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Optimizing underground, trackless loading and hauling systems

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OPTIMIZING UNDERGROUND, TRACKLESS

LOADING AND HAULING SYSTEMS

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ABSTRACT

Determining the optimum system and equipment to be utilized in today's underground trackless haulage mine is a complex problem which was dealt with in this research by systems simulation. The computer program developed by the Caterpillar Tractor Company served as the starting point for simulating the equipment's capabilities to move material over a course in a prescribed time. Extensive time studies of the unsimulated events lead to structuring this program in the deterministic mode since it was felt to be more acceptable to the operating management which would eventually use the technique. Every phase of the simulation was successfully validated before going to the next phase of the program. After identifying six systems of moving ore with a front end loader, a truck and/or ore pass with a feeder-chute, formulas describing the relationship of four of the systems were developed. The unique approach of generating simulated production and cost tables, which were both printed and stored, allowed the users to accurately determine optimum conditions either by using hand calculators, or by using the computer. A multi-purpose program was written, allowing three different operating approaches to the optimization problem. Numerous actual case studies were optimized and the results are given. General prupose Optimum Trackless Materials Moving Charts were developed to correctly define equipment and method for all cases that will be encountered in the normal practices found in most modern trackless underground mines.

Acknowledgment

There are many individuals and organizations that are worthy of acknowledgment, not only for the assistance given in the preparation of this dissertation, but in the development of the particular graduate program which lead to the opportunity to present the dissertation. Just as this engineer is now grateful, other industry oriented engineers of the future will be grateful to the University of Missouri-Rolla for recognizing the need and the potential of establishing the Doctor of Engineering program. It establishes opportunity and objective for the continuing education of the engineer dedicated to improving his own ability and his company's technical advantage. Likewise, this engineer is grateful to his employer, the St. Joe Minerals Corporation, not only for the opportunity to report the findings of the research in this dissertation, but who also recognized the value of continuing education several years ago and developed a program that sponsored a large portion of the graduate work that remained to be completed.

More specifically, the help and assistance of the candidate's Graduate Committee and the Head of the Department of Mining, Petroleum and Geological Engineering, who helped solve the multitude of problems necessary to complete the requirements of this new graduate program. Particular appreciation is expressed to the candidates advisor, Dr. James Scott for his hours of counsel, his words of encouragement and his technical advice. To Sharon Grayson, who did the typing, to Duane Bowen, who helped with some of the drawings, to David Weiss, Steve Petty and Ken Kuebler who assisted in the computer programming and the research effort, this researcher will be forever grateful. Last on the list, but most certainly the most deserving of warm appreciation is the researcher's wife, Ruth and his family who for many years have had the patience and faith to support his effort in spite of the hours of neglect that were necessary to complete this work.

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P_r = Production Rate—or—Required Tonnage (tons/Hour)

L_c = Loader Capacity to LHD (Tons/Hour) (PT)

L_{co} = Loader Capacity to FEL (Tons/Hour) (PT)

L_t = Loader Cycle Time (Minutes) (PT) (Including Loading)

L_n = Number of Loaders

$L_\$$ = Loader Cost to LHD (S/Ton) (CT)

$L_{\$o}$ = Loader Cost to LHD—Number of Units Rounded Up ($/Ton)

D_c = Loader Dipper Capacity (Tons)

T_c = Truck Hauling Capacity (Tons/Hour) (PT)

T_n = Number of Trucks

$T_\$$ = Truck Hauling Cost ($/Ton) (CT)

T_{bc} = Truck Bed Capacity (Tons)

T_t = Truck Cycle Time (Minutes) (PT) (Haul and Dump only)

$S_\$$ = System Cost ($/Ton)

$S_{\$a}$ = System Cost—Number of Unite Rounded Up ($/Ton)

S_c = System Production Capacity (Tons/Hour)

Source of information: Production Table (PT); Cost Table (CT)

1.90 = This is a correction factor used to account for the cost of operating a vibrating feeder-chute.

.04 = This is a correction factor used to account for additional maintenance cost caused by loading ore into a truck in the stoping area, rather than by a chute.

Any additional subscripts of 1,2,3, or 4 implies that this formula applies only to that method.

LHD - Load Haul Dump Method (Subscript 1 in Formulas)

HFC - Haul From Chute Method (Subscript 2 in Formulas)

FEL - Front End Load Method (Subscript 3 in Formulas)

LAF - Load and Follow Method (Subscript 4 in Formulas)

LAM - Load at Mid-Point

LHD(II) - Load Haul Dump, to and from Chute

± - Throughout this paper, this is used to denote the standard deviation from the mean, of all of the data taken.

I. Introduction

A. The Development of the Problem

Unless one is very familiar with the characteristics of the heavy equipment used to move the production material in a modern, trackless underground mine, the process would seem so simple that it would be a waste of time to spend any engineering talent in planning the best method of doing it. But in truth, the systems available today to move mine production, while they have become very efficient and productive, determining how to achieve optimum productivity and cost, has become extremely complex. The complexity of the situation will become evident later. But to review how the situation has developed, one needs to look at methods of loading and moving mine production of the past.

In the early 1920's, there were several equipment manufacturers and mines that were trying to develop machines to replace the hand shovelers in the mines. But the prevalent use of mechanical loaders did not really get started world-wide until the mid 1930's. Traditionally, the early loaders were either "fixed-point" shovels, or their movement was restricted to the rail haulage system. They were at first either air powered or electrical powered. By the early 1950's rubber tired hauling equipment, powered either by internal combustion engines or electrical motors, were being introduced into the underground haulage systems. Meanwhile, the loading equipment being used, while it became larger and more efficient, was still confined to the railroad system or traveled very slowly on steel treads and was essentially still a fixed point loader. However, when these loaders were used in combination with the trackless rubber tired trucks or shuttle cars, it brought on new flexibility to most mining systems and allowed ore bodies and coal seams to be reached and mined more quickly and cheaply. With this new flexibility the mine engineer no longer had to be limited to 2 or 3 percent grades to reach his objective. This brought in industrial engineers with their time and motion studies to help determine the optimum conditions at which the trackless equipment would perform at its best.

