repositioning the water jet system. The system is repositioned whenever the advance of the tunnel face has increased the distance between the face and the water jet beyond some set distance. The results of laboratory tests indicate this standoff distance should always be kept less than 500 nozzle diameters to minimize the dispersion of the jet and reduction of impact pressure. One quarter hour repositioning time is assumed if no value is input.

6. Subroutine JETMNT

Subroutine JETMNT accounts for water jet system maintenance and repair periods. In terms of performance parameters, the user provides either a machine availability or down time per maintenance and the time between maintenance.

The characteristic maintenance requirements for a water jet system are not known at this time and should be treated parametrically in a systematic manner to identify the impact of various maintenance requirements on the performance of any system under consideration.

E. PROJECTILE IMPACT

1. General Discussion

The impact of solid projectiles at velocities of 4000 ft/sec or greater is sufficient to generate stresses of well over 1 million psi to fracture the hardest rock. The projectiles may be launched by existing rapid-fire gun techniques for which the technology is well developed and the size and cost of the guns are within reasonable limits. Existing rapid-fire guns are capable of accelerating projectiles weighing several pounds to velocities over 7000 ft/sec at sustained firing rates of several rounds per minute.

Typical stresses produced by projectile impact can be calculated from the same plane shock relation which was discussed in the previous section for high-pressure pulses of water:

$$p = \frac{\rho_p c_p \rho_t c_t V_p}{\rho_p c_p + \rho_t c_t}$$

where p = impact pressure

ρ_p = projectile density

c_p = shock propagation velocity in projectile

ρ_t = target (rock) density

c_t = shock propagation velocity in target (rock)

V_p = projectile velocity

The shock propagation velocity will depend on shock pressure. An initial estimate may be made assuming a shock velocity equal to the acoustic velocity which is the lower bound of propagation velocities for very weak shocks.

Inserting typical numbers,

ρ_p = 137 lb/ft^3

c_p = 11,980 ft/sec

ρ_t = 165 lb/ft^3

c_t = 19,685 ft/sec

V_p = 5000 ft/sec

and the impact pressure is found to be:

p = 1.2 million psi

If the projectile length is equal to its diameter, and if the projectile material is similar to that of the rock, the transfer of energy into the rock is enhanced.

A comprehensive treatment of various theories of impact craters formed by high-velocity impact may be found in Refs. 43, 45, and 46 to which the reader is referred for further information. A summary of experimental results is now presented to provide the basis for modeling.

The data published by Moore et al.[43] is shown in Fig. 40, in which the weight of ejected rock is plotted as a function of projectile energy. The curve which best fits their data is supported by recent experiments at Physics International,[44] which show that the ejected mass is proportional to the projectile energy raised to the power 1.189. While this is a small variation from simple geometric scaling (ejected mass proportional to projectile energy), it has important ramifications when the mass of the projectile is increased from a few grams to several pounds. According to the Griffith theory of failure, this scaling effect is caused by the increased number of cracks and faults in the rock sample as the size of the region affected by the impact increases.

The equation fitted to the published data of Moore et al. provides the basis for calculating the required energy per volume of rock broken, which is used to model projectile impact performance. For basalt, (which has the same density as granite):

$$W_E = 1.63 \times 10^{-5} \left[\left(\frac{\rho_p}{\rho_t} \right)^{1/2} E_p \right]^{1.189}$$

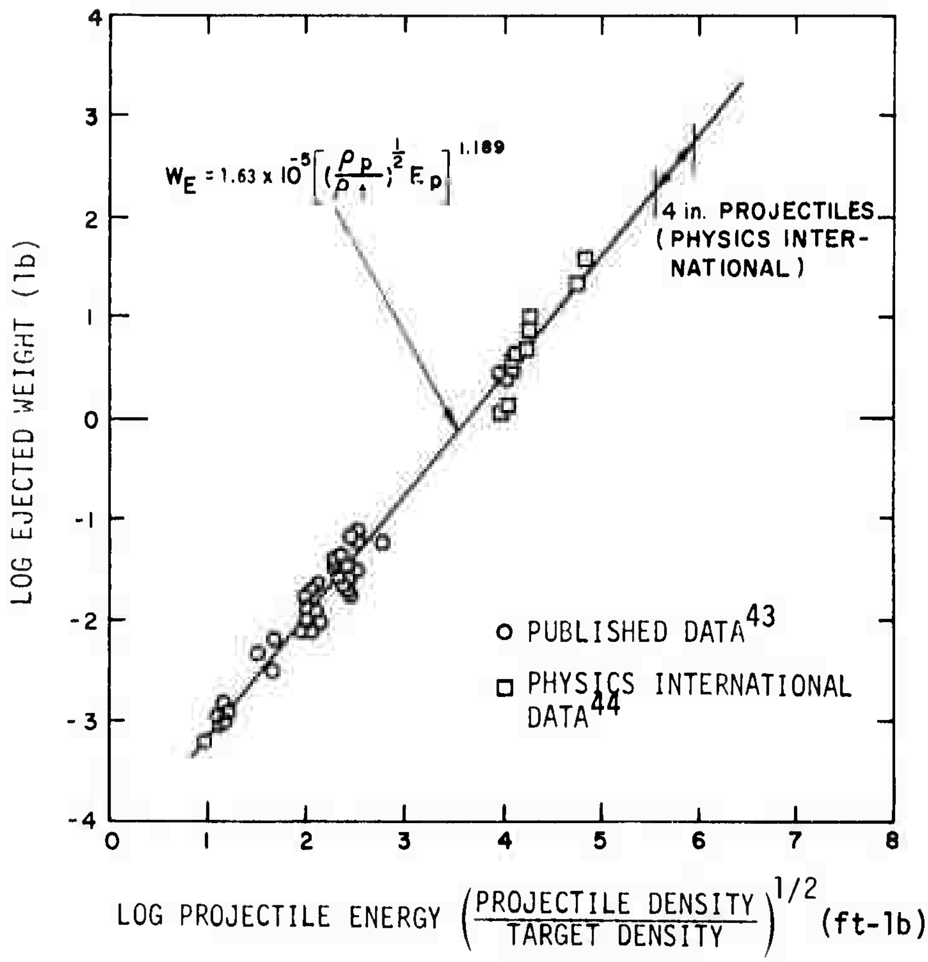


Figure 40. Experimental Relationship for Projectile Impact. (Solid line represents least squares fit to published data.[43])

where W_E = ejected rock weight (lb)

ρ_p = projectile density

ρ_t = target (rock) density

E_p = projectile energy (ft-lb)

This relation can be rewritten in terms of energy per volume of rock fragmented:

$$E = 6.14 \times 10^4 \rho_t \left(\frac{\rho_t}{\rho_p}\right)^{0.595} E_p^{-.189}$$

where E = energy required per rock broken (ft-lb/ft^3)

ρ_t = target (rock) weight density (lb/ft^3)

E_p = projectile energy (ft-lb)

For comparison with water jet impact, the energy required per volume in units of [(in-lb/in^3) × 10^3] is

$$E = 0.426 \rho_t \left(\frac{\rho_t}{\rho_p}\right)^{0.595} E_p^{-.189}$$

where E = energy required per rock broken [(in-lb/in^3) × 10^3) and the other variables have the units given above.

The projectile energy, E_p, can be calculated from its velocity, V_p, and its weight, W_p:

$$E_p = \frac{1}{2}\left(\frac{W_p}{32.2}\right) V_p^2$$

Inserting the typical numbers used previously, assuming a 4 lb projectile,

$$E_p = 1.55 \times 10^6 \text{ ft-lb}$$

$$W_E = 334 \text{ lb}$$

and the energy per rock volume broken is:

$$E = 5.3 \times 10^3 \text{ in.-lb/in}^3$$

This value compares favorably with boring machine performance (Fig. 22), percussive drilling (Fig. 19), and drill and blast.

Figure 41 presents representative energy curves calculated by Physics International for various projectiles and projectile velocities.

Results of Physics International's initial experiments using a gun with a bore diameter of 1 1/2 in., with methane-oxygen propellant,* are consistent with the published work of others. The energy efficiency of the granite impacts was about 13×10^3 in.-lb/in^3. It should be noted that 3 1/2 times the projectile energy was required as chemical energy in the propellant, or an efficiency of gun operation of about 0.29. Further waste of energy occurred upon impact when perhaps only 10 to 24% of the projectile kinetic energy may have been expended for fragmentation, the remainder according to Gault and Heitowit[45] being expended as waste heat and fragment kinetic energy (Fig. 42). Thus, for this method of

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*Rapid-fire gun techniques would use other means of projectile propulsion.

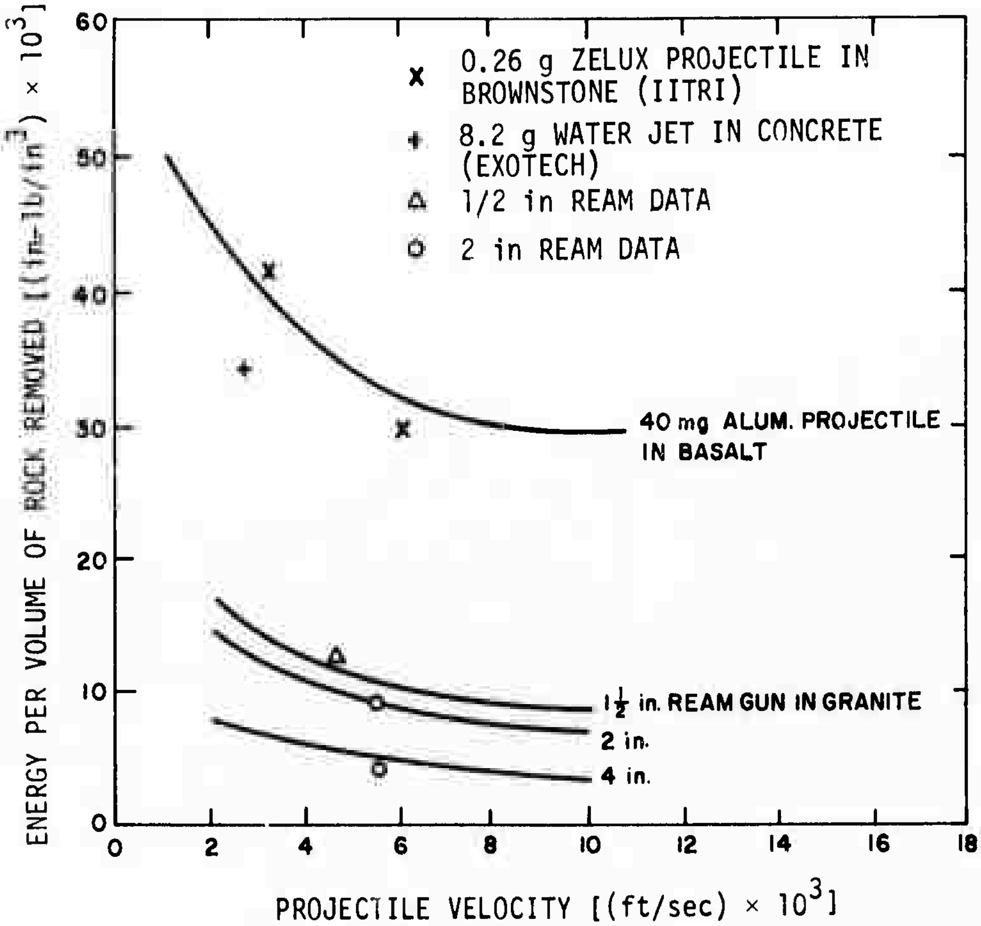

Figure 41. Projectile Performance (Sample Data provided by Physics International)[44]

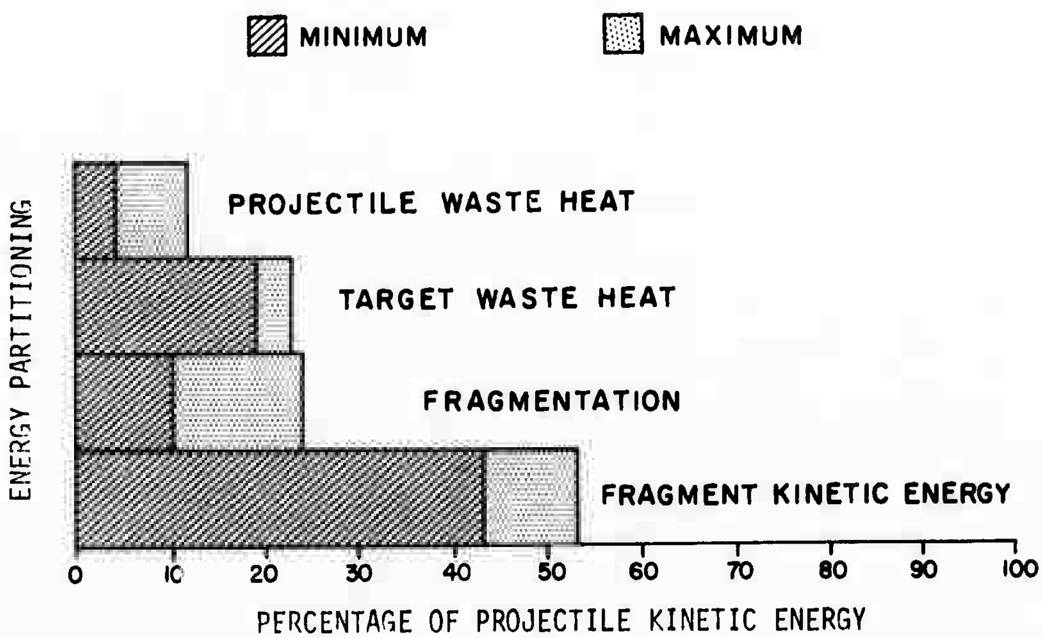

Figure 42. Projectile Energy Partitioning.[45]
Aluminum projectile into basalt

pellet firing and impact, for every 100 units of chemical power supplied as propellent, 93 to 97 units of waste heat may have to be removed by the combination of gun coolant and environmental control.

Table 16 lists the set of activities which have been modeled to account for projectile impact fragmentation in the system simulation.

TABLE 16

PROJECTILE IMPACT ACTIVITIES

<u>Subroutine PROJTL</u>

Rock fragmentation by projectile impact

<u>Subroutine PRJBR</u>

Barrel replacement of gun

<u>Subroutine PRJAGN</u>

Projectile gun repositioning and alignment

<u>Subroutine PRJMNT</u>

Projectile system repair and maintenance, exclusive of barrel replacement

Table 17 lists input specifications and parameters which are required for the projectile impact subroutines. Some estimated characteristics of a projectile system provided by Physics International are listed in Table 18 as a guide to some of the input values.

Some of the advantages of a projectile system are:

1. It is adaptable to different tunnel sizes or cross sections.
2. It can effectively break hard rock at potentially high rates.

TABLE 17

PROJECTILE PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

Equipment Information

1. Number of guns________________________________

2. Weight of projectile (lb)______________________

3. Volume of projectile (in^3)___________________

4. Velocity of projectile (ft/sec)________________

5. Firing rate, each gun (projectiles/min)________

6. Gun efficiency________________________________

7. Impact efficiency_____________________________

8. Time to replace one gun barrel (hr)____________

9. Maintenance parameters:
   (a) Availability of projectile system (%)_____
   (b) In lieu of (a), average down time per maintenance period (hr)________________
   (c) Average time between maintenance periods (hr)______________________________

TABLE 17 (cont.)

PROJECTILE PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

1. Plant & Equipment

Item	Ownership (Rental) Cost Per	Hr (check one)	Ft	Unusual Developmental Cost Per	Hr (check one)	Ft
MAJOR ITEMS:						
Projectile Cannon						
ADDITIONAL ITEMS:						

2. Job Materials

Item	Cost		Lifetime	
	Value	Unit	Value	Unit
MAJOR ITEMS:				
Propulsion		$/Projectile	-	-
Projectile		$/Projectile	-	-
Gun Barrel		$/Barrel	-	-
Minor Servicing & Repairs (per maintenance)		$		
ADDITIONAL ITEMS:				
		$/ft		

3. Direct Labor

Labor Type	Number Required/Shift	Rate $/hr
MAJOR TYPES:		
Gun Operator		
Gun Loader		
Electrician		
Mechanic		
Miner		
ADDITIONAL TYPES:		

4. Permanent Materials

Item	Cost	
	Value	Unit
(none required)		

TABLE 18

REAM SYSTEM CHARACTERISTICS

Dual cannon with integrated mucking system	$620,000
Firing rate	up to 30 shots/min
Projectile weight	4 lb

Crew Size*

Diameter 10 ft	Diameter 20 ft
1 gun operator	2 gun operators
1 gun loader	2 gun loaders
1 electrician	1 electrician
1 mechanic	1 mechanic
1 miner	2 miners

Projectile Cost	$.50/projectile
Barrel Replacement Cost	$.13/projectile

*Estimates of crew required for rock fragmentation alone taken from estimates of total crew size provided by Physics International.

3. It can draw from existing technology for development of rapid fire guns.

4. It produces less disturbance of the rock than conventional drill-and-blast methods.

5. There is less labor required than for conventional drill and blast.

6. The distance between the projectile gun and the tunnel face can vary up to 150 ft with little loss of efficiency.

7. Projectiles can be removed along with the muck.

The disadvantages are:

1. Flying rock and projectiles
2. Noise, heat, and dust
3. Toxic fumes from propellant
4. Difficult muck removal while firing
5. Cost of projectiles
6. Logistics of projectile and propellant supply
7. Unsuitable in soft ground and possibly also in mixed ground
8. Interference with ground support activities

2. Subroutine PROJTL

Subroutine PROJTL calculates the advance of the tunnel, volume of rock broken, heat added to the tunnel environment, power cost, and projectile cost during the fragmentation operation. The sequence of calculations is similar to that for the firing of an intermittant water jet with the energy relationship of the projectile substituted for that of the water jet. The added job material cost for the expenditure of projectiles is

$$\text{Projectile cost} = N_g(\text{PPM})C_p\ \Delta t(60)$$

where N_g = number of guns

PPM = firing rate (projectiles/min)

Δt = time interval (hr)

C_p = cost per projectile (\$/projectile)

3. Subroutine PRJBR

Subroutine PRJBR accounts for the time and cost of barrel replacement. If a barrel requires replacement after firing 10,000 projectiles at a rate of 30/min, replacement would occur every 5.5 operating hours. The user estimates barrel lifetime, barrel replacement time, and barrel cost.

4. Subroutine PRJAGN

Subroutine PRJAGN accounts for repositioning of the projectile gun system. The time delay for repositioning is assumed to be 1/4 hr unless otherwise specified. The system is repositioned whenever the advance of the tunnel face has increased the distance between the face and the gun beyond some user set distance. The gun can operate effectively at up to 150 ft from the face but shorter distances would be desirable to minimize projectile hazard and allow ground support installation.

5. Subroutine PRJMNT

Subroutine PRJMNT accounts for projectile system maintenance and repair periods exclusive of barrel replacement. In terms of performance parameters, the user provides either a projectile system availability or down time per maintenance and the time between maintenance.

The characteristic maintenance requirements for a projectile system are not known at this time and should be treated parametrically in a systematic manner to identify the impact that various maintenance requirements would have.

IV. MATERIALS HANDLING

A. INTRODUCTION AND GENERAL DISCUSSION

1. Present Scope of Model

As an element of the excavation process, materials handling includes both transport of broken rock or soil (muck) away from the tunnel face as well as men and construction materials to and from the face.

The general processes considered for modeling under materials handling are given in Table 19.* They have been grouped by function according to those used to transport muck from the excavation face to the main line, and those used as the main line (or long-haul system) to transport muck along the tunnel to a discharge point at a shaft or portal. The processes are also categorized according to whether material is transported continuously or intermittently in individual unitized modules.

Continuous general processes include mechanical conveyors, hydraulic pipe, and pneumatic pipe. All these are similar in that an independent medium (i.e., a belt of fluid) normally moving through a closed loop is used to transport the muck continuously from a loading point at or near the face of a tunnel to a discharge point at a main line or a portal or shaft. These continuous systems are generally limited to muck transport only; therefore, a supplemental intermittent type system is generally needed for transporting construction materials and personnel.

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*In this study, consideration was focused on systems best suited for horizontal transport along a tunnel. Vertical systems for shafts are not included although some of those given in Table 19 can be used for both.

TABLE 19

MATERIALS HANDLING PROCESSES

FACE TO MAIN-LINE TRANSPORT

1. Continuous
   a. Integrated-conveyor loaders
   b. Hydraulic pipe
2. Intermittent
   a. Loaders
   b. Shovels
   c. Shuttle cars
   d. Scoop-trams (load-haul-dump equipment)

MAIN-LINE TRANSPORT

1. Continuous
   a. Mechanical conveyors
   b. Hydraulic pipe
   c. Pneumatic pipe
2. Intermittent
   a. Conventional rail system
   b. Truck system
   c. Monorail, side-rail system, etc.

Intermittent general processes for face to main-line transport include loaders, shovels, shuttle cars and scoop-trams. For long haul transport they include conventional rail, truck, monorail, and side-rail systems. The distinguishing characteristic of these compared to continuous flow is the separation of materials into discrete quantities which are carried by mobile units, either individually or in interconnected trains.

In view of and consistent with the time constraints of this study the following processes were selected from Table 19 for modeling at this time.

1. For face to main-line transport
    a. Integrated-conveyor loader
    b. Intermittent loaders and shovels
2. For main-line transport
    a. Continuous mechanical conveyors
    b. Conventional rail systems
    c. Truck systems

A detailed description of what is included in these processes covering also the activities and performance and cost characteristics that have been derived for them is given in Secs. IV-B through IV-E.

2. <u>Processes Not Presently Modeled</u>

The face to main-line systems given in Table 19 but not presently modeled include shuttle cars and scoop trams. By their very nature, these systems would probably find greatest application in construction related to tributary tunnels that branch from a main tunnel (i.e., a situation where there might be considerable distance separating the tributary face and the main line system). Future versions of the model to account for these possibilities could easily adapt the presently existing truck model (see Sec. IV-C) to shuttle cars and scoop-trams without major changes.

Novel main-line rail systems such as the monorail and side-rail (or variations thereof) are similar in that each consists of modules that run on rails or guideways. The monorail utilizes a single steel beam mounted above the load-carrying vehicle, whereas the side-rail system utilizes two rails at the sides of the load carrying vehicle. The module may be driven as a unit or coupled into trains. Both systems normally run above the tunnel floor, limiting their capacity and increasing their equipment cost because of structural support considerations. Support also makes the extension of these systems during rapid excavation operations difficult and time consuming. At this time both concepts are still in the developing stage and the available performance and cost information is therefore based on somewhat limited experience. Nonetheless studies in the past have shown that as their capacity increases these systems seem to improve in cost-effectiveness relative to other systems.[47] Moreover, in small tunnels the monorail offers the possibility of freeing the tunnel floor for other uses.

The authors feel that these systems could be easily included in the model at a future data by adapting the existing conventional rail haul model to reflect their individual performance and cost characteristics.

Hydraulic pipe systems transport muck using water pumped through pipelines at a velocity sufficient to propel the muck (normally crushed to form a slurry with the water) along the pipe. This system has had limited application to tunneling, particularly in hard rock, but it has been used as a relatively low-cost method for moving large quantities of bulk materials in the mining and dredging industries where continuous operation over long periods is possible.

The application of hydraulic systems to tunneling is presently limited by inherent difficulties in extending the system without complete shutdown, and by rapid wear of the transport pipe when the transported rock is high abrasive. The necessity of having small rock particles (<0.125 in.) to form a slurry normally requires secondary crushing equipment ahead of the mixing and charging units.[47] Therefore, extending the system in the near-face zone would tend to be rather complex. Crushers and mixing tanks must be advanced as well as pipe segments, and all this tends to disrupt flow. Also, the use of large quantities of high-pressure water in a tunnel may be undesirable because of the hazard of flooding.

The pneumatic pipe system exhibits even greater inherent difficulties related to hard rock tunneling. In this system the transporting medium is air drawn or blown through a pipe at velocities sufficient to propel the material along. Although system extension tends to be less complex, the material that can be transported is presently limited to dry, low-density, small-size materials (<3 in.). Also capacities and haul distances presently tend to be low, whereas capital equipment and operating costs are relatively high compared to other systems.[47]

Inclusion of hydraulic and pneumatic pipe systems in the model at a future date should present no major difficulties. With the exception of the performance and cost parameters and relationships, the basic logistical framework for modeling these systems would be very similar to the existing conveyor model for continuous transport--see Sec. IV-D. In this respect the transport subroutines would require substantial changes in performance equations (but not logistics) and also an additional subroutine to account for secondary crushing would be required. The maintenance and system extension subroutines, with the exception of the use of different performance parameters, would be very nearly identical.

3. Materials Handling Model - Interactions and Limitations

This section reviews the major interactions, summarized in Fig. 6, that may exist between the materials handling system and other elements of the excavation process including tunnel geometry and geology. It also discusses the extent to which these interactions have been included in the model at this time.

Tunnel Geometry: Tunnel geometry includes cross section, length, alignment, and grade. All affect the required design of the materials handling system to sustain a desired materials handling rate. For example, tunnel cross section establishes the amounts of materials (muck and construction) that must be handled for a given face advance rate; length impacts on system capacity in terms of cycle times; alignment and grade affect system capacity in that curves and slopes cause trains to slow down, conveyor equipment to wear out at transfer points, etc. In addition, not only does tunnel cross section impact on materials handling rate, but it also interacts by excluding or limiting the performance of certain systems due to minimum operating space requirements (e.g., truck systems are normally excluded from tunnels less than 14 ft in diameter).[23]

Presently, only tunnel cross section and length and the impact they have on muck tonnage rates are modeled. Operating area requirements and the affects of variable grades and alignment are not (i.e., a nominally horizontal, straight segment of tunnel is presumed).

Geology: Geology affects materials handling directly through in situ rock properties such as density and abrasiveness. Geology (particularly such characteristics as rock quality, degree of fracture or general structure), together with the rock fragmentation process, also impacts on materials handling through the size distribution, angularity, and swell of broken rock. Geology (i.e., rock quality, density), together with ground support, also impacts in establishing the amounts and type of support required at the face. At this time only density variations and the swell factor (taken as a constant) are included in the model.

Rock Fragmentation: Materials handling may limit the overall utilization of the rock fragmentation system. This may happen because the materials handling capacity is inadequate and cannot keep up with fragmentation. (For example, in a conventional rail system all trains may be in transit and none available for loading at the face.) The materials handling system may also require frequent periods for maintenance and repair, and the time required for system extension may not be compatible with the excavation rate. All these interactions could cause the excavation system to be shut down at various intervals, and all have been built into the model at this time. Reciprocally, rock fragmentation impacts on materials handling utilization by establishing the rates at which material must be moved to and from the face to sustain the face advance, and also through the fragmentation system's characteristic breakdown and maintenance requirements. As discussed before, rock fragmentation also interacts indirectly, together with the geology, to establish muck size distributions, angularity, swell factors, etc., which can affect the capacity of the materials handling system. Only the interaction of excavation system advance rate, availability and swell factor (modeled as a constant) on materials handling utilization and capacity has been modeled at present.

Ground Support: Materials handling interacts with ground support installation in two important ways: the system must provide an adequate supply of support materials in the near-face zone, and it must do so without interferring with the ground support equipment (i.e., geometric constraints limit the amount of equipment in the near-face zone at any given time). Both these considerations have not been modeled at this time. It is therefore presumed that support material supplies are always adequate, and the equipment required from each element never exclude each other.

Environmental Control: The materials handling system impacts on the environmental control system in that it might expel waste heat, gases, and fumes (.e.g, diesel exhaust) and dust. It also impacts through the ventilation requirements demanded by the labor force required for operation. Dilution requirements for diesel exhaust gases and ventilation for labor has been included in the model. Heat and dust interactions have not been.

As in the case of ground support, the environmental control system also interacts on materials handling through equipment size constraints in the near-face zone, as well as material supply requirements (pipes, etc.) for extension. Neither of these have been modeled at this time.

B. CONVENTIONAL RAIL SYSTEM

1. Introduction

Rail systems, used for long-haul transport in tunnels, consists of trains containing individual car units mounted on wheels that ride on tracks similar to a commercial railroad.

These systems presently use locomotive, cable, or side-wheel drive for propulsion. Locomotive drive is most common, but is limited to grades of 4% maximum because it depends on friction between steel wheels and steel tracks for traction.

Locomotives are presently manufactured with a variety of power sources that include: diesel, diesel-electric, electric, and battery. For tunnel mucking operations, diesel power is the most commonly used. Electric systems find greatest application as adjuncts to continuous muck systems for transporting construction materials and personnel. Diesel-electric and battery-powered systems are not widely used, diesel electric because they are generally produced only in large sizes, and battery because they are relatively slow and have low draw-bar capability. In this study only diesel and electric-powered locomotives were considered for modeling.

Cable-drive and side-wheel drive systems have evolved in recent years in response to the grade limitations of the locomotive systems. The cable drive system, similar to that in San Francisco, is normally used only for inclined tunnel segments and requires a conventional locomotive for horizontal haulage. The side-wheel drive systems propel trains of cars using electrically powered, rotating rubber-tire drive units mounted along side the track and bearing on the car sides. This technique is relatively new but appears to offer considerable potential because of its ability to climb steep grades and maintain high-capacity rates. At present neither the cable drive or side-wheel drive systems have been modeled directly. The emphasis here was on systems used primarily for horizontal transport. The authors feel, though, that future versions of the model could easily adapt the locomotive model discussed subsequently for these systems if it proves desirable.

The conventional rail system as modeled, then, consists of diesel or electric-powered locomotives pulling trains on tracks similar to a commercial railroad. A large amount of information is available on these systems and therefore no problems were encountered in modeling it.

Conventional rail systems, though, do have some inherent limitations particularly related to their use in rapid hard-rock tunneling at depth. These include slow and inaccurate track-laying methods, and roadbed and alignment problems that can result in comparatively slow haulage speeds (20 mph maximum even under good conditions) and time delays for track laying. Also they are manually controlled, which leads to potential scheduling problems, particularly for small tunnels that may require single tracks and intermittent switches and several trains operating concurrently in different directions to sustain high face-advance rates. Also rapid loading operations at the face for these systems requires a relatively large percentage of tunnel cross section for equipment (see loading equipment for the Layout Tunnel, Volume II) that contributes to congestion and can affect the utilization of other elements of the system.

As mentioned earlier, locomotive-drive systems are also limited to grades of 4% or less, which means that other modes of transport are required for inclined tunnel segments and shafts. In the case of deep tunnels, transfer equipment and discharging operations at the shaft may lead to considerable congestion and time delays.

The activities associated with the conventional rail system, including subroutine names used in the model, are summarized in Table 20. Also, the input specification sheet including performance and cost information related to the trains and tracks are given in Table 21. Subsequent sections discuss these activities and inputs in depth.

TABLE 20

SUMMARY OF CONVENTIONAL RAIL SYSTEM ACTIVITIES

Subroutine RAILHL

Updates train dynamics accounting for accelerations, decelerations and switching logistics

Subroutine RAILEX

System extension - track laying, train and switch addition

Subroutine ADDTRN

Train insertion from the discharge and maintenance area for single track systems

Subroutine RAILDS

Unloads muck at discharge point

Subroutine RAILMT

Train maintenance and repair

TABLE 21

RAIL SYSTEM PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

TRAIN INFORMATION

1. Maximum number of trains________
2. Number of trains
   (a) Initially________
   (b) Per new mile of system________
3. Number of muck cars/train________
4. Capacity of muck car: Volume (yd^3)________
   Weight (tons)[a]________
5. Weight of empty muck car (tons)________
6. Number of axles/muck car________
7. Weight of locomotive (tons)________
8. Number of axles on locomotive________
9. Drive train efficiency of locomotive[a]________
10. Traction coefficient[a]________
11. Locomotive power source (diesel or electric)________
12. Rated continuous operation horsepower________
13. Peak horsepower available________
    or
    Traction effort characteristics

TE (lb)	Speed (mph)

14. Maximum allowed speed (mph)________
15. Maximum allowed acceleration (mph^2)________

[a]If not input, values have been provided for in the model.