Meanwhile, other developments had taken place which seemingly had little connection with underground mining. In 1939, a 1/3-cubic yard mobile front end loader was introduced by "Hough." (1) It was only meant for light applications such as rehandling loose stockpiled material. Known as the HS (Hough-Small) Loader, its single, vertical lifting cylinder, and its little front tires, characterized it as a "motorized wheelbarrow." By the early 1940's, the rubber tired wheel loader was beginning to resemble today's machine: the operator was moved up towards the front of the machine where he could see, the engine was moved back over the drive axles for traction and to balance the load, and two side mounted boom arms replaced the vertical mast arrangement. But this equipment was still not to appear in underground mining for several years. Meanwhile, steel tread-mounted shovels, both of the "overshot" and "gathering arm" variety had become much larger and somewhat more mobile. In general, when used to load most ores, these models had very high productivity while operating, but equally high maintenance cost and down time.

In 1953 [2] the operating people of the American Zinc Company developed the first self-loading hauler using the small Allis Chalmers HD-6B tractor as a power unit and the same size crawler tracks beneath the loader-hauler: thus, the "Gismo" was created. [3] It hauled 5 to 6 tons, had a top speed of 4.4 mph and could negotiate +12%, -20% grades. [4] By 1958 Sanford-Day had acquired the rights to build the self-loading transporter, which would be driven by a rubber tired, diesel powered Wagner Mixermobile power unit. [2] Thus, the first "Transloader" was built and the practice of "load-haul-dump" or "LHD" took a giant step forward. It is also interesting to note that another diesel powered, rubber tired, self-loading hauler was granted a patent back in 1955. [5] This patent even incorporated hydrostatic drive, which for underground mining equipment, was well ahead of its time. However, it was designed to haul only 2-cubic yards at the rate of 6 mph and to negotiate 15% grades.

But it was the Transloader that was to set records in productivity. Their capacity ranged from 4.5 up to 18 tons, but their real asset when operated on good straight roads, was that they could travel up to 20 mph. Their use spread to many districts, but no where were they used more efficiently than they were in the mines of St. Joe Minerals Corporation in Southeast Missouri. Their operators were on an incentive bonus system and tonnages of 5 to 6 hundred tons per loader shift, hauled between 2 and 3 thousand feet were normal. Most of the rock was loaded at the face and hauled all the way to the ore pocket at the shaft. But where the haulage grade became too steep for fast efficient travel, the ore was dropped through an ore pass and was stockpiled on the main level where it was reloaded and hauled to the ore pocket. Where haulage was expected to exceed 3000 feet, and there was to be considerable tonnage moved (over a million tons) an efficient chute arrangement, incorporating a vibrating feeder, was installed to load 28 ton trucks. Perfecting the "Trans-loader" system took place between 1963 and 1969, with no improvements taking place after that time.

This is not to say that other trackless loading and hauling equipment was not being built and being perfected elsewhere. Indeed, the Wagner Telescopic trucks that were first developed for White Pine Copper were of major significance, as well as the Wagner Scooptram loaders developed in the mid 1960's that were to become the types of LHD unit that would spread throughout the world and revolutionize mining methods. These units were considered basically as low profile, front end loaders, and were primarily built to either load into low profile trucks, or LHD short distances. Such units were also tried in the St. Joe mines in 1965, and it was found that where haulage ways were crooked, narrow, rough and the distance was short, this type of unit out performed the Transloader. Where these conditions prevailed and the haulage distance was long (say 2000 feet) then this type of unit, loading a low profile truck was the most efficient.

Wagner introduced another concept with this combination of equipment; "load and follow." This meant that the front end loader would load the truck, the truck would start towards the ore pocket, then the loader would fill its bucket and follow the truck to the ore pocket. Unfortunately, while the concept was good, the elongated, low profile front end loader being built at that time was just not designed to travel in these conditions nearly as fast as the truck. Therefore, it did not prove successful, at least in St. Joe's mines, and the Transloader remained the dominant prime mover until about 1970. About the same time as the Transloader was introduced as an articulated loader, the concept of building standard front end loaders as articulated vehicles was catching on. Small standard front end loaders were already being used in limited applications underground as early as 1966[(6)]. But it wasn't until they became articulated that they had the needed maneuverability, stability and speed to attract the interest of the underground operators. At this point, most equipment manufacturers started producing multiple sizes of these units from about a 3-cubic yard dipper capacity to 10-cubic yard dipper capacity. Another major transition had also taken place within this type unit. The flywheel horsepower available, per inch of bucket cutting edge had more than tripled.[1] This is a rough index of the loader's "breakout" or digging ability. This feature represents the transition of this equipment from a machine that would be primarily designed to handle stockpiled material, to a machine that would handle rough, shot rock in a tight underground mining heading.

Testing of these highly flexible maneuverable, fast and extremely powerful machines started in underground trackless mines in the late 1960's and early 1970's. In 1969, the first Caterpillar 980 loader was put underground at St. Joe's Fletcher Mine. It was recognized that the standard Caterpillar bucket on this unit was too small to be successful as a LHD unit. Therefore, a Balderson bucket was installed on the loader. The machine performance matched that of the TL-70 Transloader. It was then decided to try to go one step further and put a Caterpillar 988 loader in the Fletcher Mine. St. Joe had to redesign the linkage and the dipper for this machine to develop a 10 ton capacity for the LHD application. Yet, this change did not detract from the same loader acting as a front end loader, loading a truck. Since these innovations were developed, numerous 988s and 980s have been put into the St. Joe Southeast Missouri mines. Meanwhile, trucks of the 28 to 40 ton class have also been added. This is the setting that poses a very complex optimization problem.

B. The Scope and Complexity of the Problem

One of the most pressing challenges for today's trackless mine operations in trying to achieve maximum efficiency, is moving the ore from the stope where it was broken to the shaft where it is dumped. The problem of optimization is brought on by the very same features of the equipment that caused it to be selected in the first place. That is, the versatility, the flexibility and the tremendous power of today's trackless mining and construction equipment not only creates a multitude of possibilities for both equipment selection as to size and brand, but also to application for each specific job that must be done. There has always been some degree of flexibility in trackless mining and this is the primary reason why it developed. But with fixed point loading of some type of mechanical shovel into a given type of conveyance such as a truck or shuttle car, the flexibility was extremely limited by today's standards. But so were the decisions as to how the equipment was to be utilized. Then came the development of the load-haul-dump (LHD) concept with equipment that loaded only itself and hauled to the dump point. Here again, the equipment is extremely flexible in use but is not extremely versatile in application. If you use only equipment that will load itself, then there are only two practical ways to move the ore. Either LHD all the way to the shaft or dump it at an intermediate ore pass and pick it up below with another LHD and haul it to the shaft. Therefore, it was no problem of deciding the most efficient application of the equipment. There was, of course, a question of how far you could LHD and still achieve a minimum cost. More recently, the concept of LHD has been expanded to include equipment that can either front end load (FEL) or load-haul-dump.