16. In lieu of items 5-15
    (a) Peak speed empty (mph)________
    full (mph)________
    (b) Peak acceleration empty (mph^2)________
    full (mph^2)________
    (c) Horsepower required at peak speed Empty________
    Full________
    (d) Horsepower required at peak acceleration
    Empty________
    Full________
17. Average braking deceleration empty (mph^2)________
    full (mph^2)________
18. Fuel consumption rate (for diesel loc.) (gal/hp·hr)[a]________
19. Maintenance parameters
    (a) Single train availability________
    (b) In lieu of (19-a) down time per maintenance period (hr)________
    (c) Time between maintenance period (days)________
    (d) Number of trains allowed in maintenance at one time________
20. Muck loading - number of trains allowed in loading area at one time________
21. Muck unloading________
    (a) Maximum number of trains in discharge area________
    (b) Time to unload one train (hr)________
    (c) Maximum allowed speed of train entering & leaving discharge area (mph)________
    (d) Distance from portal to discharge area (ft)________

TRACK INFORMATION

22. Single or double track________
23. Length of switch (ft)________
24. Distance between switches (ft)________
25. Time to assemble one switch (hr)________
26. Rate California switch can be moved along main line (ft/hr)________
27. Allowed train speed in switch (mph)________
28. Rate at which track can be extended (ft/hr)________
29. For continuous excavation processes maximum excavator runaway tolerance (ft)________
30. Required loading area ahead of switch
    (a) Maximum permitted (ft)________
    (b) Minimum permitted (ft)________

TABLE 21 (cont.)

RAIL SYSTEM PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

1. Plant & Equipment

Item	Ownership (Rental Cost Per	Hr	Ft (check one)	Unusual Developmental Cost per	Hr	Ft (check one)
MAJOR ITEMS:						
Locomotive unit						
Muck car unit						
Track materials						
Switch						
Unloading equipment						
Maintenance shop						
ADDITIONAL ITEMS:						

2. Job Materials

Item	COST		LIFETIME	
	Value	Unit	Value	Unit
MAJOR ITEMS:				
Power: Diesel*		$/gal		
Electric*		$/kW·hr		
Minor servicing & Repairs (per maintenance)		$		
ADDITIONAL ITEMS:				
		$/ft		

3. Direct Labor

Labor Type	Number Required/Shift	Rate $/hr
MAJOR TYPES:		
Motormen		
Brakemen		
Dispatchers		
Dumpmen		
Bull gang		
Foreman		
Laborer		
Maintenance		
Foreman		
Mechanics		
Electricians		
ADDITIONAL TYPES:		

4. Permanent Materials

Item	Cost Value	Unit
(None Required)		

2. Subroutine RAILHL

a. Function

Subroutine RAILHL simulates train dynamics accounting for accelerations, decelerations, and, in the case of single tracks, switching logistics. In addition to the system performance and cost parameters, inputs to the subroutine include: the time increment, present length of available track, present number of trains and switches in the system, the location of each switch and state of each train (i.e., empty or full, accelerating, deceleration, constant velocity, in a swtich, in the loading queue, etc.). The outputs include: an updated train state vector accounting for motions and state changes that occurred during the time interval; an updated load queue vector containing trains available for loading; and finally job material, plant and equipment, and labor costs accrued during the time interval.

Figure 43 presents the rail haul geometries provided for in the model. As shown, this includes both double- and single-track systems. For the double-track case, a typical sequence followed by an empty train, entering at the shaft or portal at the right of the figure, is initially to accelerate to some maximum allowed speed (provided by the uses), then to continue along at this speed until it approaches the land area queue. At this time it decelerates and then either stops in the load area queue (if the load area is filled) or proceeds directly into the loading area where it then is loaded (see Sec. IV-E). Once loaded the train follows a similar sequence (i.e., acceleration, constant speed, and deceleration) back to the discharge area.

In the case of a single track, the sequence between switches is also the same but differs in that a train slowing down to enter a switch must make a decision to stop or proceed through. In actual tunneling operations a block signaling system would be used to alert the train operator, by l... ts or some other signal, regarding the condition of

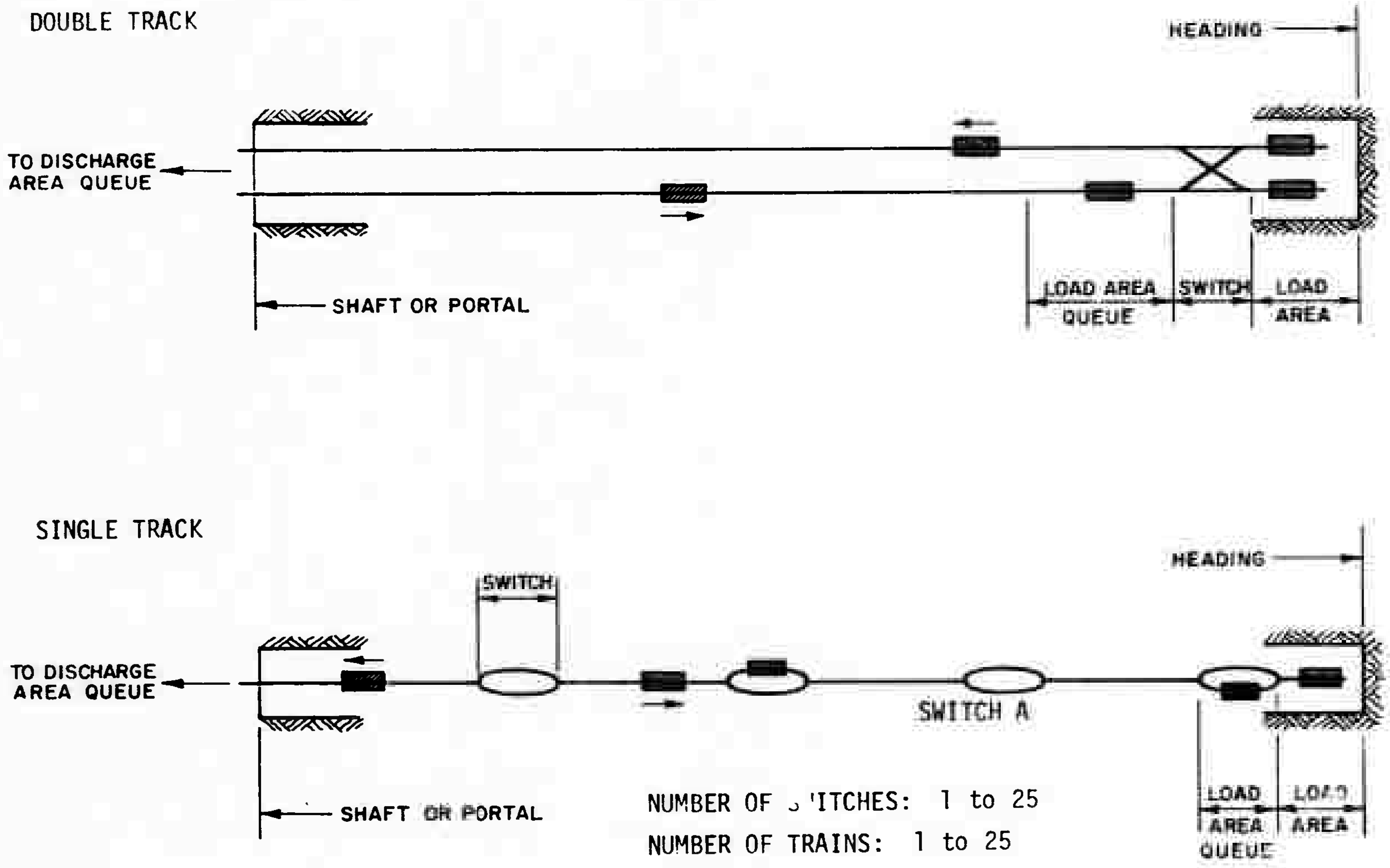

Figure 43. Rail Haul Geometries

the track ahead. In the model, the decision to stop or proceed is made by examining the next segment of track, including the switch directly ahead. The switching algorithm logistics is built to allow empty trains to proceed to the face as quickly as possible. Consequently, where a decision between an empty or full train stopping in a given switch must be made, the empty one will have preference and be allowed to continue.

Another interesting feature of the switching algorithm is that it models time-phased shutdowns of individual trains when rock fragmentation or some other subsystems cause stoppages. For example, suppose excavation has stopped with the load area queue filled with trains waiting to be loaded. In this case all other trains in motion will continue only until they must stop. For instance, consider an empty train approaching the first switch before the load area (identified as switch A in Fig. 43). This train will recognize that the load queue is full and stop in switch A. Now because switch A is also filled subsequent empty trains following will stop in switches preceding switch A until eventually the discharge area is shut down. This sequence works in the opposite way as well--if something should stop dumping operations in the discharge area, full trains will pile up in switches until eventually excavation will be forced to shut down.

3. Performance Model

The user of the train model must first establish those basic characteristics of the train system listed in Table 21. This includes the maximum number of trains required, the number of muck cars per train, and the capacity of each car. The best values for these parameters is not always obvious for the case to be modeled. For a given tunnel geometry and geology, ideally one would like the program itself to establish the best system configuration. At this time, the program is not built to do this, but it can be used parametrically, adjusting these parameters over a series of runs and noting their effect on system performance.

In this respect, the user may find the following equations useful for roughly establishing a higher bound on the number of trains required, given a car capacity and train size. These equations assume that a train is always available at the face for loading, that the geology and heading advance rate is uniform, and that the accelerations and decelerations are equal and peak speeds are the same for full and empty trains.

$$N_t = \frac{A_T \dot{R} S_f T_\ell}{V_c N_c} \tag{1}$$

where Double-Track Case:

$$T_\ell = \frac{T_D + \dfrac{2L}{5280 v_m} + \dfrac{2v_m}{a_m}}{N_t - 1} \tag{2}$$

Single-Track Case:

$$T_\ell = \frac{T_c}{N_t} \tag{3}$$

N_t = maximum number of trains required

N_c = number of cars per train

V_c = volume capacity of car (ft^3)

A_T = area of tunnel (ft^2)

L = maximum distance between face and discharge space (ft)

S_f = muck swell factor

v_m = maximum train speed (mph)

a_m = maximum train acceleration (mph^2)

$\dot{R}$ = estimated tunnel advance rate (ft/hr)

T_ℓ = time to load one train distance between face and discharge space (hr)

T_D = time to unload one train (hr)

T_c = cycle time for one train (hr)

In the above equations a muck car capacity is required. Muck cars are manufactured with a variety of capacities, sizes, and features. Capacities range from 1/2 yd^3 (1 ton) to the large 100-ton cars. Widths range from 2.5 ft to as high as 11 ft. The tendency to retain a low profile seems to be general with the exception of the 1 ton to 5 ton cars where height is considerably greater than width. Cars and their dumping arrangements include the rock dump, end dump, and the automatic roller dump. Figure 44 and Table 22, taken from Refs. 47 and 23, respectively, may help the user establish approximate car sizes and capacities to use in the model and in Eqs. 1-3.

The function of subroutine RAILHL is to calculate train motions. The logic of the program presumes that a given train can be in one of twelve different states depending on its location and the status of operations at the time of interest. These states are:

1. Stopped and empty or loading
2. Full and stopped
3. Empty and accelerating
4. Full and accelerating
5. Empty and at peak speed
6. Full and at peak speed
7. Empty and decelerating

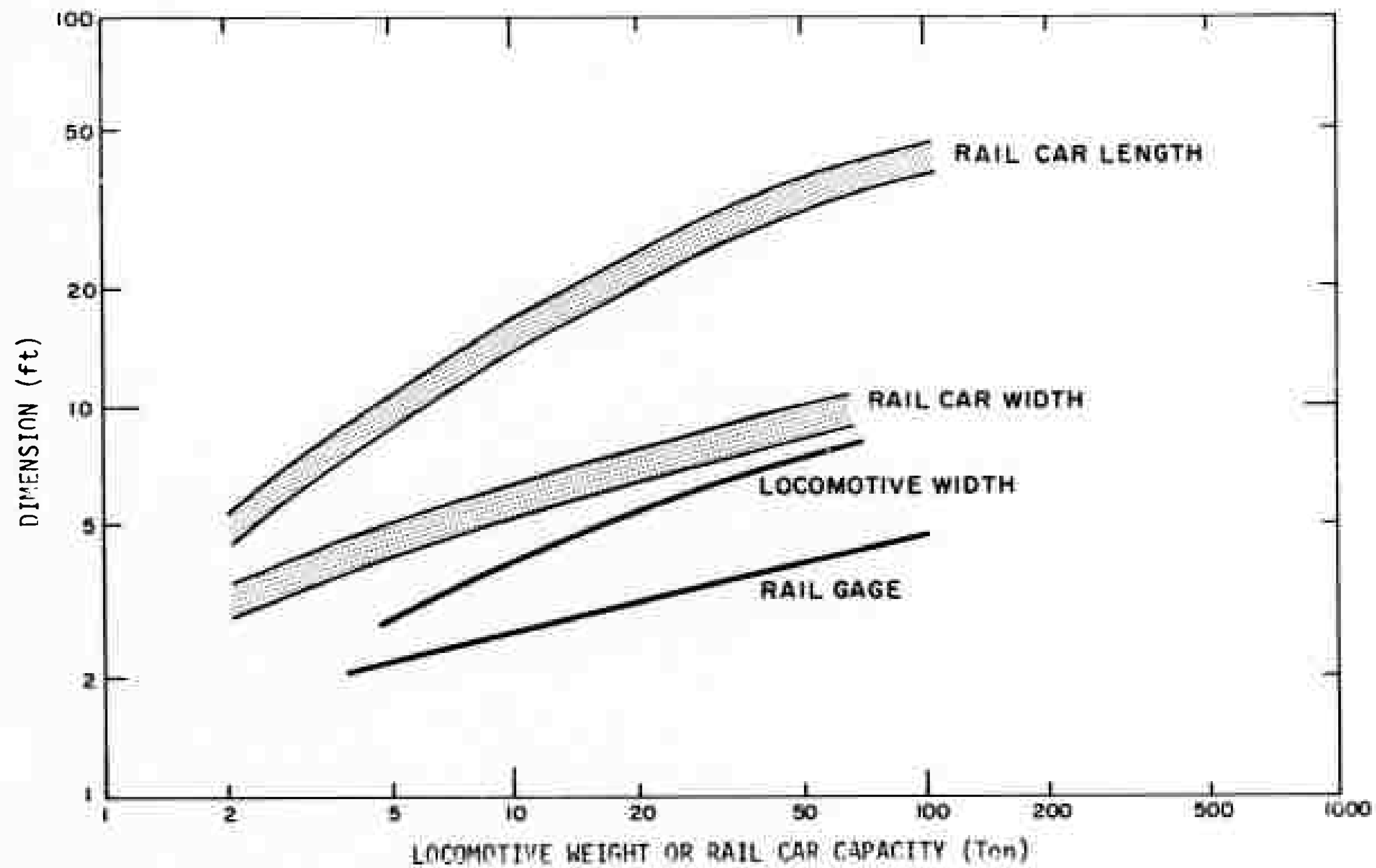

Figure 44. Conventional Rail Equipment Dimension/Weight Relationships[47]

TABLE 22

DIMENSION OF RAIL MUCK CARS
(SIDE-DUMP CARS MANUFACTURED BY RAY MORAN)[23]

	Capacity of car										
	8 cu yd	9 cu yd	9 cu yd	9 cu yd	9 cu yd	10 cu yd	10 cu yd	10 cu yd	10 cu yd	10 cu yd	12 cu yd
Track gauge	36 in.	36 in.	36 in.	36 in.	36 in.	36 in.	36 in.	36 in.	36 in.	36 in.	42 in.
C_L cplg above top of rail	16 in.	16 in.	16 in.	16 in.	16 in.	16 in.	16 in.	16 in.	16 in.	16 in.	16 in.
Length C_L to C_L cplg	15 ft 1¼ in.	16 ft 1¼ in.	11 ft 10¾ in.	14 ft 4¼ in.	13 ft 11¼ in.	16 ft 2¼ in.	15 ft 7¼ in.	15 ft 1¼ in.	14 ft 7¾ in.	14 ft 3¼ in.	15 ft 8 in.
Wheel base	6 ft 0 in.	6 ft 6 in.	5 ft 11 in.	5 ft 7 in.	5 ft 5 in.	6 ft 6 in.	6 ft 3 in.	6 ft 0 in.	5 ft 9 in.	5 ft 7 in.	6 ft 6 in.
Length inside	12 ft 0 in.	13 ft 0 in.	11 ft 9 in.	11 ft 3 in.	10 ft 10 in.	13 ft 1 in.	12 ft 6 in.	12 ft 0 in.	11 ft 6 in.	11 ft 2 in.	12 ft 11 in.
Depth inside	3 ft 9 in.	3 ft 0 in.	3 ft 9 in.	3 ft 9 in.	3 ft 9 in.	3 ft 9 in.	3 ft 9 in.	3 ft 9 in.	3 ft 9 in.	3 ft 9 in.	4 ft 0 in.
Overall width	4 ft 9 in.	5 ft 0 in.	5 ft 6 in.	5 ft 9 in.	6 ft 0 in.	5 ft 6 in.	5 ft 9 in.	6 ft 0 in.	6 ft 3 in.	6 ft 6 in.	6 ft 6 in.
Overall height	6 ft 6 in.	6 ft 6 in.	6 ft 6 in.	6 ft 6 in.	6 ft 6 in.	6 ft 7 in.	6 ft 6 in.	6 ft 6 in.	6 ft 6 in.	6 ft 6 in.	5 ft 9 in.

8. Full and decelerating
9. Empty and at switch speed
10. Full and at switch speed
11. In the discharge area
12. In the maintenance area

For accelerating trains the dynamics are updated using the following relationships which assume that accelerations over the time interval remain constant. (Note that accelerations may vary between intervals.) For empty trains

$$X_2 = X_1 + \frac{5280}{2} a_e \Delta t^2 + 5280 v_1 \Delta t \tag{4}$$

$$v_2 = v_1 + a_e \Delta t \tag{5}$$

For full trains

$$X_2 = X_1 - \frac{5280}{2} a_f \Delta t^2 - 5280 v_1 \Delta t \tag{6}$$

$$v_2 = v_1 \pm a_f \Delta t \tag{7}$$

where Δt = time interval (hr)
X_2 = position at end of interval (ft)
X_1 = position at beginning of interval (ft)
v_2 = velocity at end of interval (mph)
v_1 = velocity at beginning of interval (mph)
a_e = acceleration for interval - train empty (mph^2)
a_f = acceleration for interval - train full (mph^2)

a_e and a_f are determined for a given time interval in one of two ways. The use can select a purely kinematic solution where accelerations always remain constant and are direct user inputs (Table 21, item 16); or he can select a kinetics solution and provide power and other related characteristics of the train (Table 21, items 5-15) that are then used to calculate accelerations (subject to a maximum limit, a_m) also a user input--Table 21, item 15). The equations that follow apply to the latter. The empty and full accelerations (a_e and a_f) are related to the net accelerating force (F_e and F_f) and the gross weight of the train by the equations

$$a_e = 39.4\ F_e/G_e \leq a_m \qquad (\mathrm{mph}^2) \tag{8}$$

$$a_f = 39.4\ F_f/G_f \leq a_m \qquad (\mathrm{mph}^2) \tag{9}$$

where the 39.4 is a conversion constant and

$$F_e = T_e - R_e \qquad (\mathrm{lb}) \tag{10}$$

$$F_f = T_f - R_f \qquad (\mathrm{lb}) \tag{11}$$

$$G_e = W_c N_c + W_\ell \qquad (\mathrm{tons}) \tag{12}$$

$$G_f = N_c \left(W_c + 1.35\ V_c \right) + W_\ell \qquad (\mathrm{tons}) \tag{13}$$

where N_c = number of cars in a train

W_c = empty weight of a car (tons)

V_c = volume capacity of a car (yd^3)

W_ℓ = weight of the locomotive (tons)

T_e = available tractive effort - empty train (lb)

T_f = available tractive effort - full train (lb)

R_e = rolling resistance - empty train (lb)

R_f = rolling resistance - full train (lb)

Tractive effort (T_e and T_f) depends on the available traction at the wheels or on the power delivered to the wheels for a given locomotive. It is computed as the smaller value given by the equations

$$T_e = T_f = \begin{cases} 2000\ f_t \cdot W_\ell & \text{(traction limit)} \\ \dfrac{375\ HP_a \cdot E_\ell}{v_1} & \text{(power limit)} \end{cases} \quad \text{(lb)} \qquad (14)$$

where f_t = coefficient of traction, taken as 0.25 unless input

HP_a = user-input peak available horsepower of locomotive

E_ℓ = drive train efficiency of locomotive, taken as 0.8 unless input

Instead of the above power equation, the user also has the option of providing a tractive effort curve for the locomotive, an example of which is given in Fig. 45 for a Plymouth Model MDT 30-ton Diesel locomotive. Note that this curve implies a certain peak horsepower and drive train efficiency and that the above power equation is simply an approximate fit to this curve.

The rolling resistance (R_e and R_f) required in Eqs. 10 and 11 is determined from[48]

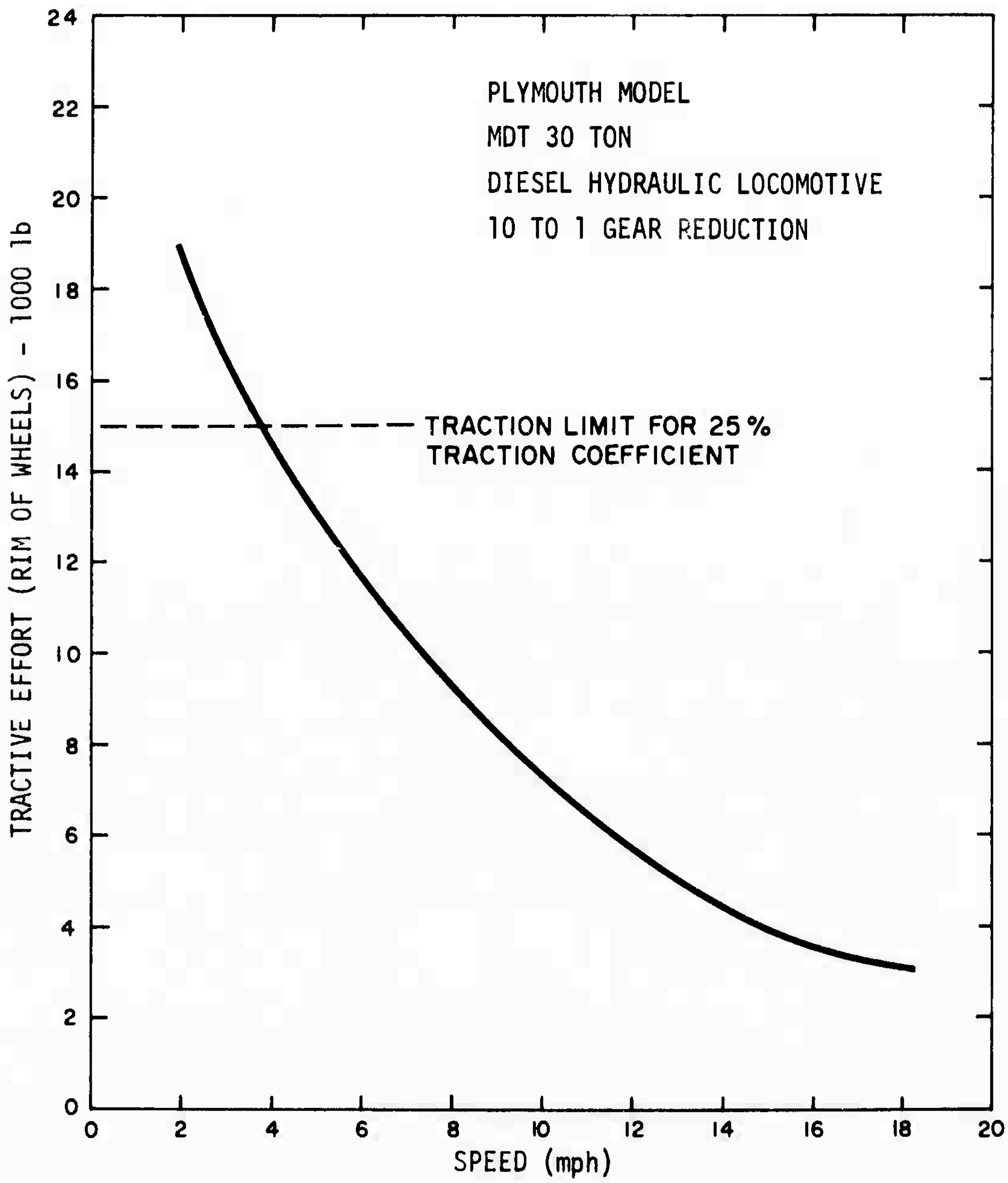

Figure 45. Example Locomotive Tractive Effort Curve (Courtesy of Plymouth Locomotive Works, Plymouth, Ohio)

$$R_e = G_e \left| 1.3 + 29/W_e + 0.0005\ A_c v_1^2/W_e M_c + 0.045\ v_1 \right| \quad (15)$$

$$R_f = G_f \left| 1.3 + 29/W_f + 0.0005\ A_c v_1^2/W_f M_c + 0.045\ v_1 \right| \quad (16)$$

where W_e and W_f are the average axle weight in the empty and full condition and are given by:

$$W_e = G_e/(N_c M_c + M_\ell) \qquad \text{(tons)} \qquad (17)$$

$$W_f = G_f/(N_c M_c + M_\ell) \qquad \text{(tons)} \qquad (18)$$

where, M_c = number of axles per muck car
M_ℓ = number of axles on the locomotive
N_c = number of muck cars
G_e = see Eq. 12
G_f = see Eq. 13

A_c, the side board area of a muck car, can be approximated by[47]

$$A_c = 6.65\ V_c^{0.35} \qquad (\text{ft}^2) \qquad (19)$$

In the case of trains traveling at peak speed, the dynamics are updated using

$$X_2 = X_1 + 5280\ v_e \Delta t \qquad (20)$$

for empty trains and

$$X_2 = X_1 - 5280\ v_f \Delta t \qquad (21)$$

for full trains, where X_2 and X_1 are given above and

v_e = peak empty speed (mph)

v_f = peak full speed (mph)

Again the user has the option of specifying v_e and v_f directly (Table 21, item 16a) or he can select a kinetics solution based on locomotive power (also specifying a maximum allowed speed, V_m--Table 21, item 14--if the power of the locomotive is such that this speed might be exceeded).* For this option the following two equations are solved simultaneously with Eq. 15 for R_e (i.e., with $v_1 = v_e$) and Eq. 16 for R_f (i.e., with $v_1 = v_f$) to obtain v_e and v_f:

$$v_e = 375\ E_\ell HP_v/R_e \leq V_m \tag{22}$$

$$v_f = 375\ E_\ell HP_v/R_f \leq V_m \tag{23}$$

---

*Because of generally poor lighting and roadbed conditions, train speeds in tunnels rarely exceed 20 mph. In some states maximum speeds are even specified by law. An example is provided by Sec. 84-29 of the Colorado Mining Laws:

Employees operating any haulage train underground must keep their trains under control at all times, and operate them at a speed not to exceed the following:

> When the track gauge is eighteen (18) in. and the rails used are twenty (20) lb, the speed shall be three (3) mi/hr on curves and six (6) mi/hr on straightaway.
>
> When the track gauge is twenty-four (24) in. and the rails used are thirty (30) lb, the speed shall be five (5) mi/hr on curves and ten (10) mi/hr on straightaway.
>
> When the track gauge is thirty-six (36) in. and the rails used are forty-five (45) lb, the speed shall be eight (8) mi/hr on curves and fifteen (15) mi/hr on straightaway.
>
> When the track gauge is forty-eight (48) in. and the rails used are sixty (60) lb or over, the speed shall be ten (10) mi/hr on curves and eighteen (18) mi/hr on straightaway.

where E_ℓ, R_e, and R_f are defined above and HP_v = rated power of the locomotive for cor^inuous operation (hp).

The equations used for decelerating trains again assume constant decelerations for the time interval. For empty trains:

$$X_2 = X_1 + 5280\ v_1 \Delta t - \frac{5280}{2}\ d_e \Delta t^2 \tag{24}$$

$$v_2 = v_1 - d_e \Delta t \tag{25}$$

and for full trains:

$$X_2 = X_1 - 5280\ v_1 \Delta t + \frac{5280}{2}\ d_f \Delta t^2 \tag{26}$$

$$v_2 = v_1 - d_f \Delta t \tag{27}$$

where X_2, X_1, v_2, and v_1 are defined above and

d_e = user input average deceleration rate for empty trains (mph^2)

d_f = user input average deceleration rate for full trains (mph^2)

Finally, for trains traveling through switches at a given user-specified speed v_s, the following equations are used

$$X_2 = X_1 + 5280\ v_s \Delta t \qquad \text{(empty trains)} \tag{28}$$

$$X_2 = X_1 - 5280\ v_s \Delta t \qquad \text{(full trains)} \tag{29}$$

4. Cost Information

Cost categories applying to subroutine RAILHL include direct labor, plant and equipment and job materials.

Typical labor types required to run the train system are listed in Table 21 under cost information. Those associated specifically with haulage include motormen, brakemen and dispatchers. It can be presumed that at least one motorman and brakeman will be required for each locomotive, and that at least one dispatcher will be required for every 2 mi of system.[47] Representative wage rates for the western United States are given in Volume II.

For plant and equipment cost, representative costs for locomotive and muck car units (taken from Ref. 47) are given in Fig. 46. The data was established in 1967; therefore, the 1.26 factor is applied to escalate to 1972 costs. One can depreciate these costs over the length of the project by selecting the per foot option of Table 21, or one can select the per hour option taking the life of the equipment conservatively at 15 yr. The latter option is probably more realistic.

The only job material costs computed in subroutine RAILHL are haulage power costs. The model assumes that energy consumed during periods of deceleration or periods spent in switches is negligible compared to accelerating or peak velocity consumption. The model does take into account, though, the difference between accelerating and peak velocity consumption for both full and empty trains.

If the user selects the kinematic option, item 16 in Table 21, the cost for power consumed in the time increment Δt is computed from

$$\text{Power Cost} = HP \cdot F_c \cdot C_p \cdot \Delta t \quad (\$) \tag{30}$$

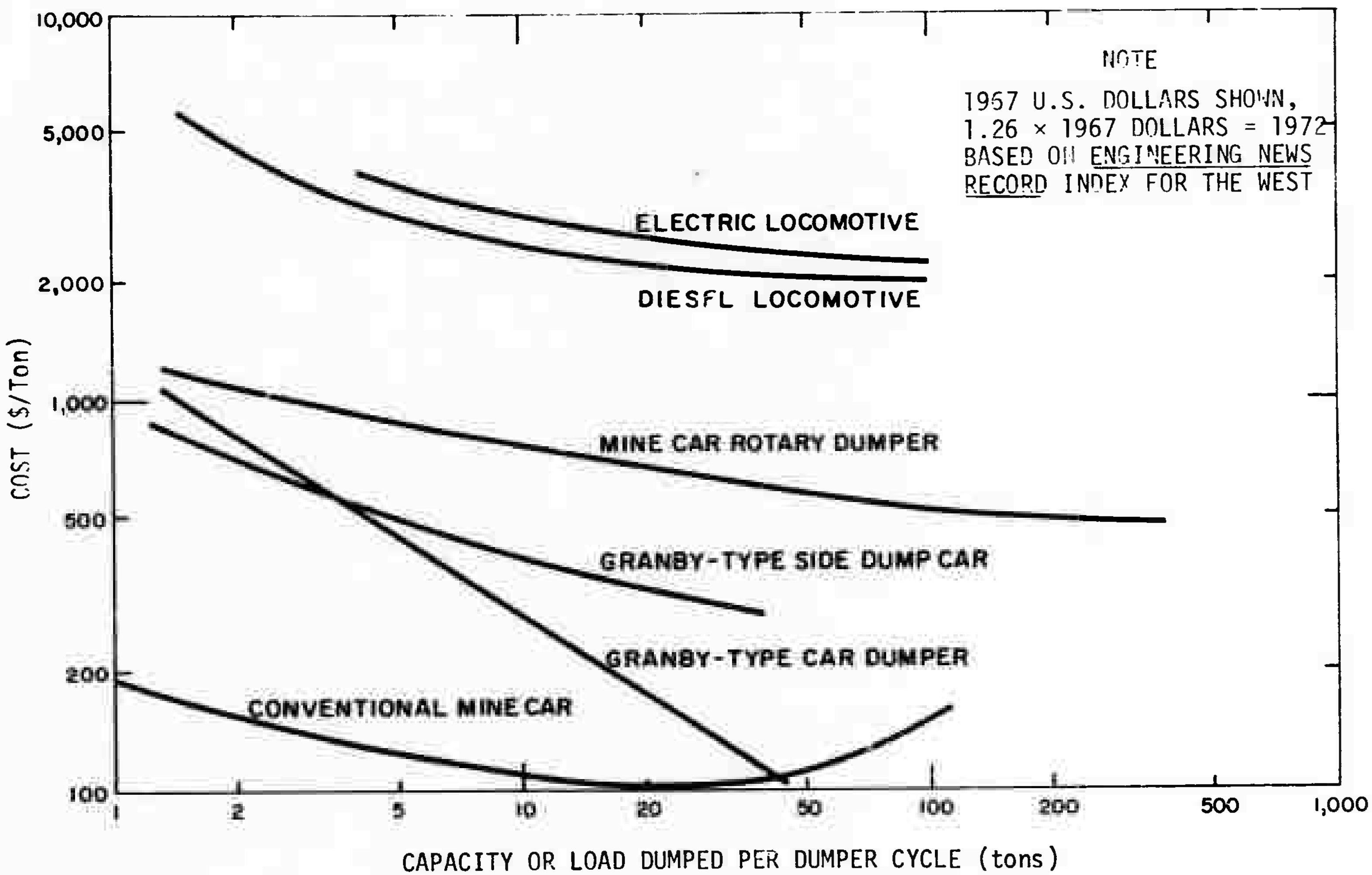

Figure 46. Locomotive System Equipment Cost [47]

where HP = user provided power levels for peak speed - empty and full - and peak acceleration - empty and full (hp)

F_c = diesel fuel consumption data taken as 0.0455 unless input (gal/hp·hr) (i.e., set to one for electric locomotives)

C_p = power cost taken as 0.14 \$/gal for diesel locomotives 0.02 \$/kW·hr for electric locomotives unless input

If the user selects the kinetics option, items 5-15 in Table 21, cost for power in the time interval again is computed from Eq. 30 with the exception that for empty accelerating trains,

$$HP = \left[\frac{a_e G_e}{39.4} + R_e\right] \cdot \frac{v_1}{375\ E_\ell} \leq HP_a \qquad \text{(hp)} \qquad (31)$$

for full accelerating trains,

$$HP = \left[\frac{a_f G_f}{39.4} + R_f\right] \cdot \frac{v_1}{375\ E_\ell} \leq HP_a \qquad \text{(hp)} \qquad (32)$$

for empty trains at peak speed,

$$HP = \frac{R_e v_e}{375\ E_\ell} \leq HP_v \qquad \text{(hp)} \qquad (33)$$

for full trains at peak speed,

$$HP = \frac{R_f v_f}{375\ E_\ell} \leq HP_v \qquad \text{(hp)} \qquad (34)$$

where the various parameters are as defined in the preceding section.

3. Subroutine RAILEX

a. Function

Subroutine RAILEX extends the rail system (including tracks, trains, and switches).

Besides system performance and cost parameters, inputs to the program include the time increment; position of the face; the face advance during last time step; the present number of trains in the system including their location and status; and the present number of switches, including their location. For a given time interval outputs include updated track length, number of trains and number of switches, as well as plant and equipment and direct labor costs.

Typical extension geometries that can be modeled are shown in Fig. 47. Illustrated is the case of a California switch, used for train switching during loading operations, with a single track laid forward of it. Other cases including double tracks and such techniques as a Jacobs sliding floor (where track is laid at the rear) could also be modeled with an appropriate selection of input parameters.

As shown in Fig. 47, the program extends the tracks in a manner consistent with the nature of the excavation scheme employed. The lower figure shows a case where excavation is cyclic, such as drill and blast. Here track is laid in cycles with its advance equal to the heading advance for the round. For excavation processes that are more continuous, the boring machine for example, track is laid continuously with time as shown in the upper figure.

In either case, when the loading area, l_a, exceeds a user input maximum value, operations cease and the California switch is brought forward to a minimum l_a that allows enough room for a full train or single car as the user desires. If the California switch is pulled by

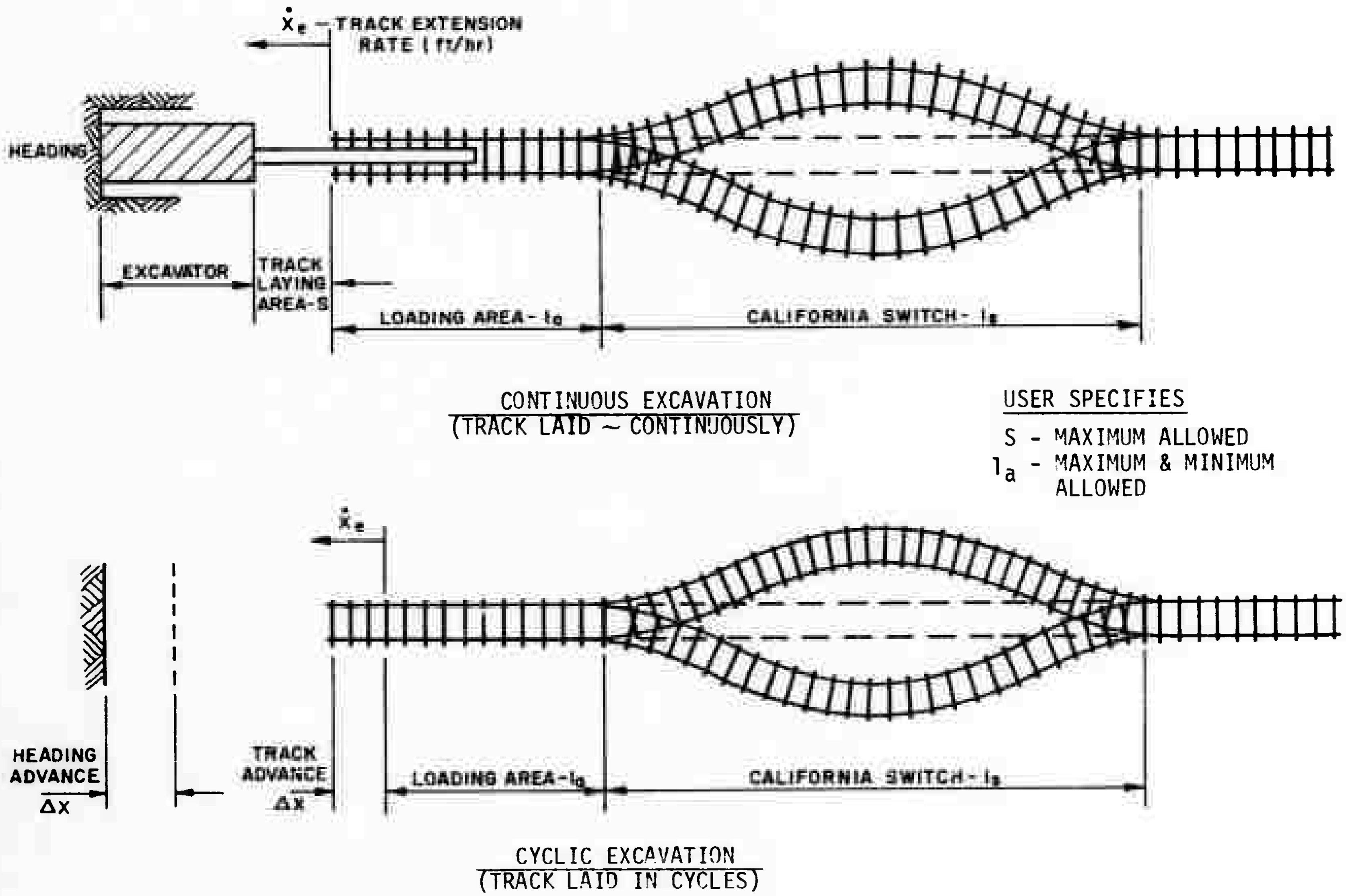

Figure 47. Typical Rail Extension Geometries

the continuous excavator (as is the case for the Layout Tunnel boring machine and loading configuration discussed in Vol. II) l_a maximum and minimum are both set to the same constant loading length.

In the case of continuous excavation, an interesting feature of RAILEX is that it models the interaction between the excavation and track-laying operations. For example, if the excavator advance rate exceeds the track extension rate, over a period of time the excavator will run away from the available track. To prevent this the user must input a runaway tolerance (i.e., a maximum tolerated value of S in Fig. 47) which if exceeded will cause excavation to shut down until track laying can catch up.* One can see how the utilization of the excavation system (Fig. 14) would be adversely affected if this should occur frequently. Indeed this would indicate that quicker track-laying methods must be developed.

b. Performance Relationships

The major parameters defining system extension characteristics in subroutine RAILEX include the number of trains required for each mile of system, the distance between switches for a single track case, and the rate at which track can be extended.

The algorithm for train addition is set up as follows. The user provides the number of trains per mile of system ($\overline{N}_t$) for one or two ranges of tunnel length. The following equations are then used to define when trains are to be added:

---

*If the excavator rate is less than the track extension rate such that over a period of time S (Fig. 47) should go to zero, the track extension rate would simple be set equal to the excavator rate until such time as S becomes greater than zero.

for $X \leq 5280L^1$

$$\left(N_t\right)_1 = \left[\left(N_t\right)_0 + \left(\bar{N}_t\right)_1 \left(\frac{X}{5280}\right)\right] \leq N_t \tag{35}$$

for $X > 5280L^1$

$$\left(N_t\right)_1 = \left[\left(N_t\right)_0 + \left(\bar{N}_t\right)_1 \left(L^1\right) + \left(\bar{N}_t\right)_2 \left(\frac{X}{5280}\right)\right] \leq N_t \tag{36}$$

where X = face position for time interval (ft)

N_t = maximum number of trains

$(N_t)_1$ = updated number of trains

$(N_t)_0$ = initial number of trains at $X = 0$

L^1 = tunnel length where train addition criteria changes (mi)

$(\bar{N}_t)_1$ = number of new trains desired per mile of system for $X \leq 5280L^1$

$(\bar{N}_t)_2$ = number of new trains desired per mile of system for $X > 5280L^1$

The program performs the above computations in integer arithmetic (i.e., all fractional quantities are truncated, with $(N_t)_1$ containing the lowest whole number). Consequently, if for example $(\bar{N}_t)_1$ or $(\bar{N}_t)_2$ is input as a fraction one-half, one train would be added every 2 mi of new track. Also note that once the maximum number of trains (N_t) is reached, no new trains would be added.

For a single track with intermittent switches the program also adds switches to the system at appropriate intervals. The user specifies the desired distance between switches (item 24, Table 21), and the program keeps track of when a new switch should be added, and also the time delays (item 25, Table 21) associated with its assembly and deployment.

As discussed previously, track laying is either cyclic or continuous (Fig. 47) depending upon the nature of the excavation process. For cyclic operations, the length of added track is taken equal to the heading advance for the round and the time for track laying computed as

$$\Delta t_e = \Delta X_r / \dot{X}_e \tag{37}$$

where ΔX_r = length of round (ft)

$\dot{X}_e$ = average track extension rate (ft/hr)

The program is then executed for successive intervals until Δt_e is exceeded, at which time the available loading length (l_a) is updated by ΔX_r.

For continuous processes, the track extension and loading lengths (S and l_a respectively in Fig. 47) are updated using

$$S = S_0 + \Delta X_E - \dot{X}_e \Delta t \qquad S_m \geq S \geq 0 \tag{38}$$

$$l_a = (l_a)_0 + \dot{X}_e \Delta t \qquad (l_a)_{mx} \geq l_a \geq (l_a)_{mn} \tag{39}$$

where Δt = time interval (hr)

ΔX_E = excavator advance in Δt (ft)

$\dot{X}_e$ = track extension rate (ft/hr)

S_0 = available track extension length at beginning of interval (ft)

$(l_a)_0$ = available loading length at beginning of interval (ft)

$(l_a)_{mx}$ = maximum loading length allowed before switch is moved (ft)

$(l_a)_{mn}$ = minimum loading length (ft)

S_m = maximum permitted excavator runaway tolerance (ft)

If S results in a value less than zero (i.e., a case where the track has caught up to the excavator), then S is simply equated to zero for that interval. If $S > S_m$ (maximum separation between excavator and track), excavation is shut down until $S \leq S_m$ again. Also if l_a becomes greater than the maximum loading area allowed $(l_a)_{mx}$, the California switch is advanced forward until l_a equals the minimum loading area $(l_a)_{mn}$. The time delay to accomplish this is calculated using

$$\Delta t_d = [(l_a)_{mx} - (l_a)_{mn}]/\dot{X}_s \tag{40}$$

where $\dot{X}_s$ = California switch speed along main line (ft/hr).

The average track extension rate, $\dot{X}_e$, depends on a number of factors including the number of tracks, the rail size, the degree of roadbed preparation, the size of the track laying crew, equipment used etc. In addition, the rail design employed is a function of the expected wheel loads, the condition of the roadbed, the spacing of cross ties, etc. (see Table 23). Good roadbed preparation and track laying procedures, comparable to those used in commercial railroads, result in low average advance rates (~ 4 ft/hr for 90-lb rail, 4-man crew) but can support high-speed trains. In tunnel work, snaky, uneven tracks, cheaply and hastily laid down with little or no attempt to properly tamp the ties or ballast for unevenness or rigidity of rail may lead to greater advance rates (~ 10 or more ft/hr) but with corresponding reductions in the speeds and wheel loads that can be supported without potential derailments, etc. Consequently, the relationship between the track extension rate, $\dot{X}_e$, and allowed train speed for various track designs, roadbed conditions, etc., should be established. This would require additional data gathering to identify realistic correlations that could then be easily incorporated into the model.

c. Cost Information

The cost categories associated with subroutine RAILEX are plant and equipment and direct labor. Plant and equipment (see Table 21) is the cost of track materials prorated on a dollar per foot basis. These materials include ballast, rails, ties, etc., and can be estimated using Fig. 48 (taken from Ref. 47).

Direct labor is the cost of the "bull gang" required for track laying (Table 21). This should include both the men at the face as well as those that handle track materials at the stockpile in the portal or shaft area.

TABLE 23

RECOMMENDED MAXIMUM LOAD, ONE WHEEL, IN POUNDS [23]

Weight of rail,* lb/yd	Tie Spacing, in.			
	24	30	36	42
8	800	600	500	400
12	1,800	1,300	1,100	1,000
16	2,700	2,200	1,800	1,500
20	3,800	3,100	2,500	2,100
25	4,700	3,800	3,100	2,700
30	6,700	5,400	4,500	3,900
35	8,100	6,400	5,400	4,600
40	9,700	7,700	6,400	5,500
45	11,300	9,100	7,600	6,500
50	13,300	10,600	8,900	7,600
55	15,300	12,300	10,200	8,800
60	17,700	14,100	11,600	10,000

*Standard lengths: 30 ft for weights up to 45 lb/yd.
33 ft for 50 lb/yd or heavier.

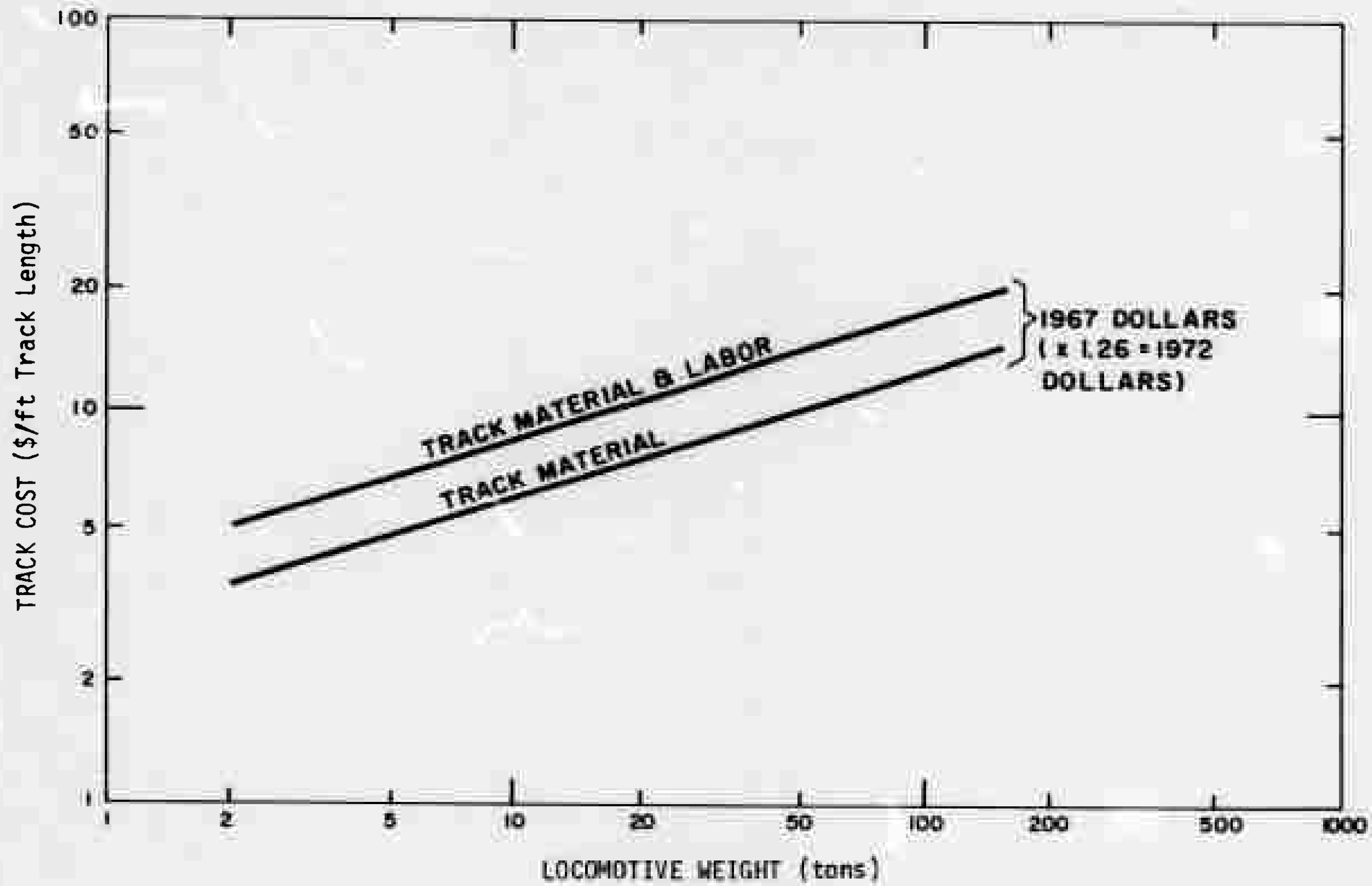

Figure 48. Conventional Rail Systems Track Cost [47]

4. Subroutine RAILDS

Subroutine RAILDS unloads muck from trains which have passed into the discharge area (Fig. 43).

To use this subroutine, the user specifies the maximum number of trains allowed in the discharge area at one time, the time it takes to unload one train, and the maximum allowed speed of a train entering and leaving the area (item 21, Table 21).

The subroutine accepts a train for unloading only if there is room available. If there is not, it is put into an unloading queue for future access.

Once accepted a train is unloaded by summing the time intervals (each time the subroutine is called) until the unloading time is equaled or exceeded for the train. At this point for the double-track case, the train is injected back into the system directly. In the case of single-track systems, it is stored in a queue until the first track segment ahead is clear (see subroutine ADDTRN).

Cost associated with subroutine RAILDS are the plant and equipment cost of the unloading equipment, and the direct labor to operate it. These are provided for in the model as user inputs.

5. Subroutine RAILMT

Subroutine RAILMT accounts for train maintenance periods. In terms of performance parameters the user provides either a unit train availability or down time per maintenance, the time between maintenance periods, and the number of trains allowed in maintenance at one time (item 19, Table 21).

The subroutine accepts trains for maintenance if there is room in the maintenance area. If not the train is allowed to continue haulage until room becomes available.

Once accepted, a train is taken out of the system for a period equal to the down time specified directly as a user input or calculated using the availability specified and time between maintenance periods. For each train in maintenance, a counter sums the intervals each time the subroutine is called, until maintenance is completed. For the double-track case, the train is then injected back into the system directly, or for the single-track case stored in a queue until the first track segment ahead is clear (see subroutine ADDTRN).

Costs associated with subroutine RAILMT include plant and equipment for the maintenance shop; job materials for minor servicing and repairs (prorated per maintenance period); and the direct labor for the maintenance crew including foreman, mechanics, and electricians. These are provided for as user inputs in Table 21.

6. Subroutine ADDTRN

Used only for single-track systems, subroutine ADDTRN is simply a utility program for inserting trains into the system. This subroutine is called by RAILEX if a new train is to be inserted for system extension, or by RAILDS or RAILMT if existing trains coming from the discharge or maintenance area are to be inserted.

If a train is available for insertion, this routine examines the first segment of track (including the switch) beyond the shaft or portal to ascertain when the train can proceed without interfering with oncoming traffic, etc. (see Fig. 43).

C. TRUCK SYSTEM

1. Introduction

By the truck system is meant a fleet of rubber-tired, self-propelled vehicles that travel uncontrained in the tunnel. Power is provided by either diesel (the most common), electric, or compressed-air motors. The diesel is normally equipped with exhaust scrubbers which remove some of the particles and irritant gases, mainly oxides of sulfur and aldehydes. Gasoline trucks are not used because of excessive concentrations of carbon monoxide in their exhausts.

Available trucks are either rigid frame constructed, or of the tractor-trailer type, both having various dumping arrangements such as bottom dumps, side dumps, and tilt bodies for rear-end dumping. A large range of capacities is also available: as small as the 3-ton, 22-hp diesel Getman KD-2 ore carrier, to as high as the 30-ton, 300-hp, tractor-rear-dump trailer, Caterpillar PR 621 (Table 24).* Both the rigid frame trucks and articulated haulers have the capability of transporting both construction materials and personnel, as well as bulk materials, although special configurations of the cargo box may be required in some instances.

The major advantage of rubber-tired systems is their flexibility. They have minimum system extension requirements (i.e., no guideways are required for operation), can operate on grades up to 10% or more, can readily be moved from one heading to another when alternate heading crews are used, and can be mobilized quickly (of great advantage in short tunnels).

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*Off-highway trucks are available in capacities exceeding 150 tons, but have limited application to tunnels because of their restrictive size and maneuverability.

TABLE 24

REPRESENTATIVE DIESEL-POWERED TRUCKS USED IN TUNNELING AND MINING

VEHICLE MODEL	Empty Wt (lbs)	Payload Max, lbs	Capacities (yd^3)* Struck	Capacities (yd^3)* Heaped	Overall Length	Overall Width	Overall Height	Loading Height	Engine Make & Model	Engine Flywheel hp	Approximate Cost (FOB-1972) $
Getman Brothers											
KD-2	4,750	7,000	1.7	----	11'-2"	5'-2"	4'-10"	4'-5 1/2"	Deutz	22hp@2200rpm	$ 7,060
KD-6	10,500	10,000	3.9	----	15'-8"	7'-0"	5'-2"	4'-7"	Deutz F4L-812	45hp@2000rpm	14,500
Aveling-Barford											
10 ton (mine car)	14,350	20,000	6.3	7.0	15'-8"	7'-9 5/8"	8'-10"	7'-6"	Leyland 400 S	138hp@2200rpm	25,100
11 1/2 ton (SP)	16,000	22,500	7.0	8.0	15'-10"	7'-11 1/4"	10'-0"	8'-2"	Leyland 401	138hp@2200rpm	26,000
15 ton (SL 300)	22,150	30,000	9.0	10.5	16'-2"	10'-3 1/4"	10'-10 3/4"	8'-10 1/8"	Leyland AU 600	163hp@1200rpm	36,500
17 ton (SL 340)	22,500	34,000	11.75	13.5	18'-3"	9'-0"	10'-7"	9'-4"	Leyland 680	201hp@2200rpm	40,000
Koehring											
460 Dumptor	17,470	18,000	5.5	7	15'-10"	8'-5"	9'-10"	7'-8 1/2"	GM 4-71 N	119hp@1800rpm	32,350
1560 Dumptor	35,730	36,500	11.0	13.5	19'-10"	10'-7"	10'-11"	9'-3"	GM 6-71 N	227hp@2100rpm	58,900
2460 Dumptor	38,000	48,000	16.0	21.0	23'-3"	11'0"	11'-8"	9'-2"	GM 8V-71 N	290hp@2100rpm	65,400
Euclid											
R-13	24,200	26,000	8.7	10.9	21'-6 1/2"	9'-2 1/2"	10'-7"	7'-10 1/2"	Detroit Diesel 4-71 N	160hp@2100rpm	36,600
R-22	36,250	44,000	14.7	19.0	25'-7"	11'-3 1/2"	12'-7 3/4"	8'-11"	Detroit Diesel 6-71 N	239hp@2100rpm	50,185
Caterpillar											
PR 621	55,765	60,000	20	24	32'-10"	11'-5"	12'-4"	6'-7"	Cat 621	300hp@2200rpm	------

*SAE 2:1 rating

The disadvantages include: the fact that they take up an excessive amount of tunnel cross-sectional area; they are wider than rail-mounted equipment of similar capacity, which necessitates wider roadbeds; they are relatively inefficient, requiring more horsepower per unit payload, which results in larger ventilation requirements; they require a well-graded, firm surface to operate (wet tunnels become a problem) and to prevent excessive and costly tire wear.

The activities associated with the truck system, including subroutine names used in the model, are summarized in Table 25. Also the input specification sheet including performance and cost information are given in Table 26. The reader should note that these activities and input specifications are very similar to the train system discussed previously in Sec. IV-B (see Tables 20 and 21). Correspondingly, the logistics and mathematical representations for the various activities are also very similar. This should not be surprising since both systems are intermittent in nature, and differ mainly in the performance and cost parameters assigned to the same basic equations, and the fact that truck systems (except for roadbed ballast) have little system extension requirements. The subsequent sections therefore highlight the major differences between the truck and train systems. In other words, mathematical performance and cost representations, unless explicitly stated otherwise, are the same as those already given in Sec. IV-B.

TABLE 25

SUMMARY OF TRUCK SYSTEM ACTIVITIES

Subroutine TRUKHL

Updates truck dynamics accounting for accelerations and decelerations

Subroutine TRUKEX

System extension - road ballast and truck addition

Subroutine TRUKDS

Unloads muck at discharge point

Subroutine TRUKMT

Truck maintenance and repair

TABLE 26

TRUCK SYSTEM PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

Truck Information

1. Maximum number of trucks__________
2. Number of trucks
   (a) Initially__________
   (b) To be added per mile of system __________
3. Capacity of truck: Volume (yd^3)__________
   Weight (tons)*__________
4. Empty weight of truck (tons)__________
5. Drive train efficiency*__________
6. Traction coefficient*__________
7. Rolling resistance factor* (lb/ton)__________
8. Rated flywheel horsepower (continuous operation)______
9. Peak horsepower available (accelerating)__________

   or

   Rimpull Characteristics

RP (lb)	Speed (mph)

10. Maximum allowed speed (mph)__________
11. Maximum allowed acceleration (mph^2)__________
12. In lieu of items 4-11
    (a) Peak speed Empty (mph)__________
    Full (mph)__________
    (b) Peak acceleration Empty (mph^2)__________
    Full (mph^2)__________
    (c) Horsepower consumed at peak speed Empty______
    Full______
    (d) Horsepower consumed at peak acceleration
    Empty______
    Full______
13. Average braking deceleration* Empty (mph^2)__________
    Full (mph^2)__________
14. Fuel consumption rate (gal/hp-hr)__________
15. Road (ballast) extension
    (a) Rate (ft/hr)__________
    (b) Excavator runaway tolerance (continuous excavation)-(ft)__________
16. Maintenance parameters
    (a) Single truck availability__________
    (b) In lieu of (15-a) down time per maintenance cycle (hr)__________
    (c) Time between maintenance periods (days)______
    (d) Number of trucks allowed in maintenance at one time__________
17. Muck loading and unloading
    (a) Number of trucks allowed in loading area at one time__________
    (b) Number of trucks allowed in discharge area______
    (c) Time to unload one truck (hr)__________
    (d) Maximum allowed speed for truck entering and leaving discharge area (mph)__________
    (e) Distance from portal to discharge area (ft)______

TABLE 26 (cont.)
TRUCK SYSTEM COST INFORMATION AND INPUT SPECIFICATIONS

1. Plant & Equipment

Item	Ownership (Rental) Cost Per	Hr (check one)	Ft	Unusual Developmental Cost Per	Hr (check one)	Ft
MAJOR ITEMS:						
Truck Unit						
Maintenance Shop						
Road Ballast						
ADDITIONAL ITEMS:						

2. Job Materials

Item	Cost Value	Cost Unit	Lifetime Value	Lifetime Unit
MAJOR ITEMS:				
Diesel Fuel*		$/gal	-	-
Minor Servicing and Repair (Tires, lubrication, etc.)		$/Maint.	-	-
ADDITIONAL ITEMS:		$/ft		

3. Direct Labor

Labor Type	Number Required/Shift	Rate $/hr
MAJOR TYPES:		
Truck Operators		
Maintenance		
Foremen		
Mechanics		
Bull Gang		
Foremen		
Laborer		
ADDITIONAL TYPES:		

4. Permanent Materials

Item	Cost Value	Cost Unit
(None Required)		

2. Subroutine TRUKHL

a. Function

Subroutine TRUKHL simulates truck dynamics accounting for accelerations and decelerations. It is similar to RAILHL in Sec. IV-B-2. Inputs to the subroutine (besides system performance and cost information) include the time increment, position of the face, present number of trucks in the system, and the location and state of each truck (i.e., empty or full, accelerating, etc.). The outputs include: an updated truck state vector accounting for motions and state changes occurring during the time interval; an updated load queue vector containing trucks available for loading; and finally job material, plant and equipment, and labor costs accrued during the interval.

For long tunnels, it is presumed that trucks can pass each other (see the similar to the rail haul geometry for double tracks shown in Fig. 43). If the truck sizes prevent their passing, niches could be excavated along the tunnel at various intervals to provide passing areas. This possibility is not presently modeled but could easily be by adapting that part of the RAILHL subroutine used to model the single-track train case as shown in Fig. 43.

b. Performance Model

With the number of cars $N_c = 1$, Eqs. 1 through 3 for the train case can also be used to establish an approximate higher bound on the number of trucks (of a given capacity) required to sustain a given advance rate. Also, Table 24 may be used to establish truck sizes and capacities compatible with a given tunnel cross section. This table compiles characteristic data on available trucks used in tunnel and mining operations.

The logic of subroutine TRUKHL presumes a given truck can be in one of ten different states (opposed to swelve for the train case, i.e., empty and full trains in switches are excluded) depending on its location and the status of operations at the time interval of interest. These states are:

1. Empty or loading and stopped
2. Full and stopped
3. Empty and accelerating
4. Full and accelerating
5. Empty and at peak speed
6. Full and at peak speed
7. Empty and decelerating
8. Full and decelerating
9. In the discharge area
10. In the maintenance area

For accelerating trucks, Eq. 4 through 7 are used but with the following substituted to calculate the empty and full accelerations (a_e and a_f) if the kinetics solution option is selected:

$$a_e = 39.4\ F_e/W_e \leq a_m \quad (\text{mph}^2) \tag{41}$$

$$a_f = 39.4\ F_f/W_f \leq a_m \quad (\text{mph}^2) \tag{42}$$

where the 39.4 is a conversion constant and

$$F_e = Rp_e - R_e \quad (\text{lb}) \tag{43}$$

$$F_f = Rp_f - R_f \quad (\text{lb}) \tag{44}$$

$$W_f = W_e + 1.35\ V_c \quad (\text{tons}) \tag{45}$$

where W_e = empty weight of truck (tons)

W_f = full weight of truck (tons)

V_c = volume capacity of truck (yd^3)

F_e = net force available for accelerating empty truck (lb)

F_f = net force available for accelerating full truck (lb)

Rp_e = available rimpull - empty truck (lb)

Rp_f = available rimpull - full truck (lb)

R_e = rolling resistance - empty truck (lb)

R_f = rolling resistance - full truck (lb)

The pulling force measured at the rim of the drive wheels is called the rimpull, and the amount available for accelerating depends on the available traction between wheel and ground or on the power delivered to the wheels for a given truck. It is computed as the smaller value given by

$$Rp_e = \begin{cases} 2000\, f_t \cdot W_e & \text{(traction limit)} \\ \dfrac{375\, HP_a \cdot E_\ell}{v_1} & \text{(power limit)} \end{cases} \quad \text{(lb)} \qquad (46)$$

and

$$Rp_f = \begin{cases} 2000\, f_t \cdot W_f & \text{(traction limit)} \\ \dfrac{375\, HP_a \cdot E_\ell}{v_1} & \text{(power limit)} \end{cases} \quad \text{(lb)} \qquad (47)$$

where f_t = coefficient of traction (see Table 27) taken as 0.36 unless input

HP_a = peak available horsepower of truck (hp)

E_ℓ = drive train efficiency of truck taken as 0.85 unless input

Instead of the above power equation, the user also has the option of providing a rimpull characteristic for the truck, an example of which is given in Fig. 49 for a Euclid R-13 Rear Dump. Note that this rimpull characteristic implies a certain peak horsepower and drive train and that the above power equation above is simply an approximate fit.