Utilizing this equipment in conjunction with trucks for hauling part of the distance and/or an ore pass and a vibrating feeder, you now have expanded the possible practical methods of moving the ore from the face to the shaft to at least six ways. Even after the equipment is selected and is being used, the mining conditions change each shift by an ever changing length of haul and the variable condition encountered by following the ore horizon. For any given mine load-haul condition, i.e., distance, grade, road conditions, operator practices, etc., there is only one least cost method.

When the equipment must be selected prior to mining and one must consider which of the six ways should be used to yield the least cost and consider all of the likely brands of loaders and trucks, then the problem becomes uncomprehensible for easy analysis and monumental for a straight forward engineering hand calculation. A method of accurately determining the optimum method of materials moving for four of these methods is the objective of this research.

There are other areas of the material moving methods for mining that are beyond the scope of this study, but which should receive equally as much attention and study. Methods of moving material by converyor belt and by railroad car are still very predominant methods of moving ore underground around the world, and can be effectively utilized in conjunction with LHD units.

C. The Industrial Signifiance of the Problem

In a recent survey by a major magazine[(7)] on trackless mining utilizing LHD equipment, questionnaires were sent to 560 mines supposedly representing 94% of the non-coal production of the Western World. Approximately 32.5% of the questionnaires were returned and indicated that 35% "of the world underground production is represented by this figure." Of the production of the answering mines, Livingston states that 75% was handled by the trackless methods. If one assumes that the questionnaire revealed a sample of world-wide underground production, then the total tons now being handled LHD equipment would approach 3/4 of a billion tons. Assuming that only 25% of this tonnage was not being mined by the optimum method, and that by reaching this optimum method, the cost would be reduced by as much as 10¢ per ton, the savings would be $18,500,000 per year. Yet, some of the comparisons of this research study have shown a difference of as much as 70¢ to 80¢ per ton revealing a theoretical potential of $130 to $150 million a year. Whether the amount to be saved by optimizing trackless equipment is $10 or $100 million, it is significant enough to justify the efforts that are being expended.

II. Review of the Relevant Related Operations Research Effort on Similar Problems

A. General Review

There is such a tremendous volume of literature that has been written on system analysis, operation research, and computer applications relating to loading and hauling ore in all types of mines, that an effort will be made in this review to discuss only those works which relate most directly with the approach that has been used here.

One of the earliest methods of this approach in the operations research-mineral industry literature is found in a 1960 University of Arizona short course in a paper by Drevdahl.(8) While the basis of determining the ownership and operations cost is covered in detail, as well as the other essentials such as vehicle rimpull, tractive ability, rolling and grade resistance, the principle mathematical development and how the equipment's operating characteristic's can predict the performance of haulage equipment is only eluded to in his and Padans(9) work as presented at the short course. Drevdahl suggests that these details "are adequately explained in [the manufacturer's] reference handbooks on equipment." He also suggests that these "equipment manufacturers have programmed the performance of their equipment on various typical job situations." In fact, the next reference to this method of equipment simulations was in a paper by E. L. Gibbs(10) discussing the technuques of General Motors in using a deterministic modeling approach to predict the capabilities of their equipment. In 1965 this author discussed the same technique with WABCO engineers and, infact, was given a computer-generated performance study of the ore-moving capabilities using their trucks on the Viburnum surface ore haul road.[11] A similar effort was reported on the system analysis approach for truck and shovel selection by L. W. Gibbs.[12] The work by Morgan and others of Caterpillar Tractor, was first brought to the attention of this researcher by the thesis work of Thieme[13] in 1968. In this work, the simulation was that involving motorized scraper units loading and hauling strippable material for the Twin Buttes project. While Thieme configured his own "system simulator," the "scraper simulator" was one developed by Caterpillar Tractor Company. It was significant to this research because it simulated for the first time (at least in public literature) a unit piece of equipment in a load-haul-dump situation. Even though the equipment was entirely different, as was the material, the method of loading and dumping, the validity of the simulation principle for analyzing the potential of the load-haul-dump problem was obvious. However, in Thieme's work, "the loading time and queue lengths [were] controlled" which for the simulation model used in this research does not apply. Morgan and Peterson's work using the stochastic simulation was published[14] in 1970 and in part is the basis of this simulation study. Other works which have also used this approach in simulating mining activities are those of Mutmansky,[15] simulating a train hauling muck away from a tunneling machine and the work of Zambas and Yegulalp[16] simulating underground truck haulage. In a Russian paper by Vasil´ev, et. al., the same technique as applied by Caterpillar was developed. They illustrated the "analytical method involving direct integration of the differential equation of motion of the truck obtained by substituting expanded values of the forces" which is essentially the same as Equation (1).

In the development of the underlying techniques of the dynamic simulation of the haulage cycle that is used in the present work, the author wishes to make clear that the basic method and the equipment parameters of their construction equipment was furnished by Caterpillar Tractor Company and is known as "Travel Time and Earthmoving Production Computer Program".[17] Since this research effort was previously done by others, the explanation of how it functions will be included in this review, rather than in the body of this report. Other equipment manufacturers, including Terex, Envirotech (Eimco) and Wagner were also most cooperative in furnishing equipment parameters to be used in the simulation study.