TABLE 27
COEFFICIENTS OF TRACTION[49]

Materials	Rubber Tires
Concrete	0.90
Clay loam, dry	0.55
Clay loam, wet	0.45
Rutted clay loam	0.40
Dry sand	0.20
Wet sand	0.40
Quarry pit	0.65
Gravel road (loose surface)	0.36
Packed snow	0.20
Ice	0.12
Firm earth	0.55
Loose earth	0.45
Coal, stockpiled	0.45

The rolling resistance (R_e and R_f) is determined using

$$R_e = f_R W_e \tag{48}$$

$$R_f = f_R W_f \tag{49}$$

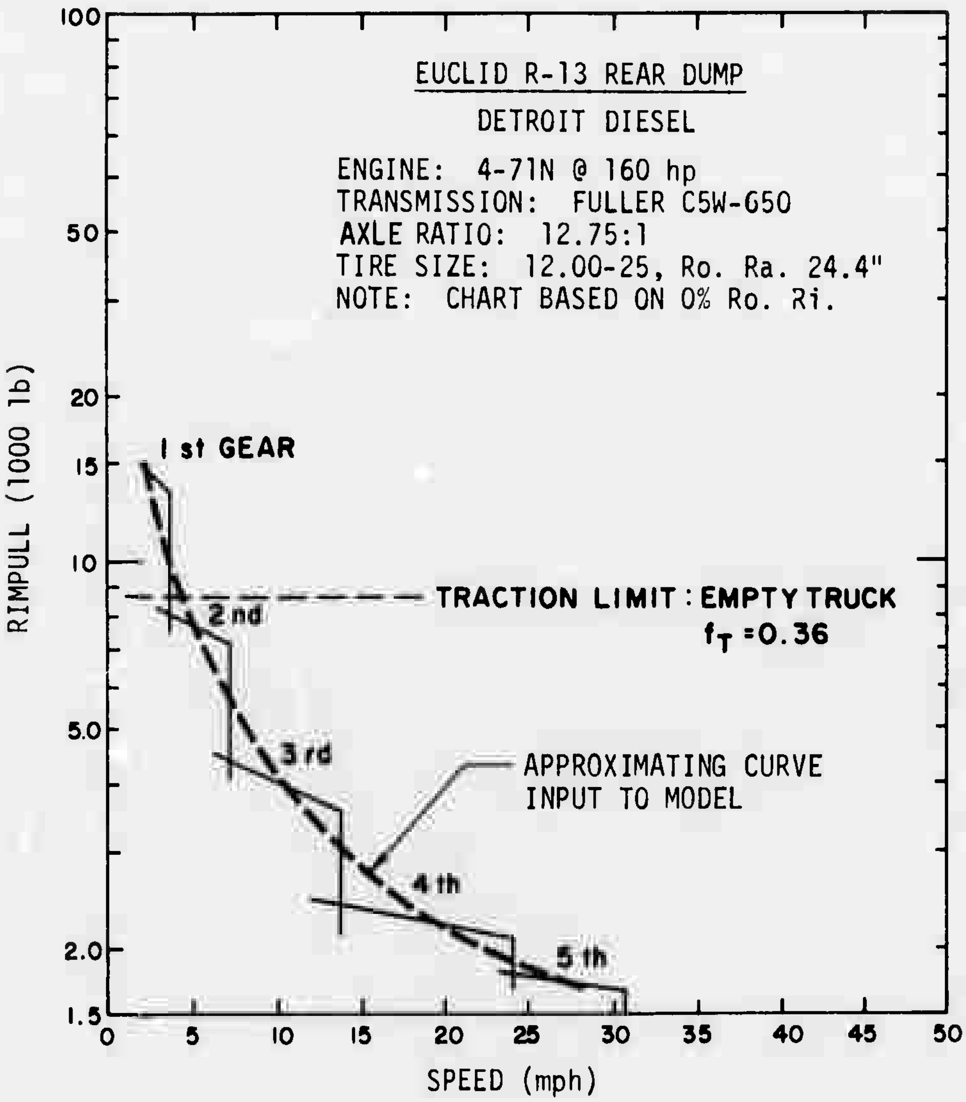

Figure 49. Example Truck Rimpull Curve

where f_R is the rolling resistance factor taken as 65 lb/ton unless input by the user. Typical rolling resistance factors for the user. Typical rolling resistance factors for various ground conditions are given in Table 28.

TABLE 28

TYPICAL ROLLING-RESISTANCE FACTORS FOR RUBBER-TIRED EQUIPMENT[50]

Description of Haul Road	lb/ton
Hard, smooth, stabilized, without penetration under load	40
Firm, smooth, flexing slightly under load	65
Rutted dirt, flexing considerably under load	100
Rutted dirt, no stabilization, somewhat soft under load	150
Soft, rutted mud or sand, deep penetration under load	200-400

For trucks traveling at peak speed, Eqs. 20 and 21 are used to update the dynamics. Again if the kinetics option is selected, the following equations are substituted (for Eqs. 22 and 23) to compute peak speeds for the empty and full conditions

$$v_e = 375\ E_1\ HP_v/R_e \leq v_m \tag{50}$$

$$v_f = 375\ E_1\ HP_v/R_f \leq v_m \tag{51}$$

where HP_v = rated net flywheel horsepower of the truck and v_m = maximum allowed velocity (mph).

For decelerating trucks, Eq. 24 through 27 are used, but here the user need not specify an average braking deceleration. If not provided, the model assumes decelerations equal accelerations.

c. Cost Information

Cost categories applying to subroutine TRUKHL include direct labor, plant and equipment, and job materials.

Typical labor types required to operate the truck system are listed in Table 26 under cost information. Those associated specifically with haulage include truck operators and dispatchers. It can be presumed that one operator will be required for each truck and that at least one dispatcher will be required for every 2 mi of system.

The plant and equipment costs associated with TRUKHL is the ownership cost of the truck units. Table 24 and Fig. 50 provide representative capital costs (excluding taxes, investment and major overhead) for estimating purposes. If one selects to depreciate costs on a per hour basis (see Table 26), the life of a unit can be taken conservatively at 15,000 hr.

The only job material costs computed in subroutine TRUKHL are haulage power costs. The model assumes that energy consumed during periods of deceleration is negligible compared to accelerating or peak velocity consumption. The model does take into account, though, the difference between accelerating and peak velocity consumption for both full and empty trucks.

Power costs are computed using Eq. 30 through 34 with the exception that the fuel consumption rate, F_c, is taken at 0.06 gal/hp-hr, and

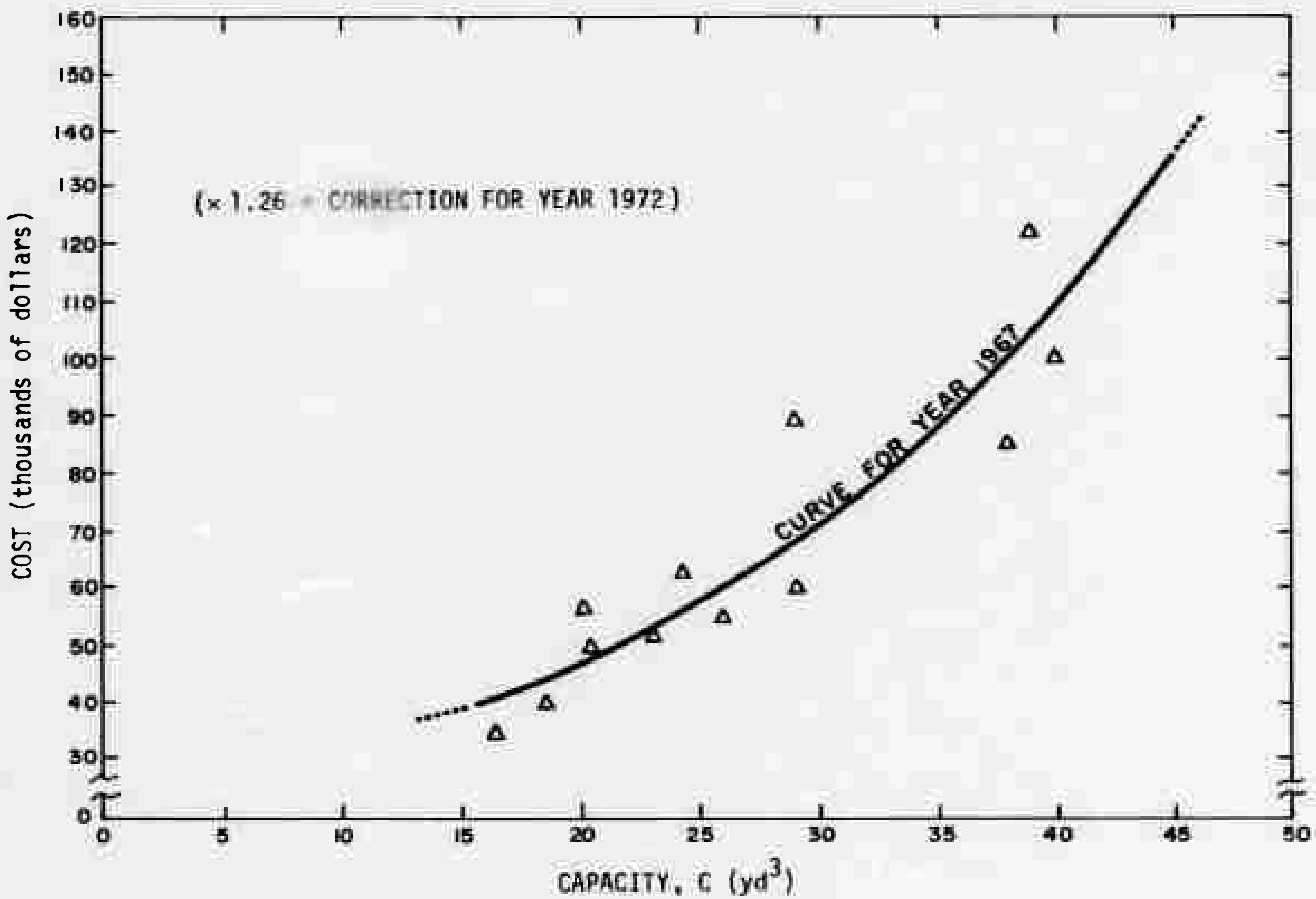

Figure 50. Truck Vehicle Cost [47]

$$G_e = W_e$$

$$G_f = W_e + 1.35\ V_c$$

in Eqs. 31 and 32, where W_e and V_c are as defined in the previous section.

3. Subroutine TRUKEX

Subroutine TRUKEX's main function is to add trucks to the system in order to maintain haulage rates as the length of a tunnel increases. The algorithm for truck addition is identical to that used in subroutine RAILEX for train insertion which is described in Sec. IV-B-3-b.

Subroutine TRUKEX can also be used to simulate truck roadway preparation. In most practical cases, this probably entails the laying of a coarse aggregate (ballast) layer of material to help stabilize and provide a well-graded roadway; but if high wheel loads or speeds must be sustained, it could also include the addition of an asphalt (or other binder) as a surface layer.

The algorithm which models roadbed preparation is similar to that used in subroutine RAILEX for track laying. The user provides a road extension rate, $\dot{X}_e$; and, if the excavation process is continuous, he also provides a maximum permitted excavator runaway tolerance (item 15, Table 26).

For cyclic excavation, the length of available roadway is updated by the heading advance for the round, ΔX_r, and the time to accomplish this, Δt_e, is computed by Eq. 37.

For continuous processes the roadbed extension area, S, which is equivalent to the track-laying area in Fig. 47, is updated for each time interval by

$$S = S_0 + \Delta X_E - \dot{X}_e \Delta t \qquad S_m > S \geq 0 \tag{52}$$

where Δt = time interval (hr)

ΔX_E = excavator advance in Δt (ft)

$\dot{X}_e$ = roadbed extension rate (ft/hr)

S_0 = roadbed extension length at beginning of interval (ft)

S_m = maximum allowed excavator runaway (ft)

If the value for S is less than zero, the roadbed has caught up to the excavator, in which case S is simply equated to zero for that interval. If the value for S is greater than S_m (the maximum allowed separation between the excavator and roadway), the excavator is shut down until $S < S_m$.

The appropriate roadbed extension rate, $\dot{X}_e$, for a given case can depend on a number of factors including: the geometry of the tunnel (e.g., circular tunnels generally require more ballast than horseshoe or vertical-walled tunnels); the degree of preparation required to provide a firm and stable roadway (this, in turn, depends on the subgrade soil conditions and water inflow rates--which can vary with the tunnel length--as well as the truck speeds and wheel loads that must be sustained); and the size of the crew engaged in roadbed preparation; the equipment used. Adequate preparation may cause relatively low average preparation rates, but in turn will result in a roadbed capable of supporting high speeds and wheel loads. On the other hand, minimal preparation may increase preparation rates but lower the speeds and wheel loads that can be sustained.

The relationship between tunnel road extension rate, allowed truck speed, and wheel loads for various degrees of roadway preparation, and tunnel subsoil and hydrological conditions is not presently available. To establish an approximation lower bound, the following equation, used in roadway construction, will give to estimate rates of laying gravel surfaces.[50] This equation assumes a crew of 3 men working with truck spreaders and roller equipment and therefore should be used with caution.

$$\dot{X}_e = 500/(W_s \cdot t_s) \qquad \text{(ft/hr)} \tag{53}$$

where W_s = width of surface (ft)

t_s = average thickness of surface (ft)

The cost categories associated with subroutine TRUKEX are plant and equipment and direct labor. Plant and equipment cost (see Table 26) is the cost of road materials prorated on a dollar per foot basis. For coarse aggregate, one can use a $\$3.50/yd^3$ unit cost for estimating purposes.[50]

Direct labor is the cost of the "bull gang" assigned to work on the roadway (see Table 25). This should include both the men at the face as well as those that handle ballast materials, etc., at a stockpile at the portal or shaft area.

4. <u>Subroutine TRUKDS</u>

Subroutine TRUKDS unloads muck from trucks which have passed into the discharge area.

The user specifies the maximum number of trucks allowed in the discharge area at one time, the time it takes to unload one truck, and the maximum allowed speed of a truck entering and leaving the area (Item 17d, Table 26).

The subroutine accepts a truck for unloading only if there is room available. If there is no room, the trucks are put into a queue for future unloading. Once accepted a truck is unloaded by summing the time intervals (each time the subroutine is called) until the unloading time is equaled or exceeded. The truck is then released and continues back to the tunnel face area.

5. Subroutine TRUKMT

Subroutine TRUKMT accounts for truck maintenance periods. The user provides either a unit truck availability or down time per maintenance, the time between maintenance periods, and the number of trucks allowed in maintenance at one time (item 16d, Table 26). Typical availability factors for truck hauling units as a function of anticipated life are given in Table 29. These values can be considered adequate for estimating purposes, assuming good preventive maintenance practices, "average" operating conditions, and operation for 50% or less of any 24-hr period.[50]

The subroutine accepts trucks for maintenance if there is room in the maintenance area. If there is no room the truck is allowed to continue haulage until room becomes available.

Once accepted, a train is taken out of the system for a period equal to the down time specified directly as a user input or calculated using the availability and time between maintenance periods specified. For each truck in maintenance, a counter sums the intervals each time the subroutine is called, until maintenance is completed. The truck is then released and continues back to the face area.

TABLE 29

AVAILABILITIES OF TRUCK HAULING UNITS

	Years of anticipated life					
	1	2	3	4	5	6
Types of unit	Average availability, percent*					
Rear dumps................	90	89	88	86	84	83
Bottom dumps, side dumps..	92	90	89	88	86	86
Rocker-type dumpers.......	88	86	83	80	78	78

*Factors such as severe operating conditions, multishift operations and unsatisfactory preventive maintenance practices can reduce machine availability by as much as 20 percent below the levels considered "average." Similarly, favorable conditions may increase first year values by as much as 5 percent and can show an increasing advantage over average conditions and subsequent years.

Costs associated with TRUKMT include plant and equipment for the maintenance shop; job materials for tires, and minor servicing and repair (prorated per maintenance period); and direct labor for the maintenance crew including foreman and mechanics. These are provided for as user inputs in Table 26.

Typical costs for tires and repairs over a 2100-hr period (typical life of a tire) can be estimated using the equation (taken from Ref. 47)

$$\text{Cost of tires and minor repairs (\$)} = 740 \cdot (V_c - 4) + 350\ (V_c - 5.5)$$

where V_c = truck capacity (yd^3).

D. MAIN-LINE CONVEYOR SYSTEMS

1. Introduction

The use of troughed-belt conveyors* for main-line transport is a relatively new and novel concept. Although conveyors have been used for many years to transport bulk earth materials in the mining and construction materials industries, they have had somewhat limited application in long haul transport for tunneling. The advantage they offer of uninterrupted flow of materials for the entire length of the system, while operating, is countered by their disadvantages, which include: system extension problems--as the length of tunnel increases, the conveyor supporting structure and belt must be extended and the belt spliced. This results in delays and interruptions in flow. In addition, conveyor systems cannot generally handle the inbound flow of construction materials and transportation of personnel, but must be supplemented by some form of unitized transport system. They are also limited in the range of muck characteristics that can be handled. Excluded materials include: large size fragments, extremely abrasive and angular materials which can cause excessive belt wear and damage; and wet-running or

---

*Troughed-belt conveyors (see Fig. 51) consist of a heavy duty belt forming a continuous longitudinal trough that facilitates the transport of granular bulk materials. The depth and angle (20° to 45°) of this trough is determined by the troughing idlers on which the belt rides. Trough belt conveyors are most suited to, and most commonly employed for transport of bulk materials and therefore have received exclusive emphasis here.

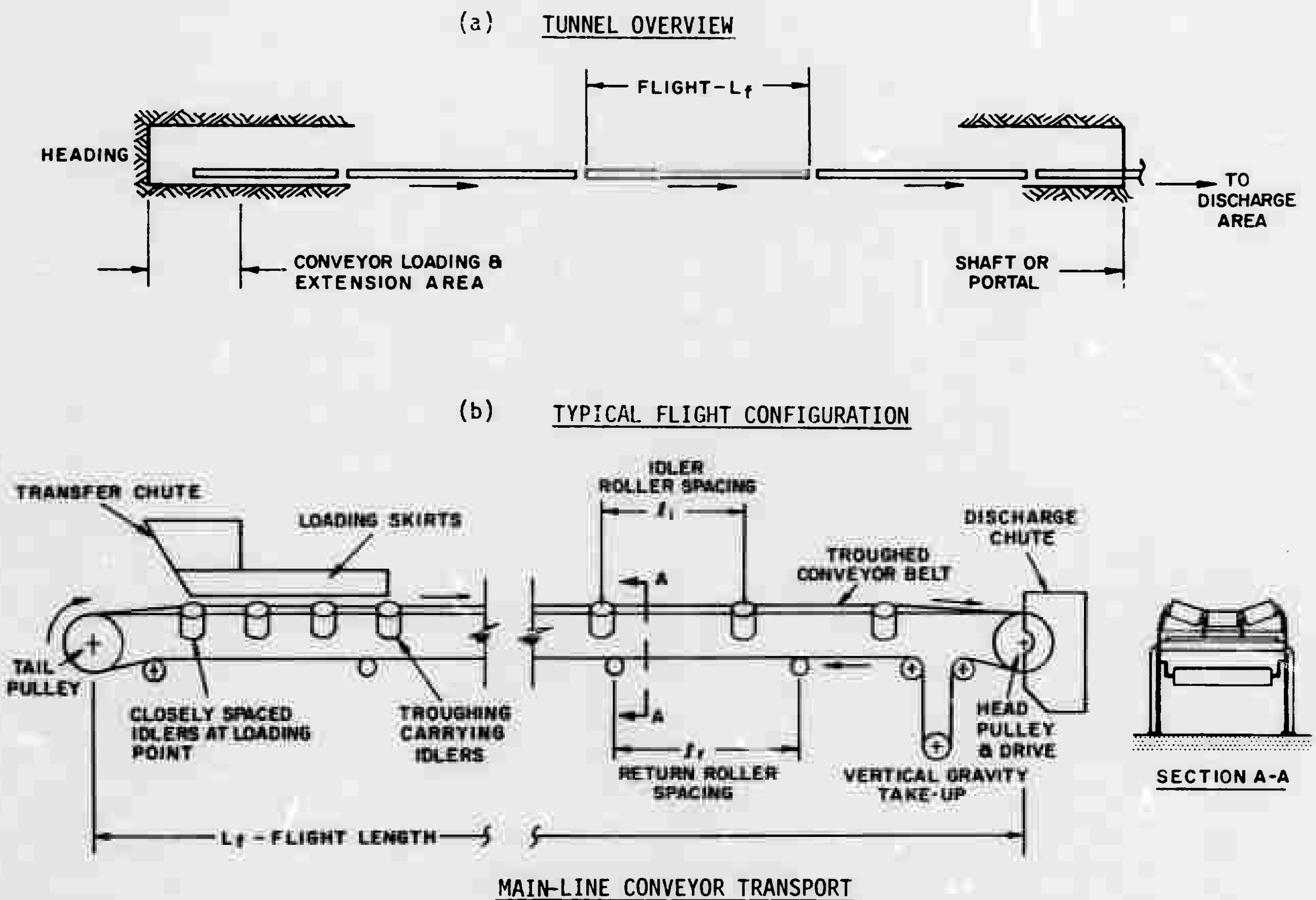

Figure 51. Main-Line Conveyor Transport Geometry

sticky materials. Also, the availability of belt conveyors is generally less compared to such systems as trains or trucks that are inherently more rugged (i.e., belts or rollers are particularly vulnerable and may require frequent replacement and maintenance under the strenuous conditions often found in tunneling operations).

Nonetheless, recent advances in technology have enhanced the capabilities of belt-conveyor systems. Belt speeds and widths are increasing (as high as 1200 fpm and 120 in.) resulting in corresponding increases in the capacity or tonnage rates that can be handled. These increases can be attributed to advances in the quality of belt materials, as well as the durability of belts, idlers, and other components. Also, with the advent of new mechanical belt splice techniques, which require considerably less time to install than the old vulcanized splice, belt splicing delays have become less of a limiting factor. Finally, developments in related systems and components such as portable crushers, screens, etc., have also been instrumental in making belt conveyor systems more versatile and adaptable to heavy construction. The activities associated with the belt-conveyor process, including subroutine names used in the model, are summarized in Table 30. Also the input specification sheet including performance and cost information is given in Table 31. Subsequent sections discuss these activities and inputs in depth.

TABLE 30

MAIN-LINE CONVEYOR TRANSPORT ACTIVITIES

Subroutine TRNPRT

Muck transport

Subroutine EXTNSN

System extension

Subroutine SURGE

Accumulates and discharges muck from surge bin

Subroutine BELT

Maintenance and repair

---

2. Subroutine TRNPRT

a. Function

The function of subroutine TRNPRT is to transport muck continuously from a loading point in the near-face zone to a discharge point in the portal or shaft area.

Besides performance and cost parameters, inputs to the subroutine include: the time increment, the position of the face, the length of available conveyor, and the volume rate of muck loaded at the face during the time increment. The major output is the cost of power required to transport muck for the time interval.

Figure 51 presents the conveyor system configuration modeled. The total system is comprised of several separate conveyor units called flights. The length of a flight is generally determined by either available power or belt tension limitations.

TABLE 31

MAIN-LINE CONVEYOR PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

Conveyor Information

1. Belt speed (ft/min)__________
2. Belt width (in.)__________
3. Muck flow cross section (ft^2)__________
4. Length of a flight (ft)__________
5. Power per unit length (hp/ft)*__________
6. In lieu of item (1-5)
   - (a) Estimated peak tonnage rate (tons/hr)__________
   - (b) Maximum allowed cross section for conveyor (ft^2)__________
7. Surge bin capacity - if any (ft^3)
8. Maintenance parameters
   - (a) Availability of conveyor (%)
   - (b) In lieu of (8-a) down time per maintenance period (hr)__________
   - (c) Time between maintenance periods (days)__________

Conveyor Extension

9. Conveyor structure extension rate (ft/hr)__________
10. Time required for belt splice (hr)__________
11. For continuous excavator and/or loading processes
    - (a) Maximum length of feeder loader (ft)__________
    - (b) Is length of feeder constant or is feeder extensible?__________
    - (c) For continuous excavators - maximum allowed runaway between machine and conveyor structure (ft)__________
12. For cyclic excavation and/or loading processes
    - (a) Maximum allowed distance between face and operational conveyor (ft)__________
    - (b) Minimum allowed distance from face and conveyor structure (ft)__________ (i.e., for blasting or mucking)

TABLE 31 (cont.)

MAIN-LINE CONVEYOR PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

1. Plant & Equipment

Item	Ownership (Rental Cost Per	Hr	Ft (check one)	Unusual Developmental Cost Per	Hr	Ft (check one)
MAJOR ITEMS:						
Supporting Structure						
Transfer Equipment						
Idler & Return Rollers						
Drive Equipment (Motors, Pulleys, etc.)						
Wiring and Controls						
Belt						
ADDITIONAL ITEMS:						

2. Job Materials

Item	Cost		Lifetime	
	Value	Unit	Value	Unit
MAJOR ITEMS:				
Power: Electric*		$/kW/hr	-	-
Servicing and repairs (per maintenance)		$	-	-
ADDITIONAL ITEMS:		$/ft	-	-

3. Direct Labor

Labor Type	Number Required/Shift	Rate $/hr
MAJOR TYPES:		
Operator		
Bull Gang		
Foreman		
Laborer		
Electricians		
Maintenance		
Foreman		
Mechanics		
Electricians		
ADDITIONAL TYPES:		

4. Permanent Materials

Item	Cost	
	Value	Unit
(None Required)		

A typical flight includes several basic components: a drive system, usually consisting of one or more motors, speed reducer, control system and head pulley, which provides the motive power for the belt; a support structure including idler rollers spaced at regular intervals that support the loaded belt and shape it into a trough configuration; return rollers, also spaced at regular intervals, to support the empty belt during its return cycle; a take-up arrangement to insure proper belt tensions at loading and other points, and also to prevent slippage at the drive pulley; transfer equipment including chutes, skirtboards, etc., used to insure a smooth transition of materials from one flight to another; and the belt itself usually composed of plies or layers of fabric, bound together by a friction and/or skim coat of rubber.

b. Performance Model

To use subroutine TRNPRT, the user must specify the general configuration and performance characteristics of the conveyor. This can be done in one of two ways: (1) he can specify the characteristics directly by providing inputs to items 1-5 in Table 31; or (2) he can select item 6, in which case, the program will determine the appropriate characteristics. If the latter option is selected, a design segment of the program is executed the first time TRNPRT is called.

The procedure and equations used in this design segment are based on the following general assumptions:*

1. The conveyor is used only for horizontal transport.

2. The material transported has the following general characteristics:**

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*The procedure outlined simplifies the conveyor design problem considerably. Nevertheless, the intention is to provide a first-order estimate of the requirements for which the procedure is felt to be adequate.

**Approximate values for broken granite.

(a) Bulk density ~ 100 lb/ft^3

(b) Angle of surcharge ~ 25°

(c) Maximum lump size: 6 in. and over

(d) Very abrasive and sharp

3. Length of a flight is determined by tension limitations for a mechanical splice in a RMA-70 fabric belt carcass.

4. Standard conveyor sizes and equipment are assumed. Also each flight is assumed to contain the types of equipment shown in Fig. 51b, which include:

   (a) Troughing idlers having three equal rollers at 20° troughing angle with 6 in. O.D. and 1 1/4 in. shaft

   (b) A bare drive pulley that is snubbed with a 220° wrap angle

   (c) An automatic vertical gravity take-up

   (d) Belt widths greater than 18 in. with a carcass material equivalent to a RMA-70 fabric and mechanical splices

Belt speeds (v_b) required to meet the user input peak tonnage rates can be calculated using

$$v_b = 33.3\ \dot{W}_p \Big/ \left[A_b \left(\frac{\rho_s}{K} \right) \right] \quad \text{(ft/min)} \tag{54}$$

where $\dot{W}_p$ = user estimated peak tonnage rate (ton/hr)

$\frac{\rho_s}{K}$ = bulk density of the muck (in situ density over swell factor) taken as 100 lb/ft^3

A_b = muck flow cross section of the belt (ft^2)

For the case of three equal roller idlers with a 20° troughing angle, and materials with an angle of surcharge equal to 25°, the muck flow cross section, A_b, is related to the belt width (w_b in inches) by[51]

$$A_b = \left(\frac{1}{144}\right)\left(0.1247\ w_b^{\ 2} - 0.8245\ w_b + 1.311\right) \quad (ft^2) \qquad (55)$$

Substitution of Eq. 55 into Eq. 54, with $\rho_s/K = 100\ lb/ft^3$, yields

$$v_b = 48\ \dot{W}_p / \left(0.1247\ w_b^{\ 2} - 0.8245\ w_b + 1.311\right) \quad (ft/min) \qquad (56)$$

Equation 56 relates the belt speed to the belt width for a given peak tonnage rate.

In the United States, conveyor belt manufacturers make the following standard sizes: 14, 16, 18, 20, 24, 30, 36, 42, 48, 54, 60, 66, and 72 in.[51] The minimum blet width required for a given application is determined by the lump size of the muck to be handled. Belts less than 18 in. in width cannot easily handle lump sizes exceeding 6 in. Therefore the 14 in. and 16 in. belt widths are excluded here.

Corresponding to these belt widths are certain maximum recommended belt speeds that depend, among other things, on the properties of the material to be conveyed. Recommended speeds for heavy, hard, sharp edged, coarse materials (those expected in hard rock applications) are given in Table 32.

TABLE 32

RECOMMENDED MAXIMUM BELT SPEEDS FOR VARIOUS BELT WIDTHS[51]
(HEAVY, SHARP-EDGED MATERIALS)

Belt Width W_b (in.)	Belt Speed v_b (ft/min)
18	300
20	350
24	400
30	450
36	500
42	550
48	600
54	600
60	600
72	600

The design algorithm in the program initially selects the smallest standard belt width, then solves Eq. 56 for v_b. Next, it checks the speeds (Table 32) to see if the maximum allowed speed is less than or greater than this computed speed. If it is less, the initially selected belt width is not adequate, in which case a larger belt width is tried until one is found where v_b maximum $\geq v_b$ computed. In this way the minimum belt width that can support the estimated peak tonnage within the recommended speed is found.

Another constraint to be satisfied, though, is the maximum allowed cross-section area of the conveyor specified by the user. Assuming standard equipment and support structure, conveyor cross sections (A_c) for various belt widths can be approximated by[47]

$$A_c = 10.84 + 0.357\ w_b - 0.00203\ w_b^2 + 0.000414\ w_b^2 \leq A_{cm}\ (ft^2) \qquad (57)$$

Substituting the belt width determined above, A_c is computed using Eq. 57 and checked to see if it exceeds the maximum user-specified cross section, A_{cm}. If it does, a flag is printed indicating a conveyor design cannot be found for the cross section and estimated peak tonnage rate input. The run is then terminated.

If a belt width and speed that meets the tonnage rate and cross-section requirements can be found, the program then estimates idler roller and return roller spacings (ℓ_i and ℓ_r, respectively in Fig. 51). The best spacing for idlers depends upon the weight of the belt plus the weight of the material load, and also on the catenary sag of the belt between the idlers. As a first estimate, Table 33 gives, as a function of belt width, troughing idler spacings used in general engineering practice when the amount of catenary belt sag is not specifically limited, and the material handled weighs approximately 100 lb/ft^3. Table 33 also provides return roller spacing recommended for general belt conveyor work.

TABLE 33
SUGGESTED NORMAL SPACING OF BELT IDLERS[51]

Belt Width w_b (in.)	Troughing Idlers, ℓ_i (ft)	Return Idlers, ℓ_r (ft)
18	5.0	10.0
20	4.5	10.0
24	4.0	10.0
30	4.0	10.0
36	4.0	10.0
42	3.5	10.0
48	3.5	10.0
54	3.5	10.0
60	3.0	10.0
66	3.0	8.0
72	3.0	8.0

The length of a flight, L_f, is determined next. This is done on the assumption that belt tension at a mechanical splice, for a RMA 70* fabric belt carcass, cannot exceed a certain amount.

For the conveyor geometry shown in Fig. 51, the maximum belt tension (T_m) occurs at the loaded side of the drive pulley and can be calculated using

$$T_m = T_e\,(1 + C_w) \qquad \text{(lb)} \qquad (58)$$

where C_w = wrap factor equal to 0.62 for a snubbed bare drive pulley, with a 220° wrap angle and automatic takeup

T_e = total effective belt tension required to move the belt, i.e., the difference between the tight side tension and slack side tension at the drive pulley (lb)

The effective belt tension can be computed using[51]

$$T_e = L_f\left[K_t\left(K_x + K_y q_b + 0.015\, q_b\right)\right] + K_y L_f A_b + F_{AC} \qquad \text{(lb)} \qquad (59)$$

where K_t = ambient temperature factor which can be taken equal to one for temperatures greater than 30°F

K_x = force, in pounds per foot of conveyor to rotate the idle and return rollers, and to cover the sliding resistance of the belt on the rollers (given below)

K_y = force, in pounds per foot of length, to flex the conveyor belt and load over idlers (given below)

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*Rubber Manufacturers Association rating meaning 70 lb/in. width per ply.

q_b = weight of belt in pounds per foot of length (see Table 34)

F_{AC} = additional allowance for accessories (i.e., skirtboard friction, non-driving pulleys, drive pulley friction, etc.) taken here as 500 lb.