B. The Equipment Simulator Theory Reviewed

As was mentioned above, all of the major equipment manufacturers, especially of trucks, have developed equipment simulators which will calculate haul and return times for their equipment, with any given load and on any given haulage grade. Such programs can be used to compare different equipment in the same situation, or the same equipment in different situations. In either case, the optimum desired results can be observed through the simulation effort of the computer program. There are, of course, several assumptions, limitations and implied rules that are applied to any given problem. In the present case and in its simplest context, the objective is to calculate the total time to start from a stopped position, accelerate, reach a maximum velocity, decelerate (brake) and then stop. Then supposedly the process is reversed back to the original position. The calculation of the elapsed travel time for the cycle is the basic problem at hand. The underlying principle is, of course, the same as that of any dynamic problem; Newton's second law involving Force (F), Mass (M) and Acceleration (A);

$$F = M A$$

However, the force which is available for acceleration (A_F) or deceleration is the difference between the tractive force (T_F) (known as rimpull) and the total of all of the forces resisting the motion, all divided by the mass of the unit (M_u). These resisting forces are the rolling resistance between the tires and the road bed (R_R), the air resistance (W_R) and the grade resistance (G_R) which may either be (+) or (-). Thus:

$$A_F = \frac{T_F - (R_R + W_R \pm G_R)}{M_u} \qquad (1)$$

However, the unit's mass must be accelerated both linearly as well as rotationally for those components that do rotate. When the unit's acceleration is applied over a small measure of time (say 0.1 to 1.0 second), the resulting increase in velocity, the distance traveled under those conditions and the time elapsed are all determined. At the new velocity, the acceleration is reapplied, which results in a new velocity, distance traveled and elapsed time. By recording each of the increments and repeating the process over and over until a specified distance of the given course is completed. Summation of these increments gives the time to complete the course and the return time if it is specified.

The mass that must be accelerated, as mentioned above, must be accelerated in rotation as well as linearly in the direction of machine travel. But when a machine part is being accelerated by the applied torque, a portion of that torque is absorbed by each of the various rotating parts in the driving system. Therefore, the torque being delivered to an engine's flywheel is greater than that delivered from the flywheel, by some difference, which is proportional to the flywheel inertia and radial acceleration. The net effect of this is, that the steady state force that can be obtained by the unit at any given speed is greater than the force delivered by the wheel to the ground to accelerate the mass of the unit. Morgan states that "this factor is most pronounced in the lower gears [of construction machinery] where the actual force delivered to the ground can be less than one-half of the steady state force." Since the effects of this diminish in the higher gear ratios, overall acceleration (deceleration) for a complete cycle is not drastically affected. But the surest way to account for this rotary inertia (and the technique used here), is to add a correction factor to the mass of the unit and thus the assumption is that with this mass correction factor, (MCF) the force for acceleration remains equal to the steady-state rimpull. However, this mass correction factor varies considerably with different brands of equipment, with the different gear reductions between the rotating mass and the wheel, as well as with the different types of drives (mechanical, torque converter or electrical). Therefore, a user of this type of program should obtain a mass-correction-factor curve for the specific equipment that is to be simulated. Caterpillar shows the MCF for torque converter drives as ranging between 0.6 and 0.4 at 5 mph and dropping to 0.05 at 25 mph. This is shown in Figure 1.

To further perfect the accuracy of the simulation, braking (deceleration) was given limiting values, consistent with operator practice. They developed the relationship, that the braking rate (BR) should be 6 feet per second squared, less 20% of the total resisting forces (R_T). This is expressed as a formula:

$$BR = 6\ Ft/Sec^2 - (R_T x .2)$$ (2) and is shown in Figure 2.

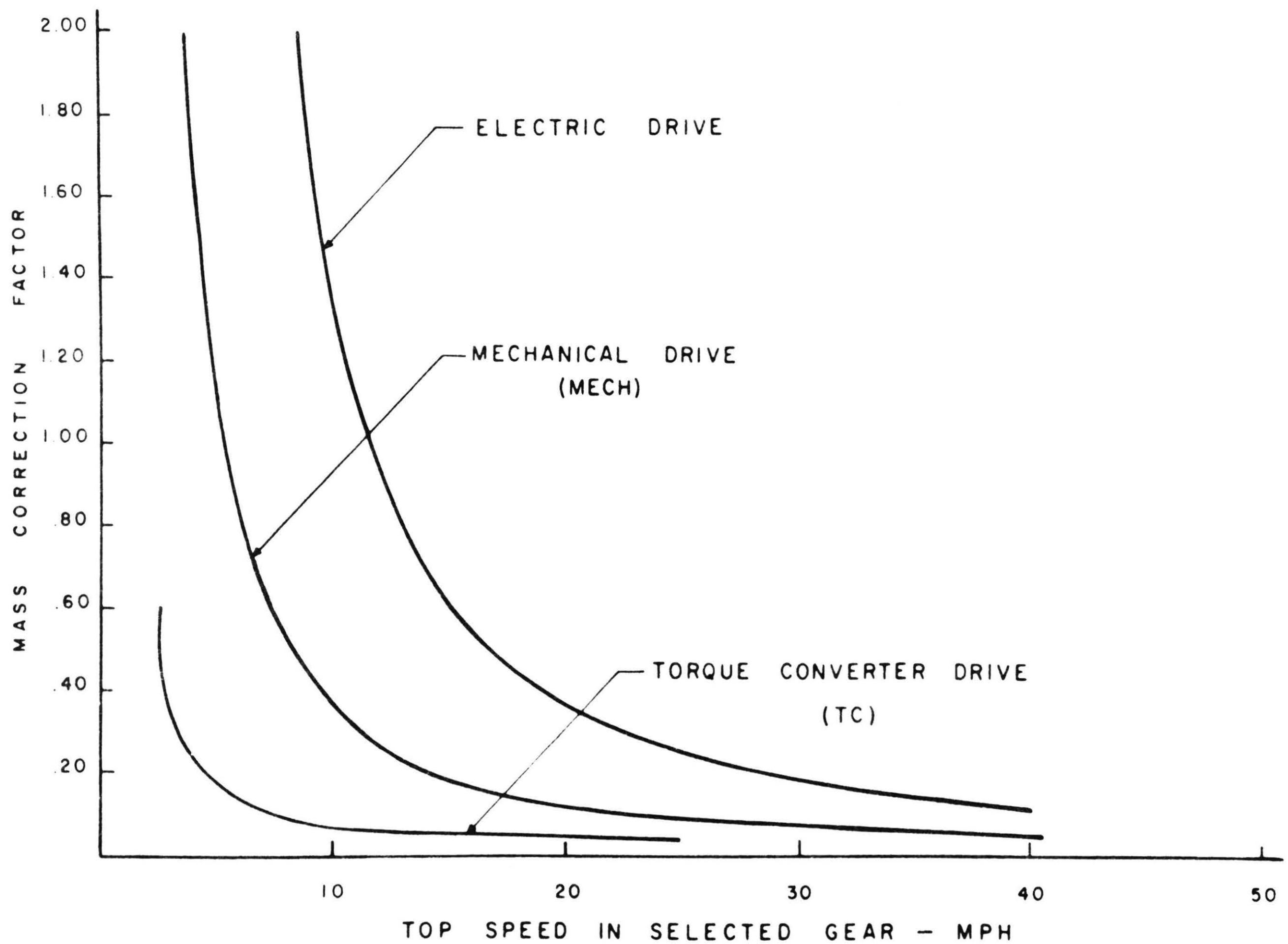

FIGURE-1 APPROXIMATE MASS CORRECTION FACTOR (AFTER CATERPILLAR)

ASSUMED BRAKING RATE FOR VEHICLE SIMULATION PROGRAM

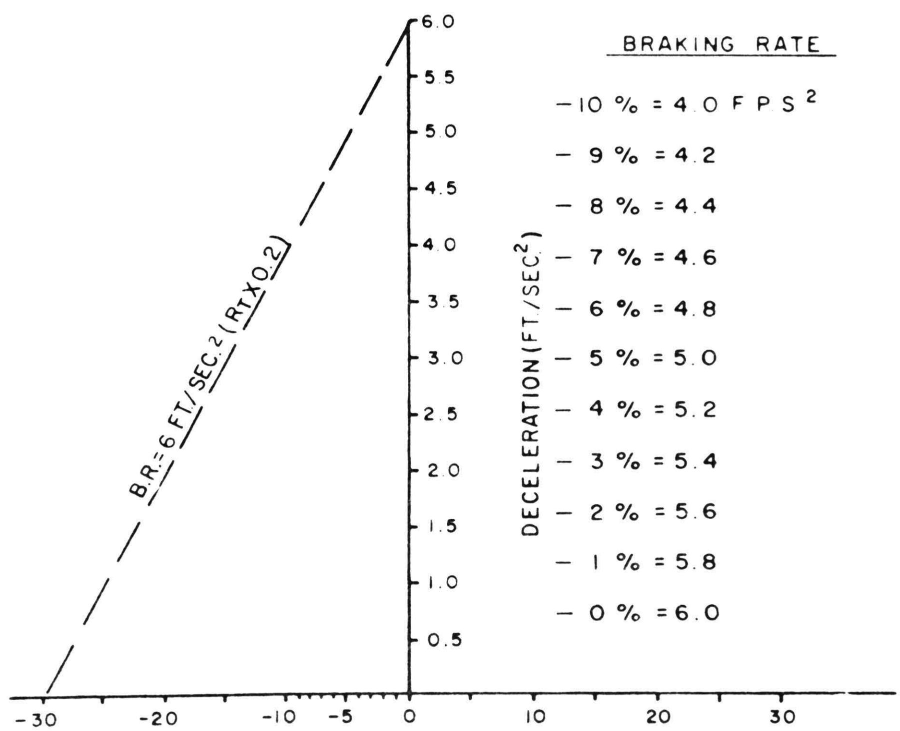

FIGURE-2. ASSUMED BRAKING RATE FOR VEHICLE SIMULATION PROGRAM (AFTER CATERPILLAR)

The resistance forces are normally put into terms of equivalent percent grade. This has been normal practice in construction equipment calculations for years. Wind or air resistance is normally insignificant up to about 40 mph on the surface. However, in a mine drift or tunnel the larger equipment might very well act as a piston at velocities over 15 or 20 mph. But in the simulation work presented here, since the velocity was limited to 15 mph, the "A_R" would probably have only slight effect. In Caterpillar's research, they determined that the combined effect of air resistance (W_R) and rolling resistance (R_R) could be empirically determined. Both tire-ground rolling resistance and air resistance vary with speed and when combined, were assumed to increase .025% mph or 1% from 0 to 40 mph. The rolling resistance at "0" mph (R_i) as a percent, is of little significance since machines do not operate there, but it serves as a reference for the rolling resistance at the normal operating range (R_N) in percent. Their research developed the following relationship:

$$R_i = -\ 0.90\% + 1.075\ R_N \qquad (3)$$

but is limited to a maximum value of R_N = 12% at which time, "R_i" = R_N. In the program, the "R_i" is then added to the percent grade, "R_G" and is used to calculate the sine of the effective grade angle. This would then be the effective vertical distance through which the unit's mass would have to be accelerated. The basic logic of the above is best shown as a flow chart. This chart is shown in Figure 3.