K_x is calculated using[51]

$$K_x = 0.00068 \left[q_b + 100\ A_b \right] + \frac{2.13}{\ell_r} \qquad \text{(lb/ft)} \qquad (60)$$

where, for a known belt width, A_b is given by Eq. 55, ℓ_r by Table 33, and q_b by Table 34. (Equation 60 assumes 100 lb/ft^3 transported material, and 6 in. diameter 1 1/4 in. shaft idler rollers.)

TABLE 34

GENERAL CONVEYOR BELT CHARACTERISTIC USED IN THE MODEL[51]

Belt Width w_b (in.)	Belt Weight q_b (lb/ft)	Number of Plies (RMA-70 - 20° Trough Angle)
18	4.0	3
20	4.5	4
24	5.7	5
30	7.2	6
36	9.6	7
42	11.5	8
48	14.2	9
54	16.9	10
60	19.4	12
66	21.8	12
72	24.3	12

As a first approximation one can assume flight lengths of the order of 2500 ft; for which $K_y \approx 0.02$ lb/ft. Actually K_y is a function of the flight length, and also the applied belt and material loads. Approximate values for the loads and flight lengths of most interest are provided in Table 35.

TABLE 35
FACTOR K_y VALUES[51]
(For combined material and belt loads — 200 lb/ft)

L_f (ft)	K_y (lb/ft)
800	0.035
1000	0.032
1400	0.030
2000	0.024
2500	0.020
3000	0.019
$L_f < 800$ ft	$K_y \sim 0.035$ lb/ft
$L_f > 3000$ ft	$K_y \sim 0.018$ lb/ft

With the above substitutions, Eq. 59 now becomes

$$T_e = L_f \left[0.00068 \left(q_b + 100\ A_b \right) + \frac{2.13}{\ell_r} + 0.035\ q_b + 0.02\ A_b \right] + 500 \quad \text{(lb)} \qquad (61)$$

and Eq. 58 becomes

$$T_m = 1.62\ T_e \qquad \text{(lb)} \qquad (62)$$

As mentioned earlier, the maximum belt tension, T_m, is limited to the tension load that a mechanical splice in an RMA-70 can achieve. The rating for such a splice is given in Ref. 51 at 55 lb/in. of width per ply. Therefore,

$$T_m = 55\ N_p w_b \qquad \text{(lb)} \qquad (63)$$

where N_p = number of plies

w_b = belt width (lb)

For a given belt width, the number of plies required is determined by the combined muck and belt loads and also by troughability requirements. Approximate values are given, as a function of belt width, in Table 35 for the conditions assumed here.

Combining Eqs. 61, 62, and 63 and solving for the flight length, L_f, one obtains

$$L_f = \frac{55\ N_p w_b}{1.62\ \lambda} - \frac{500}{\lambda} \qquad \text{(ft)} \qquad (64)$$

where

$$\lambda = \left[0.00068 \left(q_b + 100\ A_b\right) + \frac{2.13}{\ell_r} + 0.035\ q_b + 0.02\ A_b\right] \text{(lb/ft)} \qquad (65)$$

Equations 64 and 65, together with Tables 33 and 35 are used in the model to establish the flight length, given the belt width computed previously.

Once the characteristics of the conveyor are established (either by user input or by the above design algorithm) subroutine TRNPRT accepts a muck volume rate, $\dot{Q}_L$ from the loader and for a given time interval performs the following calculations and checks: the required tonnage capacity necessary to support the loader ($\dot{W}_R$) is calculated and compared to the conveyor's available peak tonnage capacity ($\dot{W}_p$). That is

$$\dot{W}_R = \left(\frac{1}{2000}\right) \dot{Q}_L \, \rho_s / K \qquad \text{(ton/hr)} \qquad (66)$$

$$\dot{W}_p = \left(\frac{60}{2000}\right) A_b v_b \, \rho_s / K \qquad \text{(ton/hr)} \qquad (67)$$

where $\dot{Q}_L$ = loaded volume rate for the time interval (ft^3/hr)

ρ_s = in situ density of muck (fbs/ft^3)

K = swell factor taken as 1.7 unless input

A_b = muck flow cross section (ft^2)

v_b = belt speed (fpm)

If $\dot{W}_p = \dot{W}_R$, the conveyor is just able to handle the loading rate.

If $\dot{W}_p > \dot{W}_R$, additional conveyor capacity is available. In this case if a surge bin is provided in the system and muck is stored in it from previous time cycles, the program will call subroutine SURGE (Sec. II-D4) and a quantity of muck equal to the additional conveyor capacity; i.e.,

$$\Delta W = (\dot{W}_p - \dot{W}_R) \, \Delta t \qquad \text{(tons)} \qquad (68)$$

will be discharged <u>from</u> the bin.

If $\dot{W}_p < \dot{W}_R$, the conveyor cannot handle the loading rate. In this case, if no surge bin is provided, the loader (and excavator for continuous processes) will be shut down for the next time increment. If a surge bin is provided and is not filled, the excess muck, computed as

$$\Delta W = (\dot{W}_R - \dot{W}_p) \, \Delta t \qquad \text{(tons)} \qquad (69)$$

will be discharged <u>into</u> the bin.

c. Cost Information

Cost categories applying to subroutine TRNPRT include direct labor and job materials.

Direct labor includes operators to start up and shut down the system, and to keep an eye on it, to make sure it is running properly. The cost of this labor is provided for in Table 31 as a user input.

The only job material costs computed in TRNPRT are conveyor power costs. For a given time interval these are computed from

$$\text{Power Cost} = (0.746)\ HP_t \cdot C_p \cdot \Delta t/\rho \qquad (\$) \qquad (70)$$

where Δt = time interval (hr)

ρ = motor and reduction equipment efficiency taken as 0.76

C_p = power cost rate taken as 0.02 \$/kW-hr unless input

HP_t = total power consumption of the conveyor, determined as follows (hp)

If the user inputs the conveyor characteristics, he can also input the horsepower required per foot of conveyor (item 5, Table 31). In this case the total power consumption is computed as

$$HP_t = \overline{HP} \cdot L_c \qquad (\text{hp}) \qquad (71)$$

where $\overline{HP}$ = input power per foot of conveyor (hp)

L_c = total length of available conveyor (ft)

If he selects not to input $\overline{HP}$, the program then calculates the total power requirements using the CEMA horsepower formula from Ref. 51 . This formula is based on the simple equation from engineering mechanics where, for one conveyor flight,

$$HP_f = \frac{T_e v_b}{33{,}000} \qquad \text{(hp)} \tag{72}$$

where T_e = the total effective tension required to move the belt given by Eq. 59 (lb)

v_b = belt speed (ft/min)

If one uses the same assumptions regarding the factors K_t, K_x, and F_{AC} that were discussed in the previous section, but now allows K_y to vary according to the conveyor length (Table 34), Eq. 59 for T_e reduces to

$$T_e = L_f \left[0.00068 \left(q_b + 100\ A_b \right) + \frac{2.13}{\ell_r} + \left(K_y + 0.015 \right) q_b \right] \tag{73}$$

$$+ K_y L_f A_b + 500 \qquad \text{(lb)}$$

where, for a given belt width, q_b is given by Table 34, A_b by Eq. 55, and ℓ_r by Table 33; and for a given conveyor length K_y is given by Table 35.

We combine Eqs. 72 and 73 and assume the total conveyor is composed of N_f flights of length L_f, plus a segment in the near-face zone which is not yet a full flight in length. The power for the total system can now be computed from

$$HP_t = \frac{N_f v_b}{33{,}000} \left(L_f \lambda + K_{y1} L_f A_b + 550 \right) + \frac{\left(L_c - N_f L_f \right) v_b}{33{,}000} \left(\lambda + K_{y2} A_b \right) + \frac{550\ v_b}{33{,}000} \tag{74}$$

where $$\lambda = 0.00068 \left(q_b + 100\ A_b \right) + \frac{2.13}{\ell_r} + \left(K_y + .015 \right) q_b \tag{75}$$

K_{y1} = value for L_f from Table 35

K_{y2} = value for $L_c - N_f L_f$ from Table 35

3. Subroutine EXTNSN

a. Function and Scope

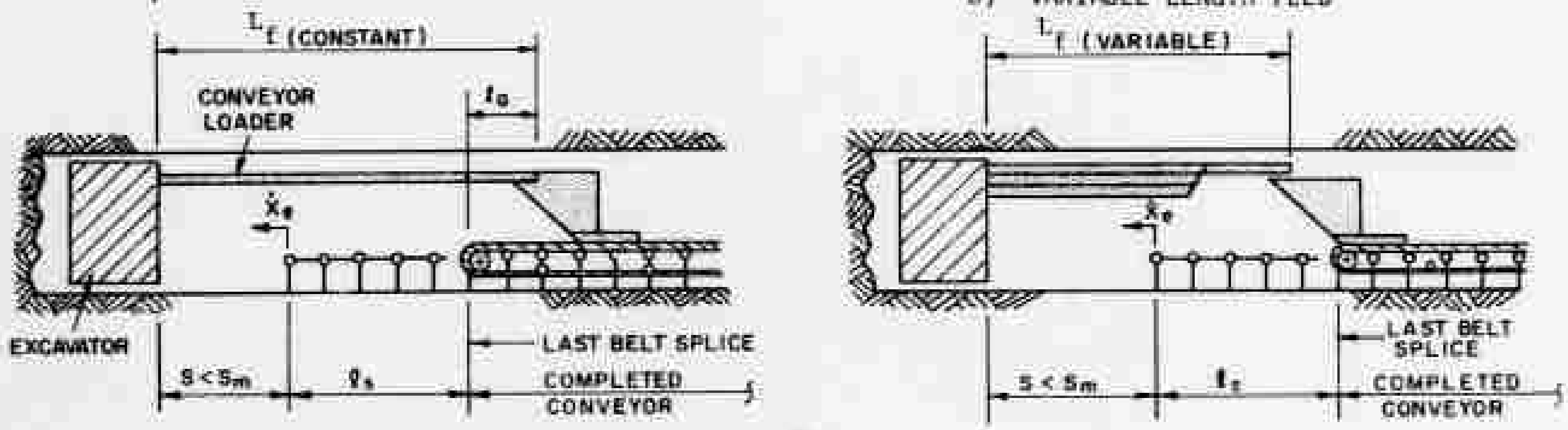
Subroutine EXTNSN extends the conveyor system accounting for extension of the supporting structure, and the conveyor belt, including splices.

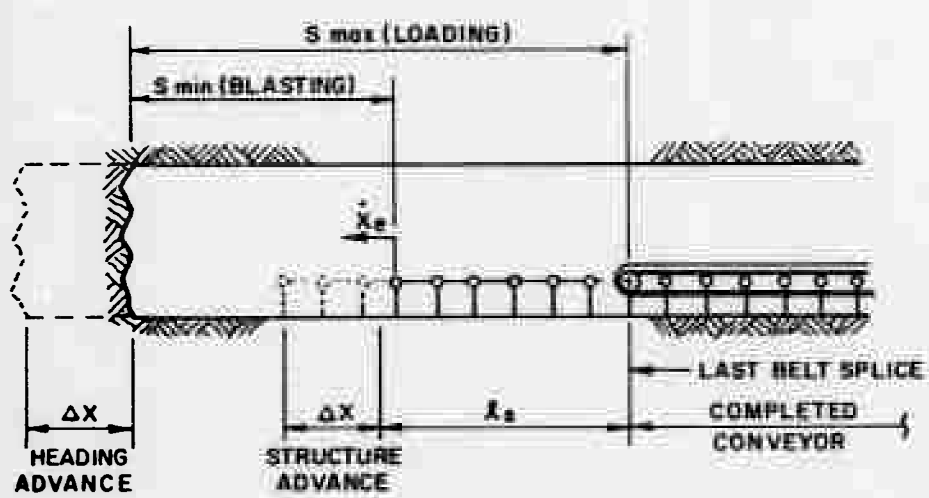
Besides conveyor performance and cost parameters, inputs to the program include the time increment, position of the face; the face advance during the last time step; the length of operational conveyor; and length of conveyor structure installed beyond the last bell splice. For a given time interval, outputs include updated length of operational conveyor and conveyor structure; as well as plant and equipment and direct labor cost.

Typical extension geometries modeled are shown in Fig. 52. The first two cases illustrated apply to continuous excavation processes where the conveyor structure is assembled in a nearly continuous manner as excavation proceeds. The last applies to cyclic excavation processes such as drill and blast where the conveyor structure is assembled in cycles.

Continuous excavation processes employ a mobile feeder conveyor attached to and moving with the excavator. This feeder conveyor, either extensible (Fig. 52b) or constant in length (Fig. 52a), would permit uninterrupted muck loading. The feeder mechanism would consist of a belt conveyor mounted on a structural framework. This framework would straddle the completed main-line conveyor and move along on wheels with the excavator. The feeder conveyor would take the muck from the excavator and feed it onto the completed main line conveyor belt through a transfer chute. The feeder conveyor could also be offset from and feed the conveyor through a cross-feed conveyor or chute. In either case, the mobile feeder conveyor would be of sufficient length to allow working space and time for installation of additional segments of conveyor

CONTINUOUS EXCAVATION





$\dot{X}_e$ - STRUCTURE EXTENSION RATE (ft/hr)

ℓ_a - AVAILABLE CONVEYOR AHEAD OF LOADER (ft)

ℓ_s - CONVEYOR STRUCTURE EXTENSION (ft)

S_m - MAXIMUM PERMITTED EXCAVATOR RUNAWAY (ft)

L_f - LENGTH OF FEEDER CONVEYOR (ft)

Figure 52. Typical Conveyor Extension Geometries Modeled

structure (ℓ_s), assembled directly in the near face area or inserted as preassembled units, while excavation remains in progress. When the length of the new conveyor structure equals the feeder length, excavation and muck loading would have to be stopped to permit the splicing of an additional length of belt onto the existing one. As can be seen, this extension technique allows a continuous face advance but also requires a continuous construction effort to extend the conveyor structure. Nevertheless, the process is not completely continuous since a complete shutdown is still necessary for belt splices.

In modeling continuous excavation, an interesting feature of subroutine EXTNSN is that it accounts for the interaction between the excavation and conveyor extension operations. For example if the excavator advance rate exceeds the conveyor structure extension rate, over a period of time the excavator will run away from the conveyor work. To prevent this the user may input a runaway tolerance (i.e., a maximum tolerated value of S in Fig. 52) which if exceeded will cause excavation to shut down until the conveyor structure can catch up.* One can see how the utilization of the excavation system (Fig. 14) would be adversely affected if this should occur frequently. Indeed this would indicate that quicker conveyor extension methods must be developed.

Although not shown directly in Fig. 52, continuous excavation processes with cyclic extension could also be modeled. Such a procedure is presently being used in the experimental program sponsored by the Copper Range Company in White Pine, Michigan.** It can be

---

* If the excavator advance rate is less than the conveyor extension rate such that, over a period of time, S (Fig. 52) should go to zero, the conveyor extension rate would simply be set equal to the excavator rate until such time as S becomes greater than zero.

** The White Pine research program includes a boring machine with a 300-ft feeder conveyor, and a 36-in. belt main-line conveyor. The tunnel bored is 18 ft in diameter.

illustrated using Fig. 52a. Initially the excavation would begin with the feeder conveyor a distance ℓ_f along the available conveyor (i.e., $\ell_f = \ell_a$, $\ell_s = 0$, $S = 0$ in Fig. 52a). Excavation would then proceed but not conveyor extension.

When the length of the available conveyor ahead of the feeder becomes zero ($\ell_a = 0$), excavation would cease completely, and conveyor extension would commence until a new conveyor segment and belt splice is completed (i.e., $\ell_f = \ell_a$ again). Because this procedure is cyclic, the time advantage of parallel excavation and extension is lost.

If the excavation process is drill and blast, the model also allows the conveyor structure to be advanced in cycles, but with the structure advance equal to the heading advance for the given round. In this case, a minimum distance of the structure from the face is maintained for blasting (S_{min} in Fig. 52c). Also, if the user-specified maximum permitted distance of the face from the available conveyor (S_{max}) is exceeded, a belt splice is made and a new conveyor segment becomes available for loading.

b. Performance Model

The major parameters defining conveyor extension performance in EXTNSN are the conveyor structure extension rate, and the time required for a belt splice.

The conveyor structure extension rate, $\dot{X}_e$, depends on a number of factors including the characteristics of the conveyor and supporting structure; how the structure itself is supported (i.e., floor, wall, or roof); the assembly technique employed (i.e., whether the framing members and rollers are assembled and installed directly in the near-face zone, or inserted as a preassembled unit); the size of the assembly crew; and acceptable levels on misalignments and other similar considerations that might impact on belt speed, belt wear, and general maintenance. Presently, no generalized guidelines are available.

Also the time required for a belt splice will depend on the type of splice (mechanical or vulcanized), the width and thickness of the belt, etc. Experience at the Copper Range Mine* indicates a time of the order of 2 hours for a 36-in.-wide belt using a mechanical splice.

As discussed previously, conveyor structure extension is either cyclic or continuous (Fig. 52) depending upon the nature of the excavation process and system extension technique employed. For cyclic operations, the length of added structure is taken equal to the heading advance for the round and the time for structure extension computed as

$$\Delta t_e = \Delta X_r / \dot{X}_e$$

where ΔX_r = length of round (ft)

$\dot{X}_e$ = average structure extension rate (ft/hr)

The program is then executed for successive time intervals until Δt_e is exceeded, at which time the conveyor structure length (ℓ_s) is updated by ΔX_r. That is,

$$\ell_s = (\ell_s)_0 + \Delta X_r \qquad \text{(ft)}$$

In the case of continuous processes, the conveyor structure completed beyond the last belt splice; the available conveyor ahead of the feeder (for constant length feeders only--see Fig. 52a); the length of the feeder (for extensible feeders only--see Fig. 52b); and the distance between the conveyor structure and the excavator (ℓ_s, ℓ_a, ℓ_f, and S, respectively in Fig. 52) are updated using

$$\ell_s = (\ell_s)_0 + \dot{X}_e \Delta t \qquad \text{(ft)}$$

*See second footnote two pages earlier.

$$\ell_a = (\ell_a)_0 - \Delta X_E \qquad \text{(constant length feeder only) (ft)}$$

$$\ell_f = (\ell_f)_0 + \Delta X_E \qquad \text{(extensible feeder only) (ft)}$$

$$S = \ell_f - \ell_s - \ell_a \qquad S_m \geq S \geq 0$$

where Δt = time interval (hr)

ΔX_E = excavator advance in Δt (ft)

$\dot{X}_e$ = conveyor structure extension rate (ft/hr)

S_m = maximum tolerated excavator runaway (ft)

and $(\ell_s)_0$, $(\ell_a)_0$, $(\ell_f)_0$ = values of the respective parameter at beginning of interval.

If the value of S is less than zero (i.e., a case where the conveyor structure has caught up to the excavator), then S is simply equated to zero for that interval and ℓ_s is readjusted to equal the excavator advance for the interval; or

$$\ell_s = (\ell_s)_0 + \Delta X_E$$

If the value of S is greater than S_m (i.e., the excavator is running away from the conveyor), excavation is shut down for the succeeding time intervals until $S < S_m$.

Once the above parameters have been updated, the program checks to see if a belt splice is required. The criterion that determines this depends on the configuration modeled. For cyclic processes (Fig. 52c) a splice is needed if

$$\ell_s \geq S_{max} - S_{min}$$

for continuous processes with a constant length feeder (Fig. 52a) if

$$\ell_a \leq 0$$

and for continuous processes with an extensible feeder (Fig. 52b) if

$$\ell_f \geq \ell_{fm}$$

where S_{max} = user input maximum distance between face and available conveyor (cyclic processes only)(ft)

S_{min} = user input minimum distance between face and conveyor structure (cyclic processes only) (ft)

ℓ_{fm} = user input maximum extended feeder length (extensible feeder case only) (ft)

If a belt splice is required, the conveyor system will be shut down, but excavation and/or loading will continue if the user has provided a surge bin (see Sec. IV-D-4) and it is not filled at that time. If no surge bin has been provided, or the available surge bin has been filled (or becomes filled during splicing), excavation or loading will also cease during belt splicing.

In any case, once the conveyor is shut down for splicing, a period of time will elapse to allow the conveyor structure to catch up to the excavator, and also to make the splice. This time is computed by

$$\Delta t_e = (\ell_f - \ell_a - \ell_s)/\dot{X}_e + \Delta t_{bs} \qquad \text{(hr)}$$

where Δt_{bs} = user input time to make a belt splice (hr).

The program maintains a counter which sums the intervals each time EXTNSN is called until Δt_e is exceeded.

c. Cost Information

The cost categories applying to subroutine EXTNSN are plant and equipment and direct labor.

Plant and equipment are the ownership costs of the conveyor including supporting structure, idler and return rollers, the conveyor belt, transfer equipment, drive equipment, etc. The user can input these costs, prorated on a per foot basis (see Table 31), if he selects to input his own conveyor characteristics, i.e., items 1-5 in Table 31. If he selects the design option, item 6 in Table 31 (see subroutine TRNSPRT), he can exercise the design segment of the program independently and then provide costs to the model based on the conveyor characteristics determined. As an aid, the following equations (taken from Ref. 47) may help serve as a guideline.*

$$\text{BELT COST} = (0.027 + 0.015\, w_b)\, w_b/2 \qquad (\$/\text{ft})$$

$$\text{END EQUIPMENT} = 246\, \frac{HP_f^{\,0.8064}}{L_f} \qquad (\$/\text{ft})$$

$$\text{IDLER ROLLERS} = \left[2.12 + 0.885\left(w_b - 8\right)\right]\left(\frac{1}{\ell_i}\right) + \frac{200\, W_b}{\ell_c} \qquad (\$/\text{ft})$$

$$\text{RETURN ROLLERS} = \left[10.6 + 0.505\, w_b\right]\left(\frac{1}{\ell_r}\right) \qquad (\$/\text{ft})$$

$$\text{SUPPORTING STRUCTURE} = 0.20\, w_b \qquad (\$/\text{ft})$$

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*These represent initial capital investment costs (1970), and do not include an allowance for interest, insurance, taxes, major overhauls, and repairs, etc.

where the parameters are as defined in the previous section. Direct labor is the cost of labor required to erect the conveyor. This may include laborers, electricians, etc., and is provided as user inputs in Table 31.

4. Subroutine SURGE

Subroutine SURGE accumulates and discharges muck to and from a surge bin. If the user desires a surge bin in the conveyor system, he simply inputs its volume capacity under item 7 in Table 31.

The program has two entry points: one for muck addition, and one for muck removal. It accepts the incremental volume to be added or discharged from the calling program, and keeps track of the amount of muck in the bin at any given time. It also checks to see if this amount exceeds the bin capacity. If so, the subroutine returns a flag indicating the bin is full to the calling program. This in turn causes the loader (and excavator in the case of continuous processes) to be turned off on the next time increment.

5. Subroutine BELT

Subroutine BELT accounts for conveyor repair and maintenance periods. The user provides either the availability of the conveyor or the average down time per maintenance period, and also the time between maintenance periods (item 8, Table 31).

During maintenance, excavating and loading activities are stopped, and a counter sums the intervals each time the subroutine is called, until the down time for maintenance is completed.

Costs associated with subroutine BELT include job materials for minor servicing and repair (prorated per maintenance period) and the direct labor for the maintenance crew. These are provided for as user inputs in Table 31.

E. FACE TO MAIN-LINE TRANSPORT

1. Machine Loaders and Shovels

a. Introduction

Loaders and shovels (i.e., mucking machines) are used in conventional drill and blast excavation to pick up broken rock at the face of the tunnel and load it onto a long haul materials handling system such as trucks or trains. They are used only for mucking and must be moved back and forth away from the face during other operations of the drill and blast cycle.

Mucking machines are available in a variety of sizes and capacities. Table 36 presents representative data on the horsepower, dimensions, and capacities of machines presently available. In smaller tunnels up to 20 ft, the overshot loader is used almost exclusively. The overshot loader (e.g., EIMCO 631, etc.) which can be either air, diesel or electric powered, is a very compact unit and is either rail, crawler track, or rubber tire mounted. It loads by crowding a bucket into the muck pile, shoveling it up, and throwing it back over the rear of the machine. It is designed to swing in a radius of approximately 120° and is extremely maneuverable and fast loading. The material thrown over the back of the unit is generally loaded either onto a conveyor belt or into a transfer unit, which could be a belt or a scraper, and then into rail or rubber-tired cars.

A unit slightly larger in capacity is the Conway electrically operated mucker (e.g., CONWAY 100-1) which is equipped with a tilting dipper. The unit crowds into the muck pile in a manner similar to the overshot loader, but by making use of a tilting bucket which pivots on the forward crowding boom, it dumps its load back onto a pan which in turn feeds onto a conveyor belt. The belt then carries material back, up, and over the mucker either into cars or onto a conveyor belt.

TABLE 36

REPRESENTATIVE MUCKING MACHINES*

	Bucket Size	Power	Type of Power	Tramming Width	Relation of Cleanup Width, Operating Height, Tramming Height, And Track Gauge: Cleanup Width	Heights Above Top of Rail: Tramming Height	Heights Above Top of Rail: Operating Height	Minimum Gauge	Belt Width	Minimum-Radius Curve	Weight	Maximum Car Height	Maximum Car Length
RAIL-MOUNTED MUCKING ON RUBBER BELTS													
Conway 100-2	1 1/2 cu. yd.	1-125 hp 1-40 hp	Electric	6 ft. 2 in.	17 ft. 8 in.	8 ft. 2 1/2 in.	12 ft. 0 in.	30 in.	42 in.	25 ft.	55,000 lb.		
					16 ft. 6 in.	8 ft. 6 1/2 in.	12 ft. 9 in.	30 in.					
					20 ft. 0 in.	9 ft. 3 1/4 in.	13 ft. 5 in.	36 in.					
					21 ft. 6 in.	9 ft. 10 1/2 in.	14 ft. 5 in.	39 3/8 in.					
					22 ft. 4 in.	9 ft. 10 1/2 in.	14 ft. 8 in.	39 3/8 in.					
Conway 100-1	1 1/4 cu. yd.	1-100 hp 1-30 hp	Electric	6 ft. 2 in.	23 ft. 0 in.	10 ft. 3 in.	15 ft. 0 in.	42 in.					
					16 ft. 6 in.	8 ft. 2 in.	11 ft. 6 1/2 in.	30 in.	38 in.	20 ft.	50,800 lb.		
					18 ft. 0 in.	8 ft. 7 in.	12 ft. 6 in.	30 in.					
					19 ft. 6 in.	9 ft. 4 in.	13 ft. 4 in.	36 in.					
					11 ft. 0 in.	8 ft. 10 in.	16 ft. 3 in.	39 3/8 in.					
					21 ft. 6 in.	9 ft. 10 in.	16 ft. 3 in.	39 3/8 in.					
Conway 75	3/4 cu. yd	1-75 hp 1-25 hp	Electric	5 ft. 5 1/2 in.	13 ft. 0 in.	10 ft. 8 1/2 in.	15 ft. 1 in.	42 in.					
					13 ft. 0 in.	7 ft. 6 3/4 in.	8 ft. 7 in	24 in.	28 in.		42,000 lb.		
					15 ft. 0 in.	7 ft. 6 3/4 in.	10 ft. 7 3/4 in.	24 in.					
					17 ft. 6 in.	8 ft. 1 in.	11 ft. 9 in.	30 in.					
					19 ft. 0 in.	9 ft. 1 in.	12 ft. 11 in.	36 in.					
					22 ft. 0 in.	10 ft. 6 3/4 in.	14 ft. 14 in.	36 in.					
Eimco 40 B	1/2 cu. yd.	635 cfm	Air or Electric	6 ft. 0 in.	12 ft.	5 ft. 10 in. 6 ft. 10 in.		28 1/2 in.			19,000 lb.	4 ft. 9 in.	9 ft. 6 in.
Eimco 40 W	1/2 cu. yd.	635 cfm	Air or Electric	6 ft. 0 in.	15 ft.	5 ft. 10 in. 6 ft. 10 in.		28 1/2 in.			19,300 lb.	7 ft. 2 in.	15 ft. 0 in.
RAIL-MOUNTED MUCKING INTO HAUL UNIT													
Eimco 12 B	4 1/2-6 cu. ft.		Air	34 in.	75-83 in.	48-51 in.	78 1/2-83 in.	15 in.			4,200 lb.	46-59 in.	
Eimco 22	7 1/2-10 cu. ft.		Air	44 3/16 in.	90-110 in.	60-63 1/4 in.	88-99 in.	18 in.			7,100 lb.	50-64 in.	
Eimco 25	8-13 1/2 cu. ft.		Air	63 7/8 in.	96-118 in.	52-72 in.	91-111 in.	18 in.			12,000 lb.	52-72 in.	
Atlas Copco													
LM 56	8.2		Air	41 in.	112 in.	55-58 in.	88-96 in.	18 in.			6,500 lb.	49-57 in.	
LM 250	110		Air	57 in.	150 in.	76-80 in.	114-125 in.	24 in.			16,200 lb.	69-[illegible] in.	
CRAWLER-MOUNTED MUCKING INTO STEEL FLIGHT CONVEYOR BELT													
Eimco 635	12 1/2 cu. ft.		Air or Electric	6 ft. 0 in.		4 ft. 10 in. 8 ft. 8 in.	5 ft. 0 in. 8 ft 8 in.	45 in.			15,000 lb.	7 ft.	11 ft.
CRAWLER-MOUNTED MUCKING INTO HAUL UNIT													
Eimco 611	12 1/2 cu. ft.		Air	5 ft. 8 5/8 in.		6 ft. 5 in.	11 ft. 2 3/8 in.				13,000 lb.	8 ft.	
Eimco 630 E	8-14 cu. ft.		Electric	5 ft. 8 5/8 in.		4 ft. 11 1/2 in.	6 ft. 8 in.				11,300 lb.	3 ft. 9 in.	
Eimco 105 or 115	1 1/2 cu. yd	143 hp	Diesel	7 ft. 8 in.		9 ft. 6 in.	10 ft. 7 in. 16 ft. 0 in.				42,500 lb.	7 ft. 2 in. 9 ft. 6 in.	
RUBBER-TIRED MUCKING INTO HAUL UNIT													
Atlas Copco													
CAVO 310	6	282 cfm	Air	75 in.		57 in.	83 in.				6,720 lb.	18 in.	
320	11	282 cfm	Air	82 in.		66 in.	105 in.				8,620 lb.	66 in.	
511	18	530 cfm	Air	93 in.		69 in.	106 in.				13,220 lb.	16 in.	
320	21	530 cfm	Air	89 in.		79 in.	131 in.				14,630 lb.	88 in.	

*Information abstracted from reference, and catalogues furnished by The Eimco Corporation, and Atlas Copco

Another type of loader which has undergone considerable development is the scraper or grab-type loader. This unit has been used more in the European tunnels and on excavations in foreign countries than in the United States. A similar unit is built by Wemco* in the United States. It operates by scraping the material back from the face onto a conveyor belt or pan feeder by means of grab arms which are usually hydraulically powered. A "duckbill" loader made by Joy** is often used in loading slabby material such as shale or coal. This is an electric loader with an apron which is crowded into the muck pile. Two arms, by alternate movements, rake the muck up the apron onto a conveyor belt. Some units have slusher type buckets mounted on a forward boom which brings the scraper forward, drops it into place; and then either by cable or hydraulic arms, scrapes the muck back onto the feeder.[47]

In very large tunnels or underground excavations, the conventional swing type shovel is used. This unit is adaptable only to large excavations and does not have the speed of operation of the other type loaders. However, where reliability is important and the speed of loading is secondary, this unit may have potential application.

The activities associated with mucking machines including subroutine names used in the model, are summarized in Table 37. Also the input specification sheet including performance and cost data are given in Table 38.

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* Wemco Division of Envirotech Corporation, Salt Lake City, Utah.