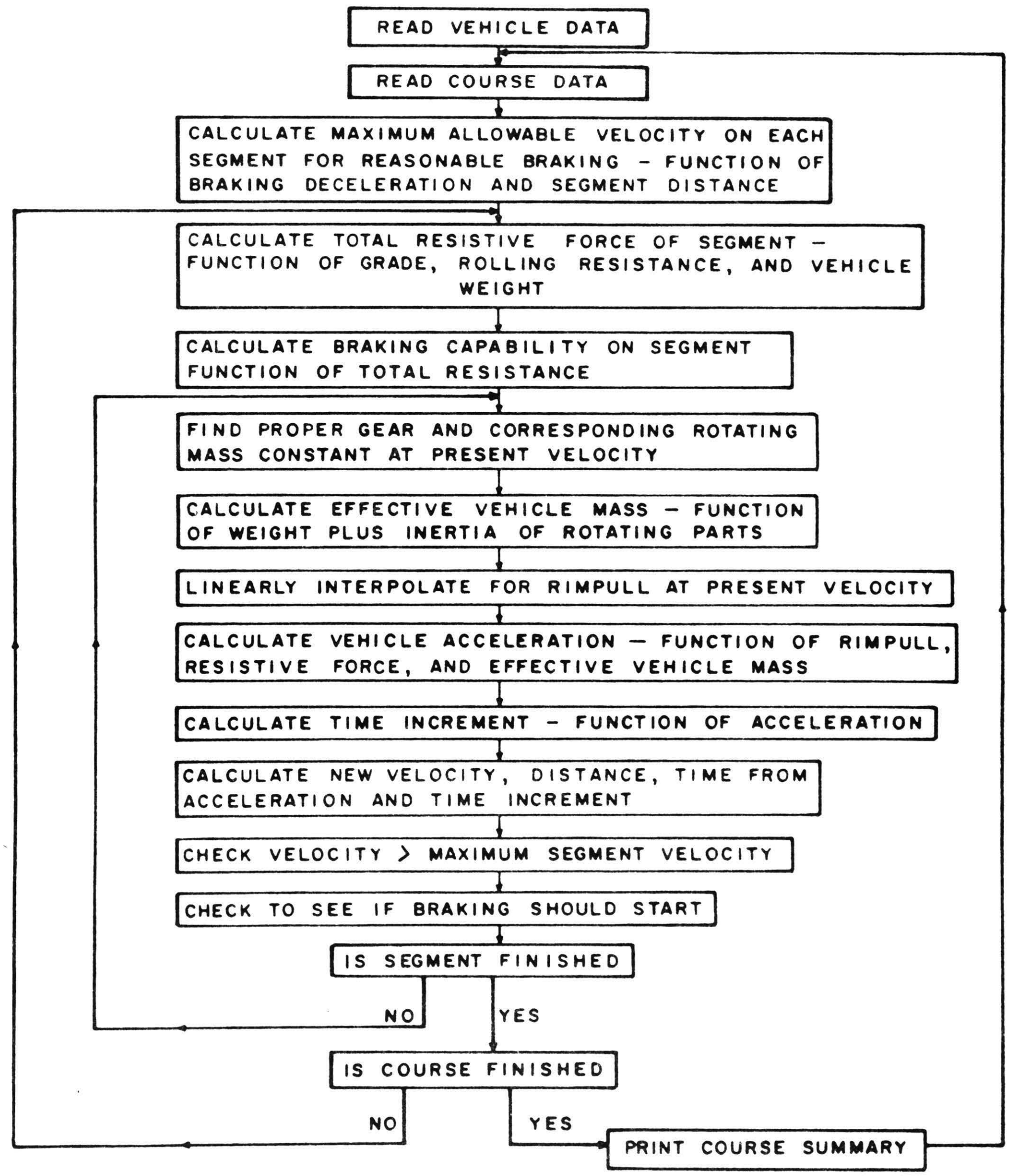

FIGURE-3 FLOW CHART OF EQUIPMENT SIMULATOR PROGRAM (AFTER CATERPILLAR)

III. The St. Joe Materials Moving Program Development

A. The Initial Program Development and Modification of the Caterpillar Program

After examining the Caterpillar, "Travel Time and Earthmoving Production Program" it was believed to perform the basic function that it was meant to do: that is, calculate the elapsed time that it takes a vehicle of a given specification to complete a given course. While this was not the specific objective of this research, it was felt that in order to test the concept of simulating LHD equipment in an underground environment, much of the original program could be utilized, in this first phase of the work. Most of the basic assumptions were maintained, but some of the limiting values were changed. A discussion of the input data and how it was obtained is now presented.

1. Equipment Descriptions

   a. Specifications

      During the entire project; seven different front end loaders (LHD) units, from four different manufacturers were simulated and three different size trucks from a single manufacturer were simulated. For the first phase of this research, the specifications of only one vehicle was needed and is shown on Table 1.* This shows the necessary data that

*Appendix A contains the other equipment specifications.

TABLE I EQUIPMENT DATA SHEET**

Equipment Designation	988
Type of Drive	3 Speed Power Shift
Tire Size	29.5 x 29
Size and Type of Bucket	10 Ton St. Joe Bucket
Payload (Lbs)	20,000
Empty Weight (Lbs)	72,000
Shift Time (Sec)	0

Rimpull-Velocity Curve

Velocity (MPH)	Rimpull (Lbs)	Velocity (MPH)	Rimpull (Lbs)
0.50	53500	9.00	6500
1.00	48000	10.00	6000
1.20	45000	11.00	5400
1.50	38500	12.00	4900
2.00	31000	13.00	4500
2.50	24000	14.00	4200
2.90	19200	15.00	3800
3.20	18300	16.00	3500
4.00	15300	17.00	3300
4.50	13700	18.00	2900
5.00	12200	19.00	2500
6.00	9800	20.00	1600
7.00	8000	21.00	900
7.50	7600	22.40	0*
8.00	7300		

Shifting Speed - Rotating Mass Constant

(Velocity MPH)	
2.9	0.37
7.0	0.12
22.40	0.05

*The last rimpull listed must be for "0" pounds pull.

**The data as given is correct for the specifications. If there is a change in any part of the power train, a curve multiplying factor must be used to adjust the rimpull curve.

describe the characteristics of the equipment that are needed. Computer formated equipment data sheets are found in Appendix A.

b. Ownership and Operating Cost

Real cost figures are proprietary information and have not been used. The figures that have been included are those which to the researcher seemed to be logical for underground equipment, in open stope, room-and-pillar mines and would correspond to cost figures given in the Mining Engineers Handbook for the same type of mining.[19] One not acquainted with the severity and abuse that underground mining equipment must be subjected to, will take note that in the example, ownership and operating cost given in Table II are nearly twice as high as what is listed in equipment manufacturer's handbooks for surface equipment.

TABLE II HOURLY OWNERSHIP AND OPERATING COST OF LOADER "A"

Ownership Cost:

Depreciation:	
Purchase price	$82090.00
Extras:	
10 Ton Bucket	6000.00
Special Tire	3200.00
Accessories:	
Scrubber-Muffler-OBF	500.00
L-M Alarm System	240.00
TOTAL PRICE	$92030.00
Less original value of tires	(11200.00)
Total amount to depreciate	$80830.00

Depreciation period:

Hours operated per year -

$$\frac{4.5^* \text{ Loaders x } 12 \text{ Hr/Day x } 255 \text{ Day/Yr}}{6 \text{ Loaders}} = 2295 \text{ Hr/Yr}$$

Number of years to write-off - 2295 Hr/Yr x 5 = 11475 Hrs.

Hourly depreciation cost - $80830 ÷ 11475 Hrs = $7.05/Hr.

Average Investment Cost: Int. rate + tax rate + ins. rate:

(.07 + .01 +.01)(92030)(.50) ÷ 2295 = 2.22.

Total hourly ownership cost: $7.05/Hr x 2.22 = $9.27.

Operating Cost:

Tire replacement cost:			
Each tire cost	(800 hours)	$2800.00	
4 recaps cost	(3200 hours)	6000.00	
	4000 hours	$8800.00	
Each tire cost/hr.		$2.20	
Machine tire cost/hr. $2.20 x 4 =			$8.80
Operating Labor:			
(day's pay + fringes + bonus)/(op. hrs./day) =			8.15
(30.24)(1.352) + (8.00) /6			
Operating Supplies:			
(cost/ton)(loading rate/hr.) =			1.47
(8,487/530208 x 91.8)			
Maintenance Labor:			
(main. labor/ton)(loading rate/hr.) =			9.32
(39815 x 1.352/530208)(91.8)			
Maintenance Mat. Excluding Tire Cost:			
(mat. cost/ton)(loading rate/hr.) =			14.89
(85997/530208)(91.8)			
Total Operating Cost (Per Hour):			$42.63
Total Ownership and Operating Cost: (Per Hour)			$51.90

*Though there are 6 loaders in the mine, only 4.5 are operated at one time.