** Joy Manufacturing Company, Pittsburgh, Pennsylvania.

TABLE 37

SUMMARY OF MACHINE LOADER OR SHOVEL ACTIVITIES

Subroutine MUKLOD

Loads muck at face

Subroutine MUKIN

Moves muckers to face

Subroutine MUKOUT

Moves muckers from face

Subroutine MUKMT

Maintenance and repair

TABLE 38

MACHINE LOADER OR SHOVEL PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

1. Average mucking rate (ft^3/hr)__________
2. Time to move to face and back[e] (hr)__________
3. For loading unitized main line systems time to change from empty to loaded car (hr)[e]______
4. Maintenance parameters
   (a) Unit availability__________
   (b) In lieu of (4-a) down time for maintenance cycle (hr)__________
   (c) Time between maintenance periods (days)___

[e] See footnote p. 179

1. Plant and Equipment

Item	Ownership (Rental) Cost Per	Hr (check one)	Ft	Unusual Developmental Cost Per	Hr (check one)	Ft
MAJOR ITEMS:						
Machine Unit						
Accessories						
ADDITIONAL ITEMS:						

2. Job Materials

Item	Cost		Lifetime	
	Value	Unit	Value	Unit
MAJOR ITEMS:				
Servicing and Repair (per maintenance period)		$/Maint.	-	-
ADDITIONAL ITEMS:		$/ft		

3. Direct Labor

Labor Type	Number Required/Shift	Rate $/hr
MAJOR TYPES:		
Mucker Operator		
Cable Tender		
Miner on Face		
Car Switching		
Maintenance Foreman		
Mechanic		
Electrician		
ADDITIONAL TYPES:		

4. Permanent Materials

Item	Cost	
	Value	Unit
(None Required)		

b. Subroutine MUKLOD

Subroutine MUKLOD simulates the loading of muck onto either a unitized or continuous main line system. In addition to system performance and cost parameters, inputs to the program include the time increment; the current volume of muck in the muck pile (including characteristics--i.e., density and swell factor); and for a unitized main-line system (trains or trucks), the unit capacities and the state vector of units available in the loading queue area (see Fig. 43 and Sec. IV-B). In addition to direct labor costs, outputs from the program include: for a unitized main-line system, the volume of muck loaded during the time increment; an updated loading queue vector with units available for loading, and an updated state vector for units filled in the time increment (i.e., if filled these are released and allowed to begin accelerating to the discharge area--see Sec. IV-B). For continuous main line systems, outputs include the volume rate of material loaded in the time increment.

The major loading performance parameters required in subroutine MUKLOD are the volume mucking rate of the machine ($\dot{Q}_{mr}$); and, for unitized main line systems, the time required to switch a loaded unit (muck car or truck) for an empty one. Unless input (see item 3, Table 38) unit switching time is taken at 3 min/unit.

Mucker input rates can depend on a number of important factors including: machine design--e.g., bucket capacity, loading cycle time, angle of swing, etc.; the shape of the muckpile--e.g., the length of the throw, etc., impacts on machine movement and clean up requirements; the characteristics of the muck--e.g., cohesiveness, size distribution, etc.; operator proficiency using the machine; scaling and scraping at the face required to disengage loose rocks, etc. Manufacturers of mucking equipment generally provide average mucking rate guidelines in their catelogues and specification sheets. (See, for example, Fig. 53 for the CAVO 310 and 511 rubber wheel mounted muckers.) These can be used as an aid in determining inputs to the model.

Capacity and haulage distance

The diagram shows the Cavo autoloader capacity as a function of transport distance. The curves indicate **average values** and both higher and lower values may be obtained depending on the loading and haulage conditions.

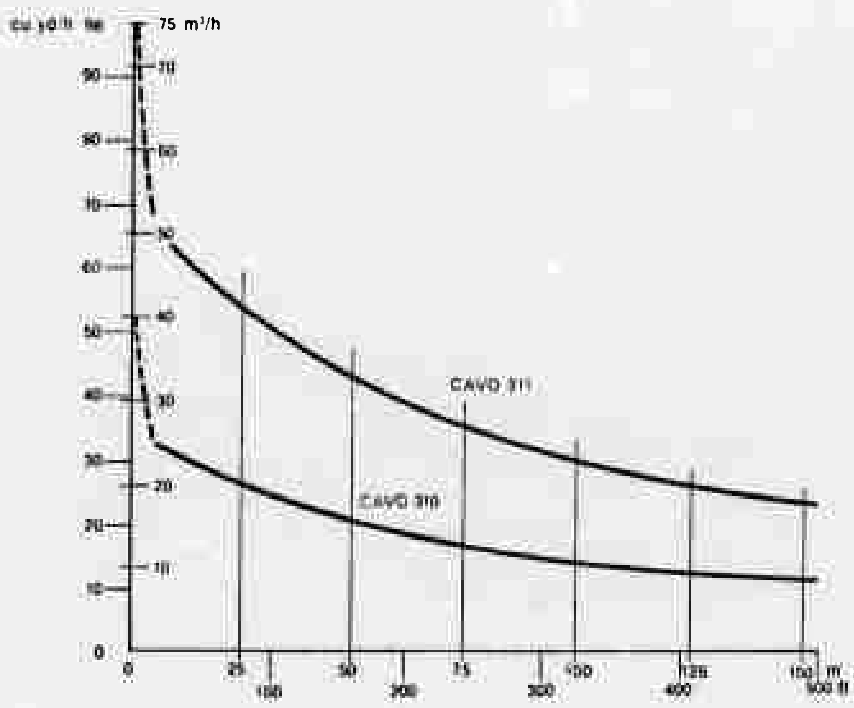

Figure 53. Example of Manufacturer Suggested Mucking Rates for the CAVO 310 and 511 Mucking Units

The major cost information used in the program is the ownership cost of the mucking machine including accessories such as cables, air hoses, compressors, etc.; and direct labor cost for the mucker operator, cable tender, car switchers, etc. These are provided for in Table 38 as user inputs.

The sequence of calculations followed in subroutine MUKLOD in loading a muck pile, initially of volume $(\Delta Q_t)_0$, into a unitized main line system is as follows: over a series of time increments,* if there is an empty or partially filled unit available for loading at the face**

$$\Delta Q = \dot{Q}_{mr}\,\Delta t$$

$$\Delta Q_t = \left[\left(\Delta Q_t\right)_0 - \sum_{\Delta t} \Delta Q\right] > 0$$

$$Q_{uf} = \left[\sum_{\Delta t} \Delta Q\right] \leq Q_{uc}$$

$$W_{uf} = \left[\sum_{\Delta t} \frac{1}{2000}\left(\frac{\rho_s}{K}\right)\Delta Q\right] \leq W_{uc}$$

where Δt = time interval (hr)

$\dot{Q}_{mr}$ = mucker loading rate (ft^3/hr)

ΔQ = volume of muck loaded in time interval (ft^3)

ΔQ_t = volume of muck in pile at end of interval (ft^3)

Q_{uf} = total volume of muck loaded into unit at the end of interval (ft^3)

W_{uf} = total weight of muck loaded into unit at the end of interval (tons)

---

* A typical mucking cycle may contain several simulation time increments.

** If no empty units are available, loading stops until one moves into the loading area.

Q_{uc} = unit volume capacity (ft^3)

W_{uc} = unit weight capacity (tons)

ρ_s = in situ muck density (lb/ft)

K = swell factor, assumed 1.7 unless input

If during a given time increment $\Delta Q_t \leq 0$, the muck pile has been completely loaded, and a flag indicating the end of mucking is returned by the subroutine.

If $Q_{uf} \geq Q_{uc}$ or $W_{uf} \geq W_{uc}$, a single unit (muck car or truck) has been loaded, and must be replaced by an empty one. In this case, a counter keeps track of the delay (3 min if not user input) for unit switching before loading begins again. If the filled unit is a truck, the program then releases it. If the unit is a muck car in a train, the program waits until all muck cars are filled before releasing the train.

For continuous main-line systems the program loads a muck pile over a series of time increments using

$$\Delta Q = \begin{cases} \dot{Q}_{mr}\,\Delta t \\ \Delta Q_t \end{cases} \quad \text{(the smaller value)}$$

$$\Delta Q_t = \left[\left(\Delta Q_t\right)_0 - \sum_{\Delta t} \Delta Q\right] > 0$$

$$\dot{Q}_L = \Delta Q/\Delta t$$

where the parameters are as defined above with the addition

$\dot{Q}_L$ = loading rate onto the continuous system (ft^3/hr)

c. Subroutine MUKIN

This subroutine accounts for the time required to move the muckers to a muck pile at the face of the tunnel. Unless user input, this delay is taken at 5 min.

d. Subroutine MUKOUT

This subroutine accounts for the time required to move the muckers away from the face after mucking operations. Unless user input, this delay is taken at 5 min.

e. Subroutine MUKMT

Subroutine MUKMT accounts for mucking machine maintenance. The user provides either a unit availability or down time per maintenance, and also the time between maintenance periods (item 4, Table 38). He also provides job materials (prorated per maintenance) for servicing and repairs and the labor crew required (see cost information, Table 38). During each maintenance, a counter sums the time intervals each time the subroutine is called until maintenance is completed. The muckers are then released for loading.

2. Integrated Conveyor Loader

a. Introduction

The loading equipment used with continuous-type excavation machines is usually designed as an integral part of the machine. The equipment consists of some loading device at the face of the equipment which picks up the broken rock and deposits it onto a transfer conveyor. This conveyor in turn carries the rock through the machine and loads it onto a long haul muck handling system at its rear. Such a loading system normally operates continuously as the excavation machine advances.

For a boring machine, the loading device at the face generally falls into three categories: a rotary bucket elevator, a chain bucket elevator, or a scraper or hole-type loader.[47] The most commonly used is the rotary bucket elevator which is attached to the boring head and rotates with the bits. These elevator buckets are rigidly attached to the head and scrape up the material from the bottom and sides of the tunnel as the head rotates, carrying it to the top where they are inverted and deposit their load onto the transfer conveyor.

A second type of bucket elevator is the chain-bucket type (also used on continuous miners in the coal industry). These units have the capability of cleaning a flat surface on the bottom of the tunnel as it is required for the bottom of a horseshoe tunnel. The buckets move from side to side against the heading, and carry the material to the top side of the boring machine by a chain driven on a sprocket and gear arrangement. They then are inverted and dump their material onto a transfer conveyor.

The third type of loader used with the mechanical moles has been developed in the European countries where the multiple head boring machine is used. This unit usually consists of several rotary heads driven by independent motors. The entire configuration is normally mounted on a central shaft and is rotated at a slower rate than the cutter heads in such a manner as to cover the entire face desired for excavation. With this type of unit, a scraper or hoe-type loader is sometimes applied. This unit consists of hydraulically operated scrapers or hoes which pull the muck back from the face and deposit it onto the transfer conveyor.

The transfer conveyor is usually a troughed-belt conveyor which transports the muck through the machine and onto another conveyor section which has been designed to load the main-line system. This last conveyor section would be configured to accommodate the specific main line system type (i.e., see the feeder conveyor used for continuous main line systems in Fig. 52, Sec. IV-D; and also the train-loading conveyor configuration for the Layout Tunnel in Vol. II.

The activities associated with the integrated conveyor loader including subroutine names are:

1. <u>Subroutine CVLOAD</u>

   Loads main line muck handling system

2. <u>Subroutine CVLMT</u>

   Maintenance and repair

The input specification sheet including performance and cost data are given in Table 39.

b. Subroutine CVLOAD

Subroutine CVLOAD loads muck obtained from a continuous excavator onto the main line system. Inputs to the program include the simulation time increment, the volume of muck excavated in that time (including characteristics--i.e., density and swell factor); and, for a unitized main line system (trains or trucks) the unit capacities and the state vector of units available in the loading queue area. For unitized main line systems the outputs include: volume of muck loaded in the time increment; an updated loading queue vector with units available for loading; and an updated state vector for units filled in the time increment (i.e., if filled these are released and allowed to begin accelerating to the discharge area--see Sec. IV-B and IV-C). For continuous main line systems, outputs include the volume rate of material loaded in the time increment.

The loading parameters defining the peak mucking rate of the conveyor loader in tons per hour can be input directly using item 3, Table 39; or the muck flow cross section (A_b in square feet) and speed of the conveyor belt (v_b in feet per minute) can be input (items 1 and 2) in which case the peak mucking rate is computed using

$$\dot{W}_{mr} = \left(\frac{60}{2000}\right) A_b v_b \left(\frac{\rho_s}{K}\right) \qquad \text{(ton/hr)}$$

TABLE 39

INTEGRATED-CONVEYOR LOADER PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

1. Muck flow cross section (ft^2)__________
2. Belt speed (ft/min)__________
3. In lieu of (1 and 2)
   Maximum mucking rate (ton/hr)__________
4. Maintenance parameters
   (a) Unit availability__________
   (b) In lieu of (4-a) down time per malfunction cycle (hr)__________
   (c) Time between maintenance periods (days)____

1. Plant & Equipment

Item	Ownership (Rental) Cost Per	Hr	Ft (check one)	Unusual Developmental Cost Per	Hr	Ft (check one)
MAJOR ITEMS:						
Conveyor Equipment						
Supporting Structure						
ADDITIONAL ITEMS:						

2. Job Materials

Item	Cost Value	Unit	Lifetime Value	Unit
MAJOR ITEMS:				
Servicing and Repair		\$/Maint.	-	-
ADDITIONAL ITEMS:				

3. Direct Labor

Labor Type	Number Requ'red/Shift	Rate \$/hr
MAJOR TYPES:		
Operator		
Maintenance		
Foreman		
Mechanics		
Electricians		
ADDITIONAL TYPES:		

4. Permanent Materials

Item	Cost Value	Unit
(None Required)		

where ρ_s = in situ density of muck (lb/ft^3)

K = swell factor, assumed 1.7 unless input

The volume mucking rate is also computed using

$$\dot{Q}_{mr} = 2000 \left(\frac{K}{\rho_s}\right) \dot{W}_{mr} \qquad (ft^3/hr)$$

if $\dot{W}_{mr}$ is input, or

$$\dot{Q}_{mr} = 60\ A_b v_b \qquad (ft^3/hr)$$

if A_b and v_b are input.

The computational sequence followed in subroutine CVLOAD for loading unitized main line systems is as follows: the volume rate ($\dot{Q}$) and tonnage rate ($\dot{W}$) of muck to be loaded in the time increment is computed as

$$\dot{Q} = \left[\Delta Q_E/\Delta t\right] \leq \dot{Q}_{mr} \qquad (ft^3/hr)$$

$$\dot{W} = \left[\Delta Q_E \left(\frac{\rho_s}{K}\right)\left(\frac{1}{2000}\right)\right] \leq \dot{W}_{mr} \qquad (ton/hr)$$

where the parameters are as defined above, but including

Δt = time increment (hr)

ΔQ_E = excavated bulk volume in Δt (ft^3)

If $\dot{Q} > \dot{Q}_{mr}$ or $\dot{W} > \dot{W}_{mr}$, the loading rate of the conveyor has been exceeded and the excavator must be turned off during the next time increment; otherwise the muck is loaded into the available unit using

$$Q_{uf} = \sum_{\Delta t} \dot{Q}\Delta t \leq Q_{uc}$$

$$W_{uf} = \sum_{\Delta t} \dot{W}\Delta t \le W_{uc}$$

where Q_{uf} = total volume capacity of the unit filled at the end of the interval (ft^3)

W_{uf} = total weight capacity of unit filled at end of the interval (tons)

Q_{uc} = volume capacity of unit (ft^3)*

W_{uc} = weight capacity of unit (tons)*

If $Q_{uf} > Q_{uc}$ or $W_{uf} > W_{uc}$ the unit has been loaded and is therefore released allowing it to proceed back to the discharge area.

In loading continuous main line processes, subroutine CVLOAD simply checks that $\dot{Q} \le \dot{Q}_{mr}$ or that $\dot{W} \le \dot{W}_{mr}$ (i.e., that the capacity of the transfer conveyor is not exceeded) and, if so, it sets

$$\dot{Q}_L = \dot{Q}$$

where $\dot{Q}_L$ is the volume rate loaded onto the main line system. If the capacity is exceeded, the excavator is turned off the next time increment.

c. Subroutine CVLMT

Subroutine CVLMT accounts for integrated loader maintenance and repair periods. The user provides a unit availability or down time per maintenance, and also the time between maintenance periods (item 4,

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*Q_{uc} and W_{uc} are the capacities of a single truck unit or a single train unit. In the train case, all cars are included.

Table 39). He also provides the job material costs (prorated per maintenance) for servicing and repairs, and the labor required (see cost information, Table 39).

If a maintenance or repair period occurs, excavation is shut down and a counter sums the intervals each time this program is called until the down period if equaled or exceeded. Excavation and loading is then allowed to resume.

V. OTHER EXCAVATION ELEMENTS

A. GROUND SUPPORT AND TUNNEL LINING

The purpose of primary ground support is to preserve and maintain the stability of the tunnel opening and reduce the hazard of falling rock during excavation. Secondary lining may be installed following this primary support for a number of reasons:

- To reinforce the primary support which may weaken with time or be subject to greater stress as the rock load gradually shifts
- To add strength to the tunnel, in the case of military use, to withstand nuclear burst shock effects and increase the hardness of the tunnel
- To add desirable geometric and surface characteristics to a tunnel (e.g., smooth, circular tunnels for water transport)
- To protect the primary support from rust and deterioration
- To control water seepage in the finished tunnel

The design of both primary support and secondary lining has some root in theory. There is, however, a limited understanding of the behavior of the rock medium surrounding the tunnel, and all theoretically derived design rules have been modified and influenced by intuition and experience to a degree where experience and sound engineering judgment are prerequisites to achieving a satisfactory design.

While better methods of designing tunnel support are clearly desirable, they can be achieved only by a better understanding of the behavior of the support system and the surrounding medium. Although significant progress is being made along these lines there is still a long way to go before a truly analytical system for designing support and lining is developed. This is particularly true of designing support and lining for military tunnels where dynamic loading must also be considered.

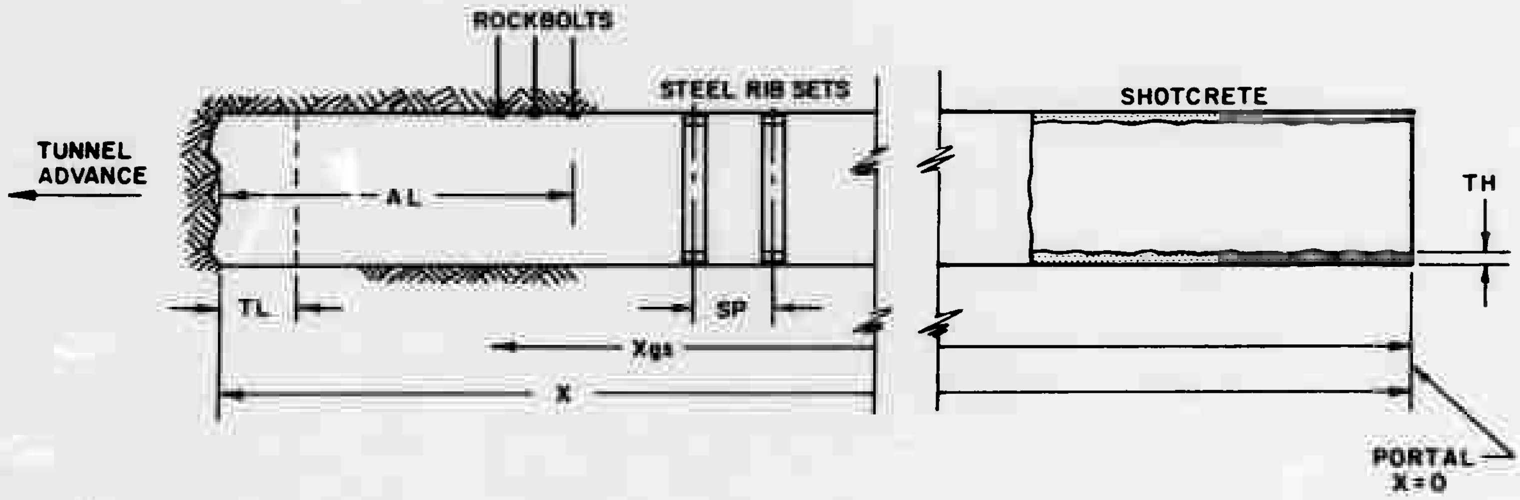
We have confined our modeling of ground support to a comparatively simply approach but one which we believe provides to the overall excavation model a reasonable representation of ground-support selection, design, and installation and one which will allow valid comparison between different excavation systems without ignoring the support requirements.

Three common types of primary support are considered in the present model--steel rib sets, rock bolts, and shotcrete--although the method of modeling the selection and installation of these can easily be used to model other types of support as well. Figure 54 illustrates the geometry of ground support used by the model for these three techniques.

The basis of ground-support design which is used by the model is <u>Design of Tunnel Liners and Support Systems</u>, by D. V. Deere et al.[52] Table 40 below is a summary of Deere's support recommendations for tunnels in rock. The reader should refer to the original and subsequent publications for a complete discussion of the origin simplifying assumptions and limitations of this table.[53,54]

The user input to the computer simulation for ground support is listed in Table 41. The example primary support selection table, item 5 within Table 41, shows the kind of information which causes the ground support system to respond to changes in geology as the tunneling proceeds. At the beginning of each time step of the simulation a selection of ground support type and amount is made based on this table. Then the following status checks are made:

- If $(X - X_{gs}) > AL$, rock fragmentation is suspended to allow ground-support installation to catch up
- If $(X - X_{gs}) < TL$, ground-support installation is suspended because of its interference with excavation
- If $TL \leq (X - X_{gs}) \leq AL$, ground-support installation proceeds at a rate set by user input



NOMENCLATURE:

X = length of excavated tunnel from portal to face, ft

Xgs = length from portal to last installed tunnel support, ft

AL = maximum allowable unsupported tunnel length, ft

TL = interference length: closest distance (ft) that ground support can be installed from the tunnel face. (Installation of ground support halts when (X-Xgs) < TL)

SP = steel rib spacing, ft

TH = shotcrete thickness, in.

Figure 54. Ground Support Geometry

TABLE 40

GUIDELINES FOR SELECTION OF PRIMARY SUPPORT FOR 20-FT TO 40-FT TUNNELS IN ROCK

Rock Quality	Construction Method	Alternative Support Systems: Steel Sets: Rock Load (B = Tunnel Width)	Steel Sets: Weight of Sets	Steel Sets: Spacing[c]	Rock Bolts[a] (Conditional use in poor and very poor rock): Spacing of Pattern Bolts	Rock Bolts: Additional Requirements and Anchorage Limitations[a]	Shotcrete[b] (Conditional use in poor and very poor rock): Total Thickness: Crown	Shotcrete: Total Thickness: Sides	Shotcrete: Additional Support[b]
Excellent[d] RQD > 90	Boring machine	(0.0 to 0.2)B	Light	None to occasional	None to occasional	Rare	None to occasional local application	None	None
	Drilling and blasting	(0.0 to 0.3)B	Light	None to occasional	None to occasional	Rare	None to occasional local application 2 to 3 in.	None	None
Good[d] RQD = 75 to 90	Boring machine	(0.0 to 0.4)B	Light	Occasional to 5 to 6 ft	Occasional to 5 to 6 ft	Occasional mesh and straps	Local application 2 to 3 in.	None	None
	Drilling and blasting	(0.3 to 0.6)B	Light	5 to 6 ft	5 to 6 ft	Occasional mesh or straps	Local application 2 to 3 in.	None	None
Fair RQD = 50 to 75	Boring machine	(0.4 to 1.0)B	Light to medium	5 to 6 ft	4 to 6 ft	Mesh and straps as required	2 to 4 in.	None	Provide for rock bolts
	Drilling and blasting	(0.6 to 1.3)B	Light to medium	4 to 5 ft	3 to 5 ft	Mesh and straps as required	4 in. or more	4 in. or more	Provide for rock bolts
Poor RQD = 25 to 50	Boring machine	(1.0 to 1.6)B	Medium circular	3 to 4 ft	3 to 5 ft	Anchorage may be hard to obtain. Considerable mesh and straps required.	4 to 6 in.	4 to 6 in.	Rock bolts as required (~4-6 ft cc.)
	Drilling and blasting	(1.3 to 2.0)B	Medium to heavy circular	2 to 4 ft	2 to 4 ft	Anchorage may be hard to obtain. Considerable mesh and straps required.	6 in. or more	6 in. or more	Rock bolts as required (~4-6 ft cc.)
Very poor RQD < 25 (Excluding squeezing and swelling ground)	Boring machine	(1.6 to 2.2)B	Medium to heavy circular	2 ft	2 to 4 ft	Anchorage may be impossible. 100 percent mesh and straps required.	6 in. or more on whole section		Medium sets as required
	Drilling and blasting	(2.0 to 2.8)B	Heavy circular	2 ft	3 ft	Anchorage may be impossible. 100 percent mesh and straps required.	6 in. or more on whole section		Medium to heavy sets as required
Very poor, squeezing or swelling ground	Both methods	up to 250 ft	Very heavy circular	2 ft	2 to 3 ft	Anchorage may be impossible. 100 percent mesh and straps required.	6 in. or more on whole section		Heavy sets as required

Note Table reflects 1969 technology in the United States. Groundwater conditions and the details of jointing and weathering should be considered in conjunction with these guidelines particularly in the poorer quality rock. See Deere et al (3) for discussion of use and limitations of the guidelines for specific situations.

[a]Bolt diameter = 1 in., length = $^1/_3$ to $^1/_4$ tunnel width. It may be difficult or impossible to obtain anchorage with mechanically anchored rock bolts in poor and very poor rock. Grouted anchors may also be unsatisfactory in very wet tunnels.

[b]Because shotcrete experience is limited, only general guidelines are given for support in the poorer quality rock.

[c]Lagging requirements for steel sets will usually be minimal in excellent rock and will range from up to 25 percent in good rock to 100 percent in very poor rock.

[d]In good and excellent quality rock, the support requirement will in general be minimal but will be dependent on joint geometry, tunnel diameter, and relative orientations of joints and tunnel.

TABLE 41

GROUND-SUPPORT PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

1. Rate of Installation of Rock Bolts (Bolts/hr) ______
2. Rate of Installation of Steel Sets (Sets/hr) ______
3. Rate of Installation of Shotcrete (yd^3/hr) ______
4. Excavation Process:
   Drill and Blast ( )
   Boring Machine ( )
   Novel Technique ( )
5. Primary Support Selection Table:

(Example)

RQD	EXCAVATION PROCESS	ALLOWABLE UNSUPPORTED LENGTH, AL, ft	INTERFERENCE LENGTH, TL, ft	SUPPORT TYPE	SPACING, ft	THICKNESS, in	ADDITIONAL SUPPORT
>95	Any	∞	0	None Req'd	---	---	---
90 - 95	Drill and Blast	10	0	Shotcrete	---	0.5 (Crown Only)	None
	Any Other	∞	0	None Req'd	---	---	---
75 - 90	Drill and Blast	5	0	Rock Bolts	5	---	None
	Boring Machine	8	4	Shotcrete	---	1.0 (Crown Only)	None
	Any Other	8	0	Shotcrete	---	1.0 (Crown Only)	None
50 - 75	Drill and Blast	4	0	Steel Ribs	4	---	None
	Boring Machine	6	4	Shotcrete	---	3.0 (Crown Only)	None
	Any Other	6	0	Shotcrete	---	3.0 (Crown Only)	None
25 - 50	Drill and Blast	3	0	Steel Ribs	3	---	Lagging
	Boring Machine	5	4	Shotcrete	---	5.0 (Crown and Sides)	Rock Bolts
	Any Other	5	0	Shotcrete	---	5.0 (Crown and Sides)	Rock Bolts
<25	Drill and Blast	2	0	Steel Ribs	2	---	Lagging
	Boring Machine	2	4	Steel Ribs	2	---	Lagging
	Any Other	2	0	Steel Ribs	2	---	Lagging

TABLE 41 (Cont.)

GROUND-SUPPORT PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

1. Plant & Equipment

Item	Ownership (Rental) Cost Per	Hr (check one)	Ft	Unusual Developmental Cost Per	Hr (check one)	Ft
MAJOR ITEMS:						
Rock Bolt Drills						
Shotcrete Plant						
ADDITIONAL ITEMS:						

2. Job Materials

Item	Cost		Lifetime	
	Value	Unit	Value	Unit
MAJOR ITEMS:				
Servicing and Repairs (per maintenance)			---	---
ADDITIONAL ITEMS:				

3. Direct Labor

Labor Type	Number Required/Shift	Rate $/hr
MAJOR TYPES:		
Installation Crew		
ADDITIONAL TYPES:		

4. Permanent Materials

Item	Cost	
	Value	Unit
Steel	$	lb
Shotcrete	$	yd^3

1. Rock Bolts

The assumptions that are made to enable modeling of rock bolt support are:

Rock bolts uniformly cover the circular arch of the tunnel roof

Bolt diameter equals 1 in.

Bolt length equals one-third the tunnel diameter

Bolt spacing is determined by the input table for the applicable RQD

If the assumed pattern of rock bolt installation is that shown in Fig. 55, the number of bolts per foot of tunnel length is:

$$\text{Bolts/Foot of Tunnel Length} = \frac{\pi D/2}{0.866\begin{pmatrix}\text{Bolt}\\ \text{Spacing}\end{pmatrix}^2}$$

Assuming that the bolt composition is AISI-SAE 1020 steel (or of similar density), the weight of steel in each rock bolt is

$$W = 0.284\pi D \qquad \text{(lb)}$$

which may be used to calculate the steel costs.

2. Shotcrete

The assumptions that are made to enable shotcrete support modeling are:

- Shotcrete is applied to an average thickness specified by the input table for the applicable RQD over the crown or crown and sides of a circular tunnel
- For an RQD between 25 and 50, rock bolts at 5 ft centers must be installed in addition to the shotcrete to provide adequate support

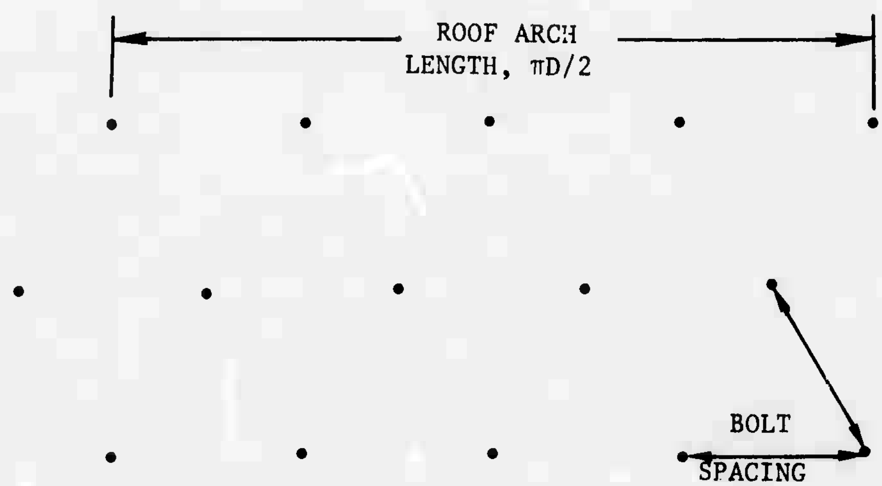


Figure 55. Assumed Rock Bolt Spacing Pattern

The volume of shotcrete used (yd^3) per foot of tunnel length is:

crown only, $V = \left(\frac{1}{2}\right)\left(\pi D\right)\left(TH\right)/324$

crown and sides, $V = \left(\frac{3}{4}\right)\left(\pi D\right)\left(TH\right)/324$

where D = tunnel diameter (ft)

TH = shotcrete thickness (in.)

3. Stel Ribs

The assumptions for steel rib design and installation are:

- Capacity of rib is according to Proctor and White, Rock Tunneling with Steel Supports[55]
- Continuous rib design with circular arch is used

The rock load per foot of tunnel width on a rib is

$$LOAD = k\ D(SP)\rho_R$$

where LOAD = rock load (lb/ft)

D = tunnel width or diameter (ft)

SP = rib set spacing (ft)

ρ_R = rock weight density (lb/ft^3)

k = rock quality factor from Table 42

When the rock load in pounds per foot of tunnel width is known, it is possible to determine the weight of steel rib that is necessary for ground support from Table 1 (p. 238) of Ref. 55. This table for continuous ribs was reduced for programming to a set of 11 linear equations, one equation for each diameter (14 ft - 34 ft) shown in the table.

TABLE 42
ROCK QUALITY LOAD FACTOR

RQD	Excavation Process	k
> 90	Any	0
75-90	Drill and Blast	0.45
	Other*	0.2
50-75	Drill and Blast	0.95
	Other	0.7
25-50	Drill and Blast	1.65
	Other	1.3
< 25	Drill and Blast	2.4
	Other	1.9

*Includes boring machine, water jet and projectile impact.

For each diameter, the set of rock load values and corresponding weight per foot values for that diameter, were fitted to a line using the least squares method, letting rock load equal the independent variable.

The program, after finding the rock load (LOAD), and knowing the diameter of the tunnel (D), then finds the weight per foot (WPF) of the rib by substituting LOAD into the equation which applies.