2. Course Description

a. Distance and Grade

For the first phase of this research, the courses were designated within the Fletcher Mine as shown in Figure 4, and were the haul-roads being used by the various loading crews between the operating stopes and the dumping pocket at the shaft. Each route was broken down into its components of distances and grade, for as many segments as was necessary to describe the individual course. A grade "resisting" the loaded travel is considered "+" and a grade assisting the loaded travel is considered "-". The "Haul Road" in the sample of Table III is the course going from the stope with a load of ore; the "Return Road" is the course going from the dumping pockets, empty towards the operating stope. The grade resistance was considered to be equal to 1% of the gross weight times the percent grade. This is equivalent to 20 pounds of resistance per ton of weight times the percent grade.

b. Rolling Resistance

There is no precise manner in which the

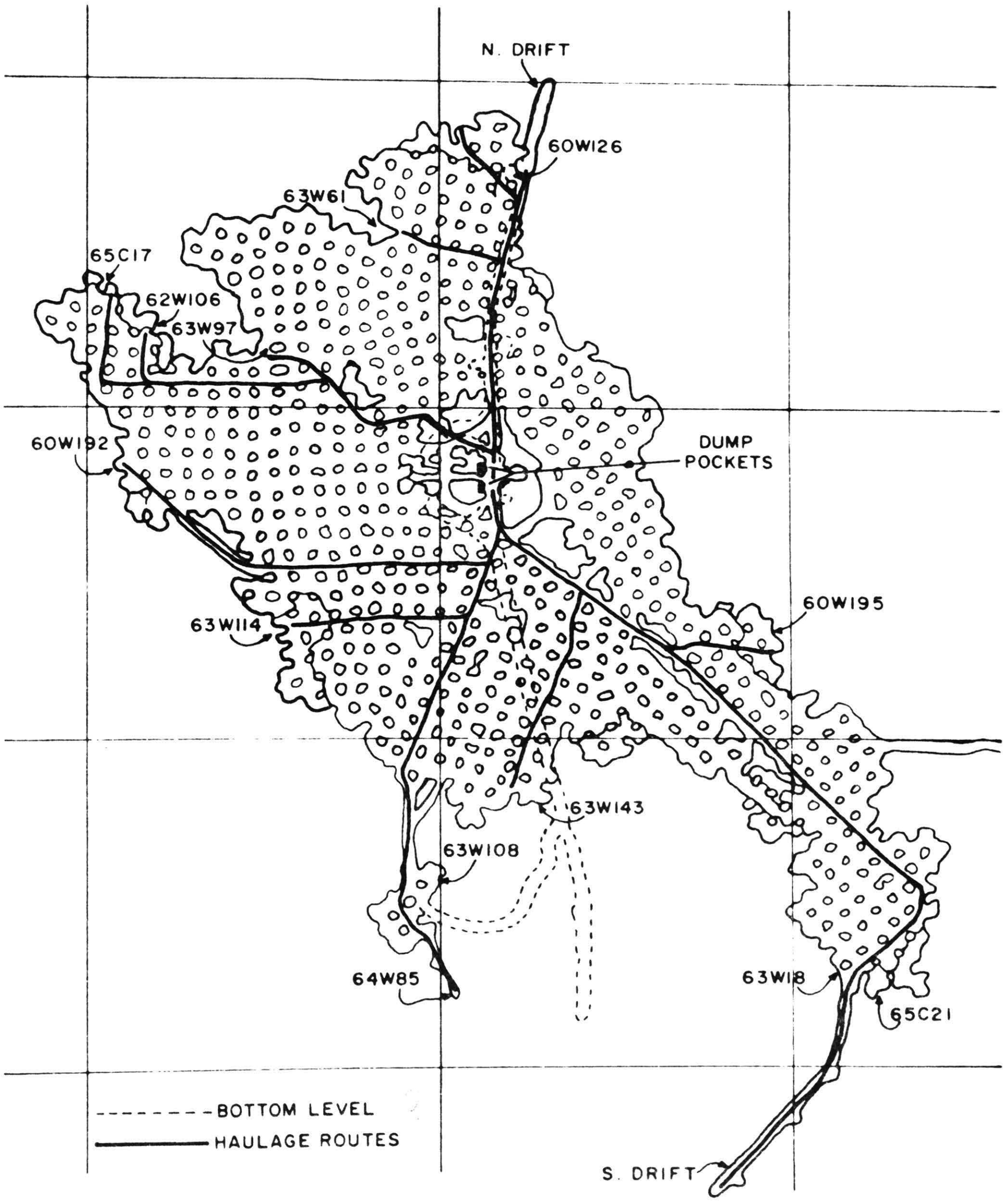

FIGURE-4 THE HAULAGE ROUTES SIMULATED IN THE FIRST PHASE OF THE RESEARCH AT FLETCHER MINE.

TABLE III LHD SIMULATION OF 988 LOADER WITH 10 TON, ST. JOE BUCKET

Payload = 20000 Lbs Empty Weight = 72000 Lbs

Course - 60W 126 To North Grizzly Initial Vehicle Speed - 0.0

Haul Road

Seg No.	Dist (Ft)	Roll Res	Grade	Vel Limit	Maxss, Vel	Top Vel	Last Vel	Accum Time (Min.)
1	340.	5.5	-0.70	0.0	13.72	12.79	12.79	0.38
2	390.	5.0	8.40	0.0	5.12	12.79	5.12	1.17
3	375.	5.0	3.20	0.0	8.07	8.07	8.07	1.72
4	35.	5.0	0.0	0.0	13.26	8.96	0.0	1.79

Course - North Grizzly to 60W 126 Initial Vehicle Speed - 0.0

Return Road

Seg No.	Dist (Ft)	Roll Res	Grade	Vel Limit	Maxss, Vel	Top Vel	Last Vel	Accum Time (Min.)
1	35.	5.0	0.0	0.0	15.97	8.93	8.93	0.06
2	375.	5.0	-3.20	15.00	20.45	15.00	15.00	0.36
3	390.	5.0	-8.40	15.00	22.40	15.00	15.00	0.66
4	340.	5.5	0.70	15.00	13.48	15.00	0.0	0.97