The equations are:

$$0 < D \leq 15 \qquad WPF = .001870 \times LOAD + 2.733$$

$$15 < D \leq 17 \qquad WPF = .002061 \times LOAD + 2.803$$

$$17 < D \leq 19 \qquad WPF = .001995 \times LOAD + 4.035$$

$$19 < D \leq 21 \qquad WPF = .002305 \times LOAD + 3.272$$

$$21 < D \leq 23 \qquad WPF = .002527 \times LOAD + 2.591$$

$$23 < D \leq 25 \qquad WPF = .002584 \times LOAD + 3.632$$

$$25 < D \leq 27 \qquad WPF = .002737 \times LOAD + 3.760$$

$$27 < D \leq 29 \qquad WPF = .002864 \times LOAD + 6.426$$

$$29 < D \leq 31 \qquad WPF = .003215 \times LOAD + 3.516$$

$$31 < D \leq 33 \qquad WPF = .003444 \times LOAD + 3.114$$

$$33 < D \leq 35 \qquad WPF = .003661 \times LOAD + 3.144$$

The weight of the entire circular rib (WPR) is then calculated as

$$WPR = (WPF)\ \pi D$$

where WPR = rib weight (lb)

WPF = weight per foot of rib (lb/ft)

D = tunnel diameter

from which the steel costs may be calculated.

B. ENVIRONMENTAL CONTROL

In the broadest sense, environmental control includes all aspects of maintaining an acceptable working environment within the tunnel, including:

- Ventilation
- Cooling
- Ground-water removal
- Dust removal
- Noise reduction
- Fire protection
- Sanitation
- Auxiliary services (e.g., lighting)
- Safety

Our analysis of the environmental control requirements is focused on the first three items above, ventilation, cooling, and ground water removal, because these are major factors affecting the performance and cost of an excavation system which vary from one tunnel to the next, one geology to the next. Modeling is straightforward, calculating the required machinery costs and power usages to meet the needs of the particular tunnel under investigation by comparison with past environmental control systems used to meet similar demands.

Figure 56 shows schematically the environmental parameters which are included in the environmental control model. Table 43 lists the performance parameters and input specifications which are required by the model.

1. Ventilation

The minimum quantity of air necessary in a tunnel is determined by human needs, legal requirements, dilution of toxic gases, and comfort or work efficiency standards. A design requirement for adequate ventilation may be calculated from the following considerations:

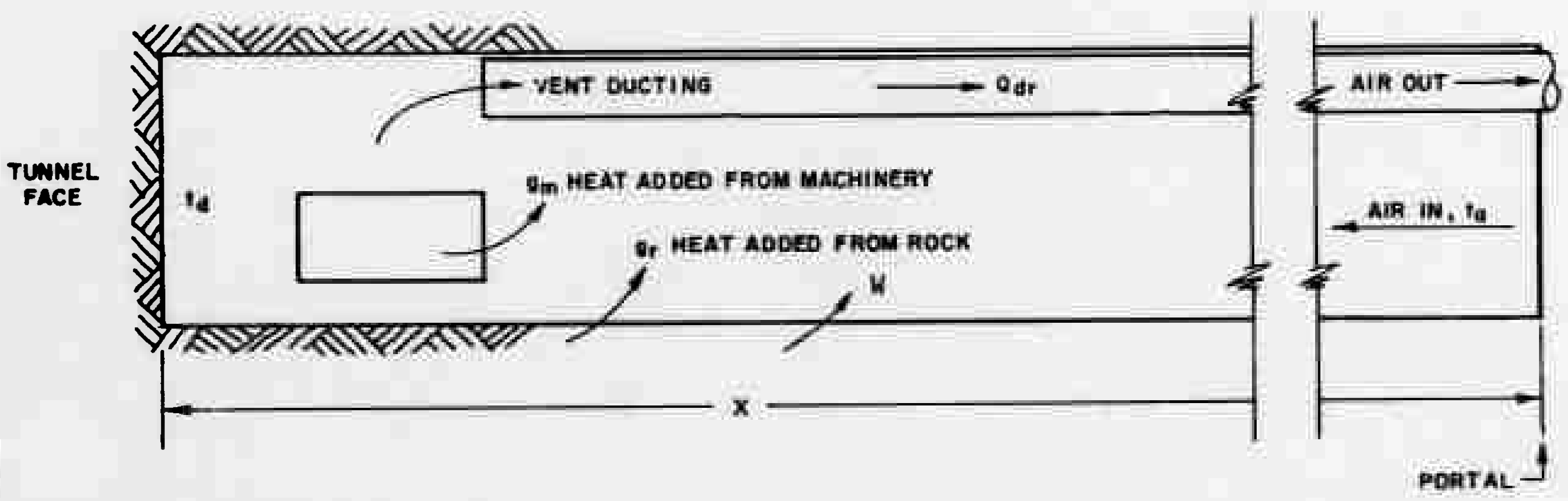
Minimum human breathing needs	20-30 (cfm/man)
Mining laws	100-200 (cfm/man)
Dilution of diesel exhaust	106 + 0.334 × (horsepower) (cfm/hr)
Comfort or work efficiency, velocity standards	50 ft/min
Dilution of toxic gases	$Q = Q_g/AMC$

where Q = volume rate of ventilation (cfm)

Q_g = volume rate of toxic gas entering tunnel (cfm)

AMC = maximum allowable concentration of toxic gas in tunnel (ppm by vol) given in Table 44

It can be seen that although minimum human breathing needs are 20-30 cfm per man, state mining and excavation laws usually require 5 to 10 times this amount to insure adequate ventilation. The specific requirement for each diesel-powered machine in the tunnel is usually established by testing procedures described in U.S. Bureau of Mines Schedule 24. The above relationship is an approximation of numerous test results for separate diesel engines. Which of the above considerations will be the controlling one for deciding what ventilation is adequate will depend on tunnel size, diesel horsepower, etc. In any case, however, there is a



ROCK TEMPERATURE: t_r (°F)

AIR TEMPERATURE: AT PORTAL t_a; AT FACE t_d < 80°F (MAX)

REQUIRED AIR FLOW: Q_{dr} (cfm)

WATER INFLOW: W (cfm)

Figure 56. Environmental Control Model

TABLE 43

ENVIRONMENTAL CONTROL PERFORMANCE PARAMETERS
AND INPUT SPECIFICATIONS

1. Maximum work force in tunnel at any time (number of men) ______________________________

2. Total diesel power in tunnel (hp) ______________

3. Minimum air volume rate per man (cfm/man)* __________

4. Minimum air volume rate per diesel horsepower (cfm/hp)* ______________________________

5. Minimum velocity of air at working face (ft/min)* _____

6. Maximum velocity of air in tunnel (ft/min)* __________

7. Toxic gas information:*

Item	Volume Rate of Gas Entering Tunnel (cfm)	Maximum Allowable Gas Concentration (ppm by vol.)
1		
2		
...		

8. Ambient air temperature at portal or shaft (°F) ______

9. Desired air temperature at working face (°F)* ________

10. Air density (lb/ft^3)* ____________________

11. Friction factor for ventilation ducting* ___________

12. Friction factor for tunnel lining* ______________

13. Specific heat of air (Btu/lb°F)* ______________

14. Conductivity of air (Btu/hr-ft°F)* ______________

TABLE 43 (Cont.)

ENVIRONMENTAL CONTROL PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

1. Plant and Equipment

Item	Ownership (Rental) Cost Per	Hr (check one)	Ft	Unusual Developmental Cost Per	Hr (check one)	Ft
MAJOR ITEMS:						
Ventilation Plant						
Mechanical Cooling Plant						
Pumping Plant						
ADDITIONAL ITEMS:						

2. Job Materials

Item	Cost		Lifetime	
	Value	Unit	Value	Unit
MAJOR ITEMS:				
Power: Electric*		$/kW·hr	-	-
Cooling Plant Operation		$/Btu	-	-
Pumping Plant Operation		$/ft^3	-	-
Minor Servicing & Repairs (per maintenance)		$		
ADDITIONAL ITEMS:		$/ft		

3. Direct Labor

Labor Type	Number Required/Shift	Rate $/hr
MAJOR TYPES:		
Electrician		
Mechanic		
Installation Crew		
ADDITIONAL TYPES:		

4. Permanent Materials

Item	Cost	
	Value	Unit
(none required)		

*Representative values are programmed into the model. A user may substitute his own values if he desires.

TABLE 44

MAXIMUM ALLOWABLE CONCENTRATION OF TOXIC GAS IN TUNNEL[55]

Gas	AMC (ppm by vol.)
Nitrogen	800,000
Carbon Dioxide	5,000
Methane	10,000
Carbon Monoxide	100
Nitrogen Oxides	5
Hydrogen Sulfide	20
Sulfur Dioxide	5
Hydrogen	0
Aldehyde	10

limit to the volume of air which can flow through a tunnel without adding environmental problems of its own. The commonly cited maximum air flow velocity in a tunnel is 600-1000 ft/min.[56]

The design volume of air flow calculated by the above method may fail to provide adequate cooling in very hot tunnels or in tunnels excavated by one of the novel techniques. For these tunnels additional air flow and perhaps even mechanical cooling will be required.

2. Cooling

For the purpose of modeling, the heat added to the tunnel environment is assumed to come from two sources, the ambient rock, and the novel rock breaking devices (water jet or projectile). Other heat sources (other machinery or curing concrete/shotcrete) have not yet been included in the model. The impact of these other sources should be investigate further. To calculate the heat transfer from the ambient rock to the air flowing in the tunnel, the program first calculates the Reynolds number, Re, for tunnel air flow conditions:

$$Re = \frac{V_t D_t}{0.165}$$

where V_t = tunnel air velocity (ft/min)

D_t = tunnel diameter (ft)

Then the Nusselt number for the air flow is determined from its relationship to Reynolds number (Fig. 57). This relationship is approximated by the following equations:

$$Re \leq 2100 \qquad Nu = 0.5\ Re^{0.3}$$

$$2100 < Re < 10{,}000 \qquad Nu = 5.0 + 26.7 \left(\frac{Re - 2100}{7900}\right)$$

$$= 0.00338\ Re + 2.10$$

$$Re \geq 10{,}000 \qquad Nu = 0.02\ Re^{0.8}$$

The heat transfer coefficient, h_c, for the tunnel is then calculated as:

$$h_c = Nuk_a/D_t$$

where k_a = conductivity of air (Btu/hr-ft-°F)
= 0.0147 if not specified

Therefore the approximate heat added to the tunnel air from the tunnel wall may be calculated as

$$g_r = h_c\ \pi D_t X \left(t_r - \frac{t_a - t_d}{2}\right)$$

where g_r = heat transfer rate of ambient rock (Btu/hr)

X = distance from portal or shaft to tunnel face (ft)

t_r = ambient rock temperature (°F)

t_a = air temperature at portal or shaft (°F)

t_d = air temperature at tunnel face °F

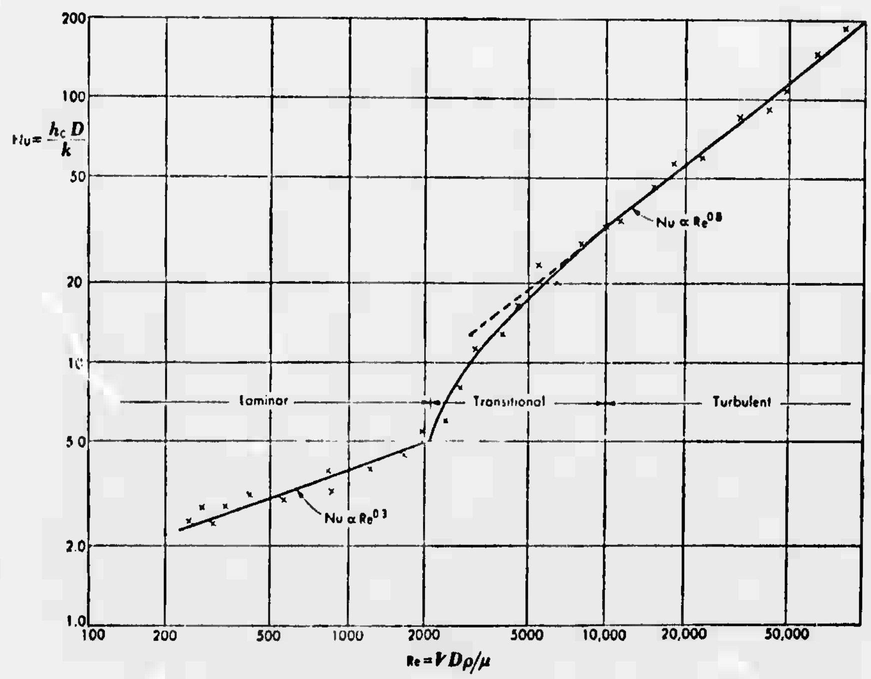


Figure 57. Nusselt Number vs. Reynolds Number for Air Flowing in a Pipe[57]

The air temperature at the tunnel face is assumed to equal the desired working temperature (80°F if not specified) to yield an approximate steady flow heat value from the above equation. The actual temperature at the face for the preliminary design air flow is calculated from the temperature rise caused by this heat from the tunnel walls and the heat added by the excavation machinery:

$$t_d = \frac{g_r + g_m}{60\, \rho_a\, Q_{dr} (C_p)_a} + t_a$$

where ρ_a = air density (lb/ft^3)

= 0.075 if not otherwise specified

g_m = heat from machinery and rock fragmentation (Btu/hr)

Q_{dr} = preliminary design air flow rate (cfm)

$= \pi \frac{D_t^2}{4} V_t$

$(C_p)_a$ = specific heat of air (Btu/lb°F)

= 0.240 if not otherwise specified

If the temperature at the face is excessive, the air flow rate is revised upward until a satisfactory working temperature is obtained. If the air flow rate reaches the maximum allowable before such a temperature is reached, it is assumed that mechanical cooling would be added to remove additional heat. The required cooling load may be calculated as the additional heat removal necessary (BTU/hr) after ventilation has reached its limit:

$$g_{cooling} = \rho_a\, Q_{dr} (C_p)_a\, (t_d - t_w) \qquad \text{(Btu/hr)}$$

where t_d = temperature at tunnel face, maximum ventilation (°F)

t_w = desired temperature (°F)

The cost of operating a mechanical cooling system is calculated as

$$\text{Cooling cost } (\$) = g_{cooling}\ r_{cooling}\ t$$

where $r_{cooling}$ = cost per Btu cooling (\$/Btu)

t = time (hr)

The cost of cooling has been assumed to be 2.0×10^{-6} dollars per Btu if not otherwise specified. This selected value is based on some representative costs of cooling[22] for mines as summarized in Table 45.

TABLE 45

REPRESENTATIVE OPERATING COSTS OF MECHANICAL COOLING OF MINES

Location	Cost \$/Btu
Morro Velho	1.8×10^{-6}
Robinson Deep	1.3×10^{-6}
Magma	1.5×10^{-6}
Richardson's System	$.5 \times 10^{-6}$

3. Power Requirements

In addition to the mechanical cooling operational costs (if any) there are operating costs for the vent motors and there may be pumping costs for dewatering the tunnel.

The ventilation power cost calculations are as follows:

Vent duct velocity,* $V_d = 2380 + 0.0483\ Q_{dr}$ (ft/min)

*From ventilation equipment specifications recommended by Joy Manufacturing Co.

where Q_{dr} = tunnel air volume rate (cfm)

$$\text{Vent duct diameter, } D_d = 2\sqrt{\frac{Q_{dr}}{\pi V_d}} \quad \text{(ft)}$$

Friction loss of air flowing in vent duct,

$$H_d = \frac{K_d X V_d^2}{1.3\ D_d} \qquad \text{(in. water)}$$

where K_d = friction factor for vent ducting, assumed = 2×10^{-9} if not otherwise specified (see Table 46)

X = tunnel length (ft)

TABLE 46

FRICTION FACTOR K_d FOR VENT DUCTING[56]

Pipe or Tubing	Friction Factor, $K \times 10^{10}$	
	Good (new)	Average (used)
Steel, wood	15	20
Jute, canvas	20	25
Spiral-type canvas	22.5	27.5

Friction loss of air flowing in tunnel,

$$H_t = \frac{K_t X V_t^2}{1.3\ D_t}$$

where K_t = friction factor for the tunnel, assumed = 1.5×10^{-8} if not otherwise specified (see Table 47)

X = tunnel length (ft)

TABLE 47

FRICTION FACTOR K_t FOR MINE AIRWAYS

Type of Airway	Irregularities of Surfaces, Areas, and Alignment	Values of $K_t \times 10^{10}$											
		Straight			Sinuous or Curved								
					Slightly			Moderately			High Degree		
		Clean (basic values)	Slightly Obstructed	Moderately Obstructed	Clean	Slightly Obstructed	Moderately Obstructed	Clean	Slightly Obstructed	Moderately Obstructed	Clean	Slightly Obstructed	Moderately Obstructed
Smooth lined	Minimum	10	15	25	20	25	35	25	30	40	35	40	50
	Average	15	*20*	30	25	30	40	30	35	45	40	45	55
	Maximum	20	25	35	30	35	45	35	40	50	45	50	60
Sedimentary rock (or coal)	Minimum	30	35	45	40	45	55	45	50	60	55	60	70
	Average	55	*60*	70	65	70	80	70	75	85	80	85	95
	Maximum	70	75	85	80	85	95	85	95	100	95	100	110
Timbered (5-ft centers)	Minimum	80	85	95	90	95	105	95	100	110	105	110	120
	Average	95	*100*	110	105	110	120	110	115	125	120	125	135
	Maximum	105	110	120	115	120	130	120	125	135	130	135	145
Igneous rock	Minimum	90	95	105	100	105	115	105	110	120	115	120	130
	Average	145	*150*	160	155	160	165	160	165	175	170	175	195
	Maximum	195	200	210	205	210	220	210	215	225	220	225	235

SOURCE: G. E. McElroy, "Engineering Factors in the Ventilation of Metal Mines," *U.S. Bur. Mines Bull. 385* (1935), p. 43.

The fan horsepower is then calculated, assuming .75 operating efficiency, to be that needed to overcome the friction losses of the air flow in the tunnel and ventilation duct:

$$P_f = \frac{1}{.75}\left(\frac{H_t + H_d}{6350}\right) Q_{dr} \quad \text{(hp)}$$

The power cost to operate the ventilation system is simply

$$\text{Fan Power Cost (\$)} = P_f \,(0.7452)\,(C_p)\; t$$

where C_p = electrical power cost (\$/kW-hr)
= 0.02 if not otherwise specified
t = operating time (hr)

The pumping and dewatering cost, as now modeled by the program, consists simply as a user-input plant and equipment cost and an operating cost calculated by

$$\text{Pumping cost (\$)} = 60\; W C_w t$$

where W = water inflow rate (cfm)
C_w = pumping plant operational cost (\$/ft^3 of water)
t = time (hr)

The value C_w, which depends on the manner in which the tunnel is dewatered, should be user-estimated for the particular tunnel of interest.

VI. SIMULATION EXAMPLE

A. INTRODUCTION

This section describes the application of the computer model to an actual tunneling project. The example chosen is the Layout Tunnel presently being constructed as part of the Strawberry Aqueduct of the Central Utah Project administered by the Bureau of Reclamation.

Figure 58 shows the location of the Layout Tunnel within the Strawberry Aqueduct system. This project consists of water conveyance tunnels, pipelines, reservoirs, and dams intended to enlarge the capacity of the existing Strawberry Reservoir.* The Layout Tunnel will extend 17,355 ft in a north-south direction from Layout Creek to Water Hollow Creek (Fig. 59). The tunnel is to be concrete lined with a finished diameter of 10 ft 4 in., constructed from a machine-bored diameter of 12 ft 11 in. (Fig. 60).[58]

The 5038 ft segment of tunnel simulated by the computer model corresponds to that length excavated by the S. A. Healy Company between June 25, 1971, and September 22, 1971. A Robbins Model No. 141-127-1 boring machine was used. Three trains, each consisting of ten 6 2/3 yd^3 (struck) muck cars, pulled by two 10-ton Plymouth Mine-o-motive DMD-24 diesel locomotives transported material. The geology was hard sandstone and conglomerate.

B. SIMULATION INPUT

The basis for the simulation input is the detailed tunnel information compiled from several sources which may be found in Vol. II.

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*The project also includes the Stillwater Tunnel proposed by the Bureau of Reclamation as a practical laboratory for research in rapid underground construction.

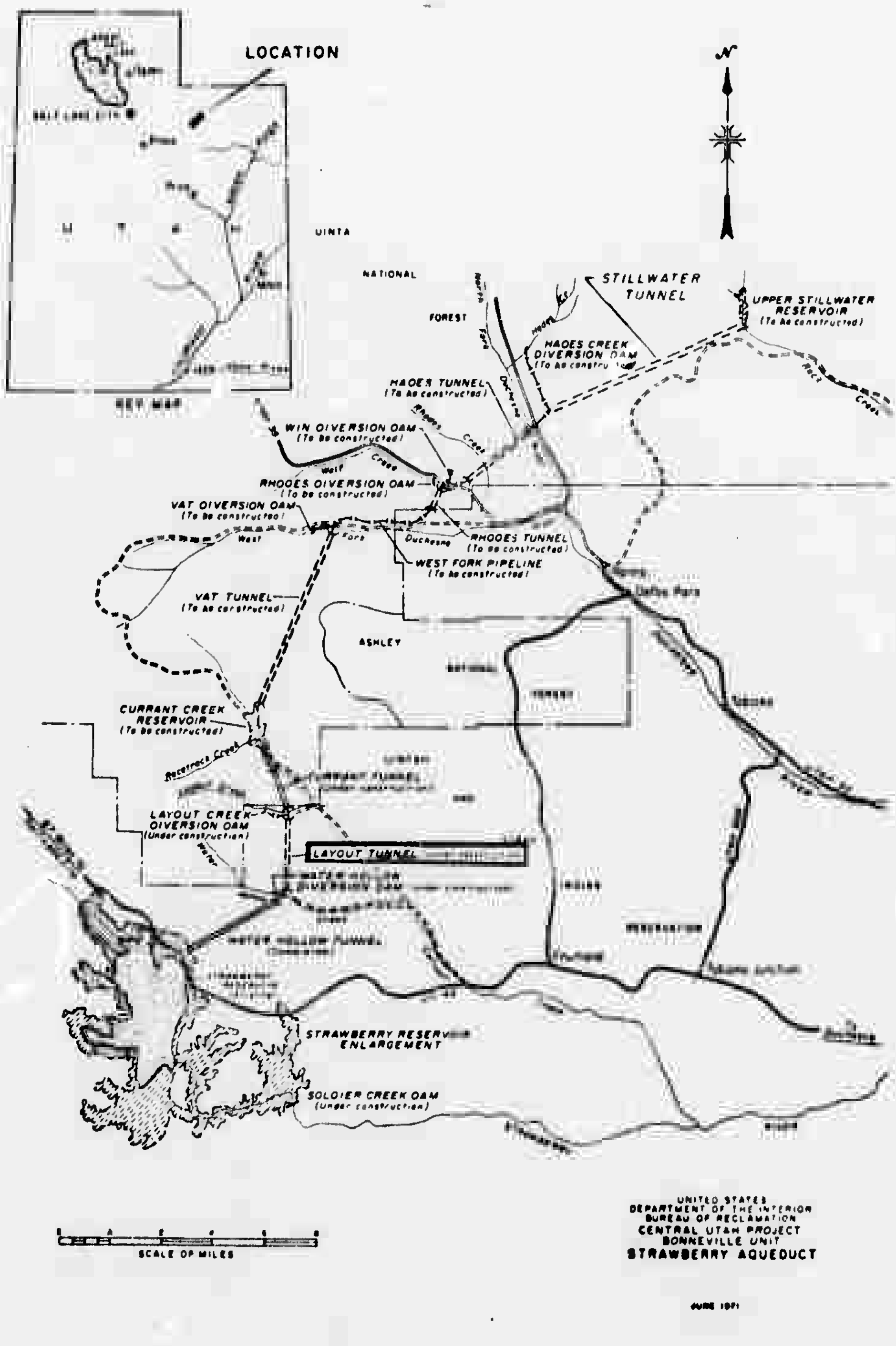


Figure 58. Layout Tunnel Location Map

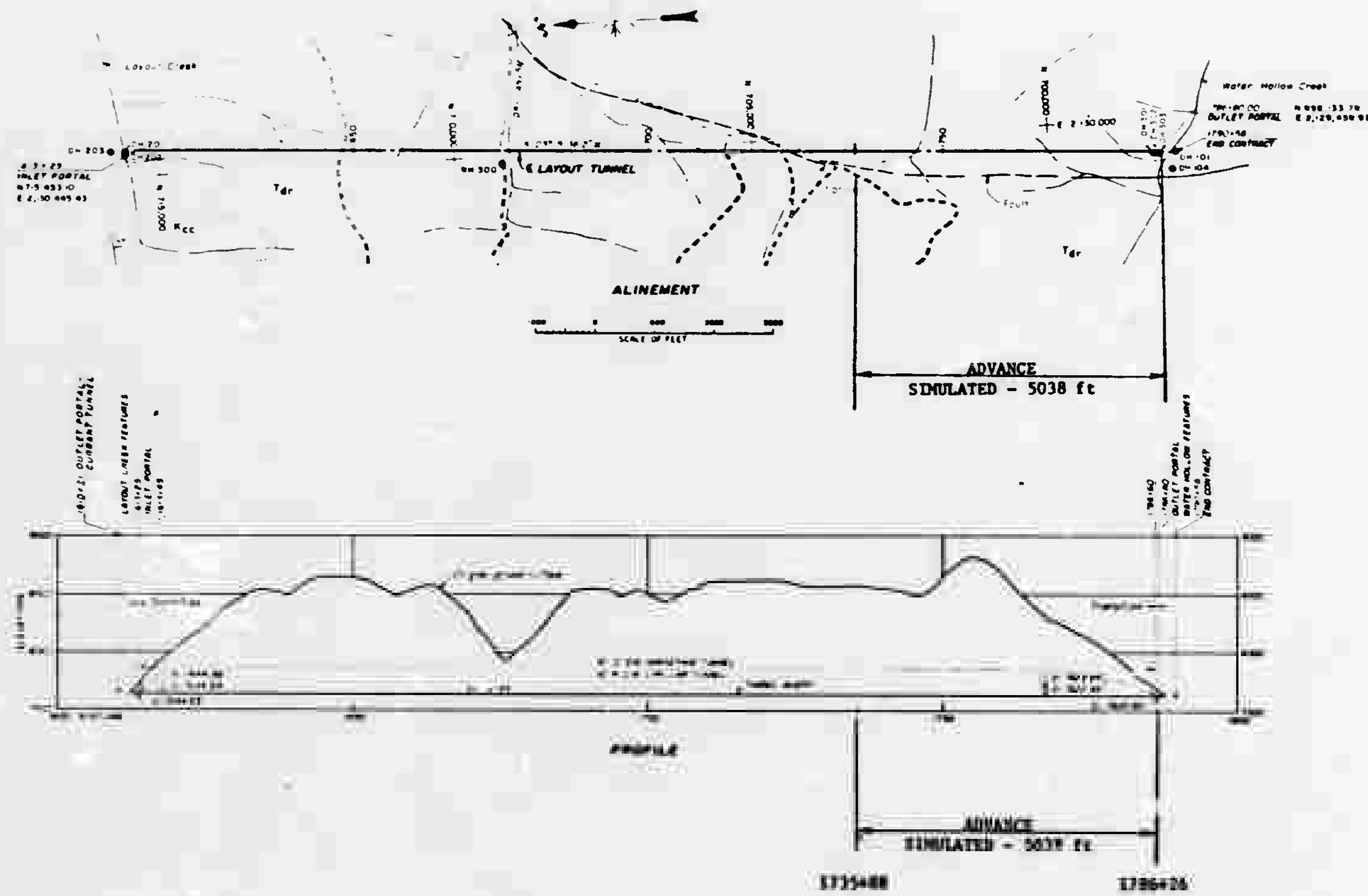
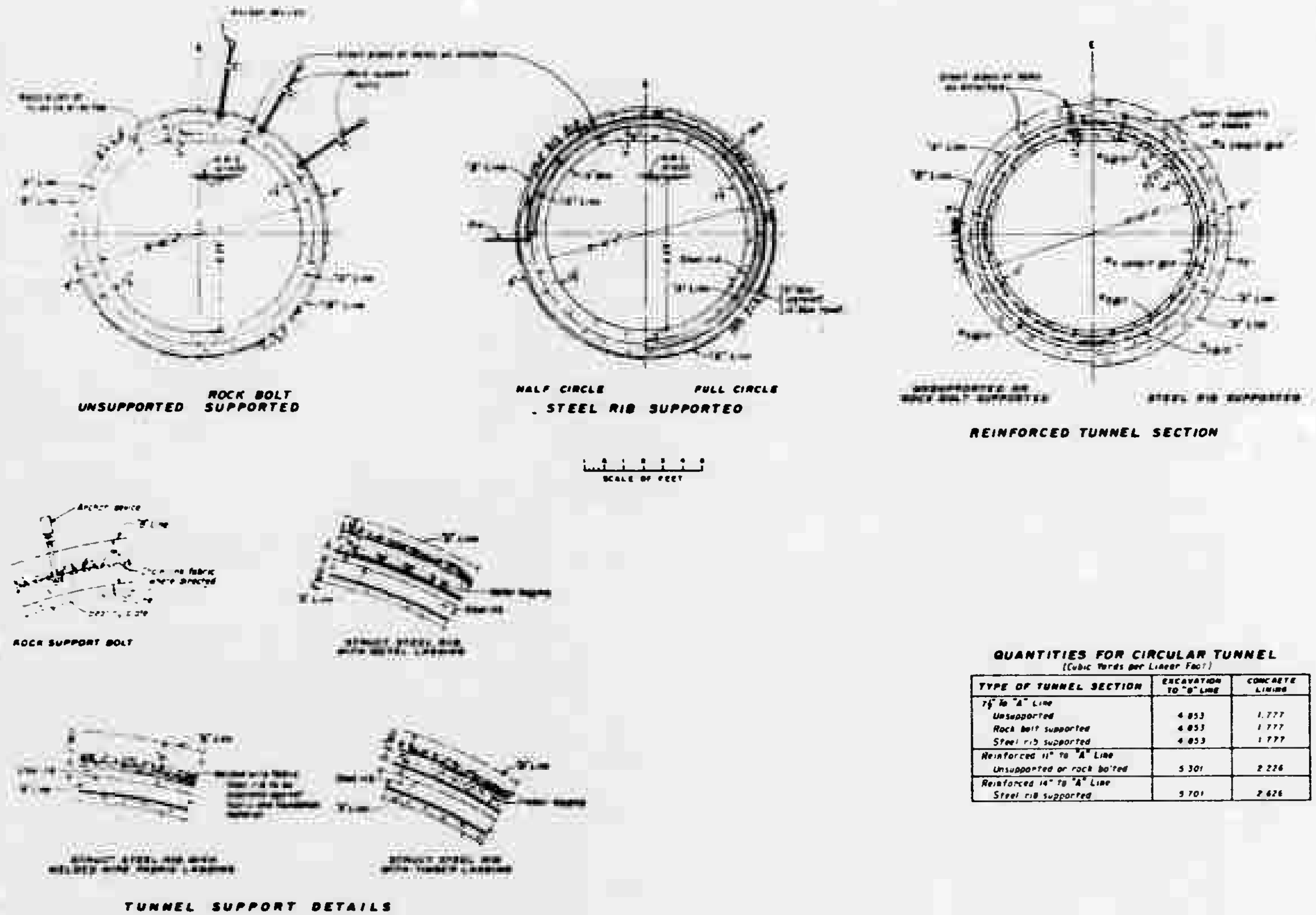


Figure 59. Layout Tunnel Alignment and Profile



QUANTITIES FOR CIRCULAR TUNNEL
(Cubic Yards per Linear Foot)

TYPE OF TUNNEL SECTION	EXCAVATION TO "B" LINE	CONCRETE LINING
7½" to "A" Line		
Unsupported	4 853	1.777
Rock bolt supported	4 853	1 777
Steel rib supported	4 853	1 777
Reinforced 11" to "A" Line		
Unsupported or rock bolted	5 301	2 226
Reinforced 14" to "A" Line		
Steel rib supported	5 701	2 626

Figure 60. Layout Tunnel Sections for Machine Bored Tunnel

We have inferred from the Bureau of Reclamation progress reports that the construction proceeded using a schedule of three 8-hour shifts per day, 6 days per week for 64.5 working days. A summary of the operating days, shift time, boring machine operating time, and progress made is shown in Table 48. During this four-month period the average rate of advance attained was 11.5 ft per operating hour of the boring machine, and the maximum advance in a day was 209 ft.

Figure 61 and Tables 49 through 53 summarize the input to the model. This figure shows the geology input: in situ density, water inflow rates, ambient rock temperature, rock quality (RQD), abrasiveness, and unconfined compressive strength for the length of the tunnel segment. Table 49 lists characteristics of the boring machine, Table 50 for the integrated conveyor loader; Table 51 for the rail system; Table 52 for ground support; and Table 53 for environmental control.

C. DISCUSSION OF RESULTS

Figure 62 compares the actual advance of the tunnel face with that predicted for two similar computer simulations. In both cases the excavation system and geology were identical. Case 1, however, assumes a boring machine availability of 0.45 throughout the project period; Case 2 assumes a reduced availability of 0.18 for the first five working days and 0.45 thereafter to account for the delay in operations during this period which is evident in the progress reports (Vol. II). Figure 63 compares actual and predicted boring machine utilization and graphically displays the distinction between Case 1 and Case 2.

The last of the performance and cost reports generated periodically by the model are shown in Figs. 64 and 65 for Case 1 and Case 2 respectively.* Progress reports indicate that it took 64.5 days to excavate

*The 5092 ft advance includes 54 ft of conventionally excavated tunnel which was not modeled; the boring machine advance was 5038 ft.

TABLE 48

SUMMARY OF LAYOUT TUNNEL OPERATION
JUNE 25 - SEPTEMBER 22, 1971

Month	Operating Days	Shift Time (hr)	Boring Machine Operating Time (hr)	Advance (ft)
June	4	96	15.5	100
July	20	480	113.4	1329
August	26	624	194.3	2314
September	14.5	348	115.0	1295
TOTAL	64.5	1548	438.2	5038

Average Advance Rate = 11.5 ft/hr

Maximum Daily Advance = 209 ft

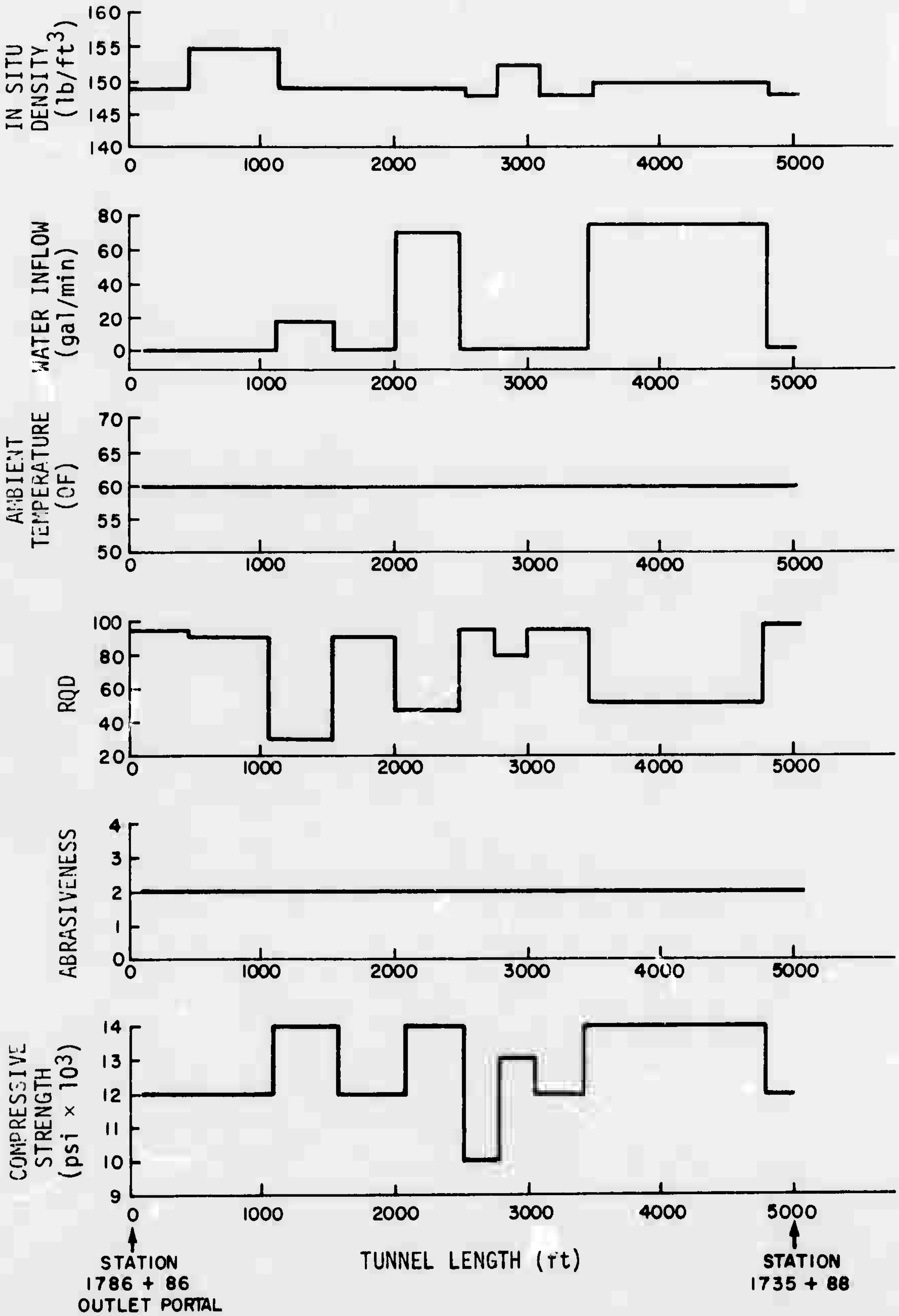


Figure 61. Geologic Properties Input

TABLE 49

BORING MACHINE PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

Boring Machine Information

1.	Rated rotational power (hp)	600	
2.	Energy required per rock volume broken (in.-lb/in^3 × 10^3)*	---	
3.	Rotational speed of boring head (rpm)	6	
4.	Time to assemble boring machine (hr)	176	
5.	Time to change bore diameter (hr)	24	
6.	Time to disassemble machine (hr)	176	
7.	Maintenance parameters (separate from cutter change):	Case 1	Case 2
	(a) availability of machine (%)	0.45	0.18 To Day Five 0.45 Thereafter
	(b) in lieu of (a) average down time per maintenance period (hr)	-----	
	(c) Average time between maintenance periods (hr)	-----	

Cutter Information (rolling cutters)

1. Total number of cutters ____________ 29
2. Radial location of cutters

Cutter No.	R (in.)

(See attached table)

3. Estimated cutter life as a function of rock abrasiveness:

	Abrasiveness Index	Travel (ft)
(Least abrasive)	1	1,000,000
(Moderately abrasive)	2	400,000
(Most abrasive)	3	100,000

4. Time required to replace one cutter (hr) ____________ 0.5
5. Minimum observed fractional wear of cutter to cause replacement during any one cutter change period ____________ 0.8

CUTTER TABLE

Cutter No.	R (in.)
1	2.875
2	5.75
3	8.625
4	11.5
5	14.375
6	17.75
7	20.125
8	23.
9	25.75
10	28.5
11	31.25
12	34.
13	36.75
14	39.5
15	42.25
16	45.
17	47.75
18	50.5
19	53.25
20	56.
21	58.75
22	61.4375
23	63.9375
24	66.0625
25	68.1875
26	70.3125
27	72.4375
28	73.6875
29	74.9375

TABLE 49 (cont.)

BORING MACHINE PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

1. Plant and Equipment

Item	Ownership (Rental) Cost Per	Hr (check one)	Ft	Unusual Developmental Cost Per	Hr (check one)	Ft
MAJOR ITEMS:						
Boring machine unit	34.60		$/ft	----		
Transmission lines	3.40		$/ft	----		
ADDITIONAL ITEMS:						

2. Job Materials

ITEM	COST		LIFETIME	
	VALUE	UNIT	VALUE	UNIT
MAJOR ITEMS:				
Power: Electric*	---	$/kw·hr		
Cutters	80	$/cutter		
Cutter Bearings (prorated)	40	$/cutter chg.		
Minor Servicing & Repairs (per maintenance)	100			
ADDITIONAL ITEMS:				

3. Direct Labor

Labor Type	Number Required/Shift	Rate $/hr
MAJOR TYPES:		
Machine Operator	1	8.46
Miners	4	5.32
Electrician	1	8.02
Mechanics	1	7.11
ADDITIONAL TYPES:		

4. Permanent Materials

Item	Cost Value	Unit
(none required)		

*Representative values are programmed into the model.

TABLE 50

INTEGRATED-CONVEYOR LOADER PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

1.	Muck flow cross section (ft^2)	---
2.	Belt speed (ft/min)	---
3.	In lieu of (1 and 2) Maximum mucking rate (ton/hr)	175
4.	Maintenance parameters	
	(a) Unit availability (%)	---
	(b) In lieu of (4-a) down time per maintenance cycle (hr)	8
	(c) Time between maintenance periods (days)	13

1. **Plant & Equipment**

Item	Ownership (Rental) Cost Per	Hr (check one)	Ft	Unusual Developmental Cost Per	Hr (check one)	Ft
MAJOR ITEMS:						
Conveyor Equipment	4.65		$/ft			
Supporting Structure	4.00		$/ft			
ADDITIONAL ITEMS:						

2. **Job Materials**

Item	Cost Value	Cost Unit	Lifetime Value	Lifetime Unit
MAJOR ITEMS:				
Servicing and Repair	50	$/Maint.	-	-
ADDITIONAL ITEMS:				

3. **Direct Labor**

Labor Type	Number Required/Shift	Rate $/hr
MAJOR TYPES:		
Operator	1	7.20
Beltman	1	7.20
ADDITIONAL TYPES:		

4. **Permanent Materials**

Item	Cost Value	Cost Unit
(None Required)		

TABLE 51

RAIL SYSTEM PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

TRAIN INFORMATION

1.	Maximum number of trains	3
2.	Number of trains	
	(a) Initially	3
	(b) To be added per new mile of system — First __ mi	0
	Thereafter	0
3.	Number of muck cars/train	10
4.	Capacity of muck car: Volume (yd^3)	7
	Weight* (tons)	-
5.	Weight of empty muck car (tons)	4.05
6.	Number of axles/muck car	2
7.	Weight of locomotive (tons)	10
8.	Number of axles on locomotive	2
9.	Drive train efficiency of locomotive*	-
10.	Traction coefficient*	-
11.	Locomotive power source (diesel or electric)	Diesel
12.	Rated continuous operation horsepower	100
13.	Peak horsepower available	140
	or	
	Traction effort table	
	TE (lb) Speed (mph)	
14.	Maximum allowed speed (mph)	12
15.	Maximum allowed acceleration (mph^2)	720
16.	In lieu of items 5-15	
	(a) Peak speed empty (mph)	-
	full (mph)	-
	(b) Peak acceleration empty (mph^2)	-
	full (mph^2)	-
	(c) Horsepower required at peak speed Empty	-
	Full	-
	(d) Horsepower required at peak acceleration Empty	-
	Full	-
17.	Average braking deceleration empty (mph^2)	720
	full (mph^2)	720
18.	Fuel consumption rate (for diesel loc.) (gal/hp·hr)	.056
19.	Maintenance parameters	
	(a) Single train availability	-
	(b) In lieu of (19-a) down time per maintenance period (hr)	2
	(c) Time between maintenance period (days)	0.916
	(d) Number of trains allowed in maintenance at one time	2
20.	Muck loading - number of trains allowed in loading area at one time	3
21.	Muck unloading	
	(a) Maximum number of trains in discharge area	1
	(b) Time to unload one train (hr)	.17
	(c) Maximum allowed speed of train entering & leaving discharge area (mph)	0
	(d) Distance from portal to discharge area (zero for shafts) (ft)	12,000.

*If not input, values have been provided for in the model.

TABLE 51 (cont.)

RAIL SYSTEM PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

TRACK INFORMATION

22.	Single or double tracks	Single
23.	Length of switch (ft)	360
24.	Distance between switches (ft)	8650
25.	Time to assemble one switch (hr)	16
26.	Rate California switch can be moved along main line (ft/hr)	2000
27.	Allowed train speed in switch (mph)	2
28.	Rate at which track can be extended (ft/hr)	12
29.	For continuous excavation processes maximum excavator runaway tolerance (ft)	75
30.	Required loading area ahead of switch	
	(a) Maximum permitted (ft)	276
	(b) Minimum permitted (ft)	276

1. Plant & Equipment

Item	Ownership (Rental) Cost Per	Hr	Ft	Unusual Developmental Cost per	Hr	Ft
		(check one)			(check one)	
MAJOR ITEMS:						
Locomotive unit	[illegible]	[illegible]				
Muck car unit	[illegible]	[illegible]				
Track materials	[illegible]		[illegible]			
Switch	--		[illegible]			
Unloading equipment	[illegible]		[illegible]			
Maintenance shop	[illegible]		[illegible]			
ADDITIONAL ITEMS:						

2. Job Materials

Item	Cost Value	Cost Unit	Lifetime Value	Lifetime Unit
MAJOR ITEMS:				
Power: Diesel[e]	-	\$/gal	-	
Electric[e]	-	\$/kW·hr	-	
Minor servicing & Repairs (per maintenance)	50	\$	-	
ADDITIONAL ITEMS:				
	-	\$/ft	-	

3. Direct Labor

Labor Type	Number Required/Shift	Rate \$/hr
MAJOR TYPES:		
Motormen	1	7.70
Brakemen	2	7.20
Dispatchers	1	7.70
Dumpmen	1	7.20
Bull gang		
Foreman	1	8.00
Laborer	3	5.32
Maintenance		
Foreman		
Mechanics	1	7.11
Electricians		
ADDITIONAL TYPES:		

4. Permanent Materials

Item	Cost Value	Cost Unit
(None required)		

TABLE 52

GROUND SUPPORT SYSTEM PARAMETERS AND INPUT SPECIFICATIONS

1. Rate of Installation of Rock Bolts (Bolts/hr) 4
2. Rate of Installation of Steel Sets (Sets/hr) 1
3. Rate of Installation of Shotcrete (yd^3/hr) 17.5
4. Excavation Process:

   Drill and Blast ( )

   Boring Machine (X)

   Novel Technique ( )

5. Primary Support Selection Table:

RQD	EXCAVATION PROCESS	ALLOWABLE UNSUPPORTED LENGTH, AL, ft	INTERFERENCE LENGTH, TL, ft	SUPPORT TYPE	SPACING, ft	THICKNESS, in	ADDITIONAL SUPPORT
>95	Any	∞	0	None Req'd	---	---	---
90 - 95	Drill and Blast	10	0	Shotcrete	---	0.5 (Crown Only)	None
	Any Other	∞	0	None Req'd	---	---	---
75 - 90	Drill and Blast	5	0	Rock Bolts	5	---	None
	Boring Machine	8	4	Shotcrete	---	1.0 (Crown Only)	None
	Any Other	8	0	Shotcrete	---	1.0 (Crown Only)	None
50 - 75	Drill and Blast	4	0	Steel Ribs	4	---	None
	Boring Machine	6	4	Shotcrete	---	3.0 (Crown Only)	None
	Any Other	6	0	Shotcrete	---	3.0 (Crown Only)	None
25 - 50	Drill and Blast	3	0	Steel Ribs	3	---	Lagging
	Boring Machine	5	4	Steel Ribs	3	---	Rock Bolts
	Any Other	5	0	Shotcrete	---	5.0 (Crown and Sides)	Rock Bolts
<25	Drill and Blast	2	0	Steel Ribs	2	---	Lagging
	Boring Machine	2	4	Steel Ribs	2	---	Lagging
	Any Other	2	0	Steel Ribs	2	---	Lagging

TABLE 52 (cont.)
GROUND SUPPORT SYSTEM PARAMETERS AND INPUT SPECIFICATIONS

1. Plant & Equipment

Item	Ownership (Rentel) Cost Per	Hr	Ft (check one)	Unueuel Developmentel Cost Per	Hr	Ft (check one)
MAJOR ITEMS:						
Rock Bolt Drills	7.50	$/hr				
Shotcrete Plent	10.00	$/hr				
ADDITIONAL ITEMS:						

2. Job Materiels

Item	Cost		Lifetime	
	Value	Unit	Value	Unit
MAJOR ITEMS:				
Servicing end Repairs (per maintenence)	50	$	---	---
ADDITIONAL ITEMS:				

3. Direct Labor

Labor Type	Number Required/Shift	Rate $/hr
MAJOR TYPES:		
Installetion Crew	8	7.00
ADDITIONAL TYPES:		
Meintenence & Portel Crew	2	7.20

4. Permanent Materials

Item	Cost	
	Value	Unit
Steal	[illegible]	$/lb
Shotcrate	[illegible]	$/yd^3

TABLE 53

ENVIRONMENTAL CONTROL PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

1.	Maximum work forces in tunnel at any time (number of men)	40
2.	Total diesel power in tunnel (hp)	300
3.	Minimum air volume rate per man (cfm/man)*	-
4.	Minimum air volume rate per diesel horsepower (cfm/hp)*	-
5.	Minimum velocity of air at work face (ft/min)*	-
6.	Maximum velocity of air in tunnel (ft/min)*	-

7. Toxic gas information:*

Item	Volume Rate of Gas Entering Tunnel (cfm)	Maximum Allowable Gas Concentration (ppm by vol.)
1	0	-
2	0	-
...		

8.	Ambient air temperature at portal or shaft (°F)	60
9.	Desired air temperature at working face (°F)*	-
10.	Air density (lb/ft^3)*	-
11.	Friction factor for ventilation ducting*	-
12.	Friction factor for tunnel lining*	-
13.	Specific heat of air (Btu/lb°F)*	-
14.	Conductivity of air (Btu/hr-ft°F)*	-

TABLE 53 (cont.)

ENVIRONMENTAL CONTROL PERFORMANCE PARAMETERS AND INPUT SPECIFICATIONS

1. Plant and Equipment

Item	Ownership (Rental) Cost Per	Hr	Ft	Unusual Developmental Cost Per	Hr	Ft
		(check one)			(check one)	
MAJOR ITEMS:						
Ventilation Plant	1		$/ft			
Mechanical Cooling Plant	0					
Pumping Plant	0					
ADDITIONAL ITEMS:						

2. Job Materials

Item	Cost Value	Cost Unit	Lifetime Value	Lifetime Unit
MAJOR ITEMS:				
Power: Electric*	-	$/kW·hr	-	-
Cooling Plant Operation	2×10^{-6}	$/Btu	-	-
Pumping Plant Operation	0	$/ft³	-	-
Minor Servicing & Repairs (per maintenance)	25	$		
ADDITIONAL ITEMS:				

$/ft

3. Direct Labor

Labor Type	Number Required/Shift	Rate $/hr
MAJOR TYPES:		
Electrician	1	8.02
Mechanic	1	7.11
Installation Crew	2	7.00
ADDITIONAL TYPES:		

4. Permanent Materials

Item	Cost Value	Cost Unit
(none required)		

*Representative values are programmed into the model.

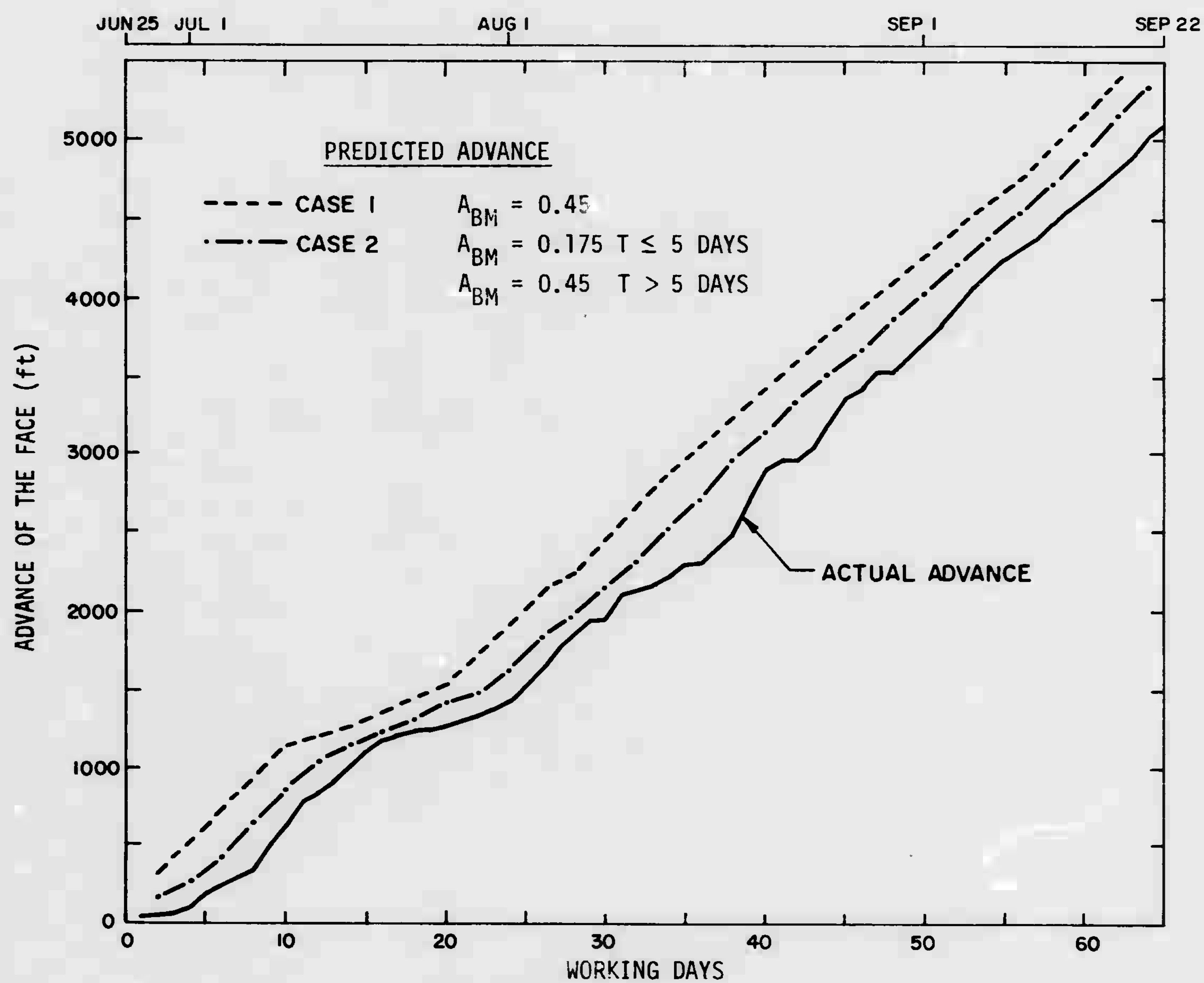


Figure 62. Comparison of Actual and Predicted Face Advance

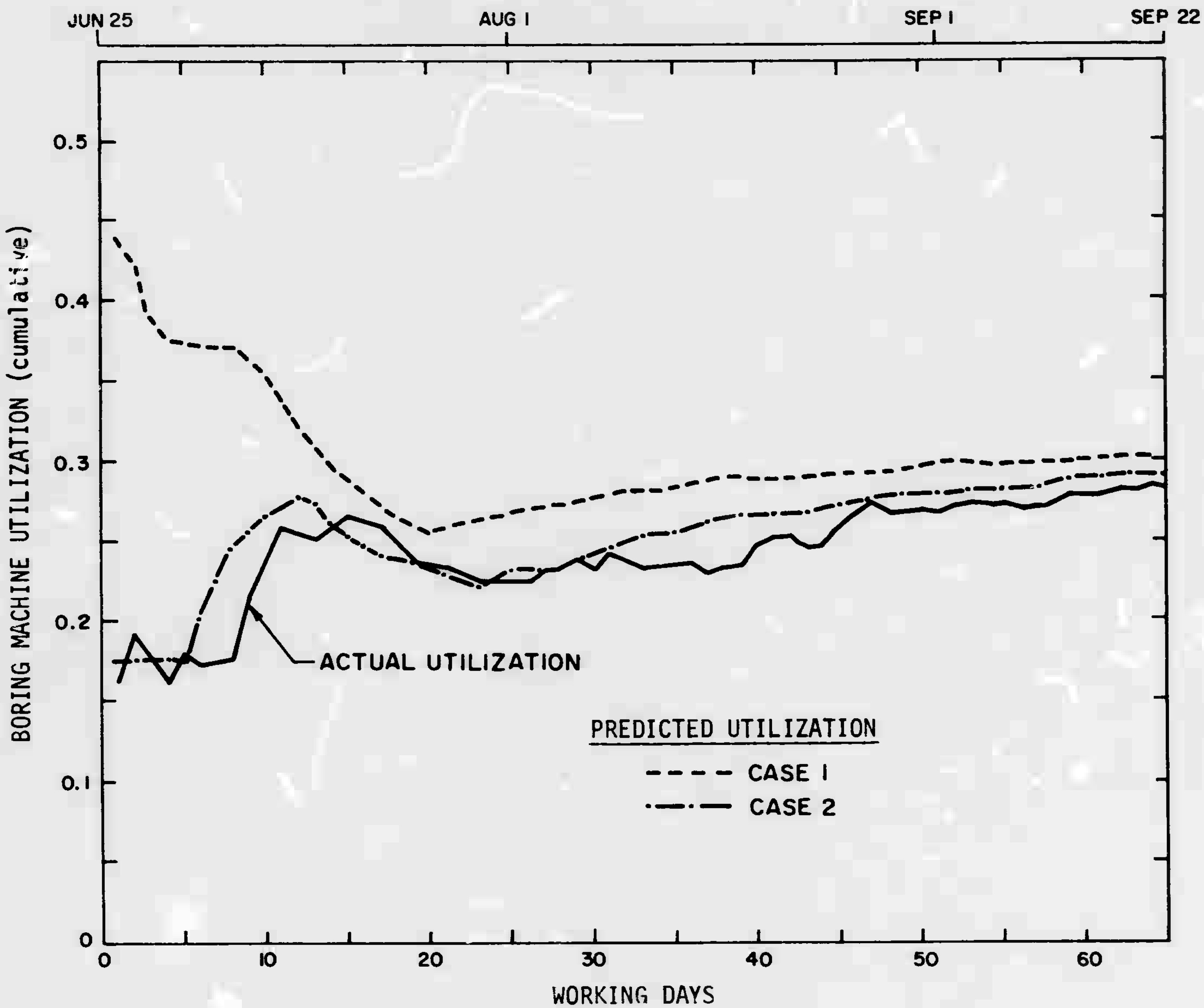


Figure 63. Comparison of Actual and Predicted Boring Machine Utilization

INTERIM PERFORMANCE REPORTS

DAYS SINCE COMMENCED EXCAVATING 59.121

HEADING POSITION(FEET) 5092.339

AVERAGE ADVANCE RATE(FEET/OPERATING HOUR) 11.833

MAXIMUM DAILY ADVANCE

DAYS SINCE COMMENCED EXCAVATING 32.000

HEADING POSITION-BEGINNING OF THAT DAY(FEET) 2553.534

HEADING POSITION-END OF THAT DAY(FEET) 2671.309

AVERAGE RATE FOR THAT DAY(FEET/OPERATING HOUR) 15.099

CUMULATIVE SUBSYSTEM PERFORMANCE..

ELEMENT/MEASURE	OPERATIONAL AVAILABILITY*	SUBSYSTEM UTILIZATION**
ROCK FRAGMENTATION	.376	.300
MATERIALS HANDLING	.955	.300
GROUND SUPPORT	.975	.447
ENVIRONMENTAL CONTROL	1.000	1.000

*OPERATIONAL AVAILABILITY=(TOTAL SHIFT TIME-DOWN TIME)/TOTAL SHIFT TIME

**SUBSYSTEM UTILIZATION=(TOTAL SHIFT TIME-DOWN TIME-IDLE TIME)/TOTAL SHIFT TIME

INTERIM COST REPORT

DAYS SINCE COMMENCED EXCAVATING 59.121

AVERAGE COST PER IN SITU CUBIC YARD EXCAVATED 31.42

AVERAGE COST PER FOOT OF TUNNEL 154.52

CUMULATIVE COST SUMMARY TO DATE..(DOLLARS)

ELEMENT/CATEGORY	DIRECT LABOR	PLANT + EQUIPMENT	JOB MATERIALS	PERMANENT MATERIALS	OVER HEAD	ELEMENT TOTAL
ROCK FRAGMENTATION	63566.04	191456.89	104006.16	0.00	35912.91	395042.00
MATERIALS HANDLING	138155.29	38928.80	7957.62	0.00	18504.47	203549.19
GROUND SUPPORT	99990.56	408.04	0.00	18304.21	11860.28	130463.09
ENVIRONMENTAL CONTROL	[illegible]	5638.34	432.86	0.00	4680.38	51444.13
CATEGORY TOTAL	343047.45	235832.07	112396.64	18304.21	70958.04	780534.41

Figure 64. Final Performance & Cost Reports (Case 1)

INTERIM PERFORMANCE REPORTS

DAYS SINCE COMMENCED EXCAVATING 61.429

HEADING POSITION(FEET) 5042.339

AVERAGE ADVANCE RATE(FEET/OPERATING HOUR) 11.833

MAXIMUM DAILY ADVANCE

DAYS SINCE COMMENCED EXCAVATING 35.000

HEADING POSITION-BEGINNING OF THAT DAY(FEET) 2526.355

HEADING POSITION-END OF THAT DAY(FEET) 2644.660

AVERAGE RATE FOR THAT DAY(FEET/OPERATING HOUR) 15.099

CUMULATIVE SUBSYSTEM PERFORMANCE..

ELEMENT/MEASURE	OPERATIONAL AVAILABILITY*	SUBSYSTEM UTILIZATION**
ROCK FRAGMENTATION	.352	.299
MATERIALS HANDLING	.967	.289
GROUND SUPPORT	.976	.484
ENVIRONMENTAL CONTROL	1.000	1.000

*OPERATIONAL AVAILABILITY=(TOTAL SHIFT TIME-DOWN TIME)/TOTAL SHIFT TIME

**SUBSYSTEM UTILIZATION=(TOTAL SHIFT TIME-DOWN TIME-IDLE TIME)/TOTAL SHIFT TIME

INTERIM COST REPORT

DAYS SINCE COMMENCED EXCAVATING 61.429

AVERAGE COST PER IN SITU CUBIC YARD EXCAVATED 32.58

AVERAGE COST PER FOOT OF TUNNEL 158.11

CUMULATIVE COST SUMMARY TO DATE..(DOLLARS)

ELEMENT/CATEGORY	DIRECT LABOR	PLANT + EQUIPMENT	JOB MATERIALS	PERMANENT MATERIALS	OVER HEAD	ELEMENT TOTAL
ROCK FRAGMENTATION	66151.84	191456.89	104706.16	0.00	36231.49	398546.38
MATERIALS HANDLING	143552.59	39153.25	8307.55	0.00	19101.34	210114.74
GROUND SUPPORT	103790.72	408.04	0.00	18254.21	12245.30	134698.27
ENVIRONMENTAL CONTROL	42946.36	5038.34	428.37	0.00	4441.31	52854.38
CATEGORY TOTAL	356441.51	236056.52	113442.09	18254.21	72419.43	796613.76

Figure 65. Final Performance & Cost Reports (Case 2)

5038 ft. The Case 1 simulation predicted 59.1 days and Case 2 61.4 days, a variance of less than 10%.

This variance can be accounted for by the small differences between the boring machine performance and utilization as predicted by the model and as experienced in the field (Table 54).

TABLE 54

COMPARISON OF PERFORMANCE RESULTS FOR THE LAYOUT TUNNEL SIMULATION (5038 ft OF ADVANCE)

	Average Advance Rate (ft/hr)	Boring Machine Utilization	Time to Advance 5038 ft (days)	VARIANCE FROM PROJECT EXPERIENCE Due to Boring Rate (days)	VARIANCE FROM PROJECT EXPERIENCE Due to Utilization (days)
PROJECT RECORDS	11.5	0.283	64.5	-	-
SIMULATION CASE 1	11.8	0.300	59.1	.5	4.9
SIMULATION CASE 2	11.8	0.289	61.4	.5	2.6

Thus the agreement between the model and the field information is close, and the cause for variance can be accounted for by the following sensitivities of the model. The average advance rate of the boring machine is sensitive to the performance curve derived in this study (Sec. III-C-3). Boring machine utilization is most affected by the assumed availability of the machine and its interaction with other elements of the excavation system.

The cumulative cost summaries generated by the model are also given in Figs. 64 and 65. Case 1 predicts a unit cost of $32/yd^3 and Case 2 a unit cost of about $33/yd^3. Both these estimates are below the $49/yd^3 which may be deduced from the Bureau of Engineer's estimates for the equivalent work and material. Although some of the costs which are included in the engineer's estimated (e.g., prorated mobilization cost) were not included in this example and would account for part of this variance, it is believed that the major part is due primarily to the uncertainty of the cost data which was supplied by the authors to the model as input. This data was based largely on estimates of representative labor crews, base wage rates, machinery cost, and depreciation rates used on similar projects rather than on a more detailed accounting of the costs associated with this particular example. It is clear that because the nature of the cost model is to function mainly as an accounting tool, accurate cost estimates are incumbent on accurate input data rather than on empirical cost-estimating equations. The two cases considered for this example represent preliminary cost estimates and reflect a minimum amount of data gathering prior to the running of the simulation.

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