Haul Time - 1.79 Minutes

Return Time - 0.97 Minutes

Fixed Time - 1.46 Minutes

Cycle Time - 4.22 Minutes

Production - 852.44 Tons/Shift

combination of rolling resistance and air resistance can be accurately determined. One rule of thumb is that for each ton of weight, there is 40 pounds of resistance per ton, plus, for each 1-inch of tire penetration into the roadway, there is an additional 30 pounds of resistance per ton. Stated another way, the normal rolling resistance would be:

$$R_N = (2\% + 1.5\%/\text{Inch of Penetration}) \quad \text{Gross Weight}$$

Caterpillar used 4% for normal earth work. But in underground mining, considering the roughness of the stoping areas, the muddy roads in some places and with a slight effect of air resistance in some drifts, the overall rolling resistance used in this research work was R = 5% to 5.5%. Drevdahl[20] gives a table and some basic rules which amount to about the same thing as the formula above.

3. Operating Description

a. Velocity

Because the equipment is operating in an underground mine, there is a definite maximum safe velocity. This, of course, is governed by the condition and straightness of the roadways as well as the overall dimension of the drift and the visibility. In St. Joe's mines, in some rare cases of long, straight, wide and smooth roadways equipment has been timed in excess of 20 mph. But this is not the average condition nor was it with this type of equipment. Therefore, it was thought that the limiting velocity should be 15 mph. The program will simulate the machine accelerating to its maximum velocity in any segment up to the 15 mph limit.

b. Fixed Time

The term "fixed time" throughout this research implies the amount of time that is required to do the task that the equipment simulator does not simulate, within each cycle. There are two general categories of fixed time. Those tasks which are done when the equipment returns to the stope, such as maneuvering into the rock pile, loading the dipper, and laying out boulders too big to pass through the pocket "grizzly." Those fixed times on the dumping end of the cycle are maneuvering into the grizzly dumps, cleaning off the grizzlies occasionally and picking up a boulder off of the grizzly when they are mistakenly brought in with the load. When the project was first conceived, it was anticipated that use of queuing theory would be necessary to resolve any queues that occur at the dump pockets. However, during the time studies that were necessary to arrive at the correct fixed times, it was observed that queues are so rare and of such short duration, that this occasional lost time was simply included in the dump portion of the fixed time.

Time studies were performed on all of the regular LHD operators at the Fletcher Mine for 10 shifts. From these studies, it was determined that the fixed time for the loading portion of the cycle to be equal to 1.16 minutes. Bear in mind that the timing started when the loader turned into the immediate heading and started scraping into the rock. The time stopped when the operator put the loader into forward gear and pulled out of the heading. Fixed time at the dump, including occasional queuing and clearing the grizzly was determined to be only 0.36 minutes. The dump portion of the fixed time was taken from when the unit entered the ore dumping room and stopped when the operator put the loader into the forward gear to leave the dump area. Therefore, the fixed time total for the LHD cycle is 1.52 minutes. It is realized that many computer simulation programs use such data in the development of a stochastic approach to apply the portions of the cycle time to the simulation. However, in this case where numerous actual field time studies could be so easily taken and where there was so little inter-reaction between the operating equipment, that queuing theory was not needed, then the direct approach of simply using an arithmetical average was not only adequate, but was much more easily understood and accepted by "lay" mine operating people.

c. Simulated Time

As stated earlier, the computer simulation records the time of each segment of the course and accumulates them into a "Haul Time" and a "Return Time." To this is added the "Fixed Time" and it becomes the uncorrected "Cycle Time." If real machinery never needed repair and if humans worked as efficient as do computers, then the cycle time derived, divided into 60 minutes, times the units capacity would be the tons per hour produced. However, equipment availability and workman efficiency must be considered. At the time just prior to this study, the equipment availability factor was 0.83 at this mine, so this factor was used. The equipment operator's were all working on an incentive system which means that a portion of their pay is derived from the amount of material that they load and haul during a shift. By observation, their efficiency was judged to be 90%. The normal shift's working hours for these stated conditions of underground work is only six hours. In summary, the operator will produce with the machinery for 44.82 minutes, per hour, for six hours a shift. (268.92 minutes per shift)

B. Validation of the Initial Simulation Program

In order to validate all of the assumptions, the time study work and the method of simulation, the program was used, simulating a two week period of actual operations, for five stoping areas during this period and the actual operators time and tonnage moved from each stope was taken from company records. The simulation produced the Ton Per Hour, which when corrected with the availability and efficiency factors yielded a new Tons Per Hour. When this figure was multiplied by the actual hours of work for each stope, the total tons for the two week period could then be compared to the actual tons for the same period. The result of the validation is shown in Table IV. At the same time, but extending for a full month period, a comparison was generated for the cost per ton of the ore moved. This validation was done in the same manner just described. The operating time that was spent in each stoping area was taken from company records. From these figures, and the simulated rate of production from every single stope that was operated that month, a cost per ton was calculated. Each of these individual costs per ton were weighted by the production of that stope to derive a total mine cost per ton for the LHD operation. This figure was worked out before the actual company cost sheets were produced. The accuracy was remarkable; it was within 1.0% of the actual recorded cost. Unfortunately, this record is confidential and cannot be reproduced. But Table IV illustrates the same degree of accuracy.

The success of these efforts to validate the work up to this point was most gratifying and developed immediate management attention. However, it is fully recognized that this type of validation is the result of many compensating errors as can be seen when Table IV is examined closely. Other comments on this phase of the work will be reserved until the results and conclusion are fully discussed.

C. Development of Multi-purpose Simulation Tables

The previous section described the basic work that was necessary to establish credibility and confidence in the steps which were to follow. The objective was still that of developing a method whereby the optimum system of

TABLE IV ORIGINAL LHD SIMULATION VALIDATION - FLETCHER MINE

STOPE NO.	TONS LOADED AND HAULED	REGULAR OPERATORS OBSERVED RATE (TPH)	SIMULATED RATE (TPH)	% ERROR IN SIMULATED RATE	SIMULATED TONS LOADED AND HAULED	WEIGHTED % ERROR OF ALL TONNAGE	COURSE DESCRIPTION AVE R_G %	DIST
64W85	11350	85.6	81.0	-5.4%	10740		2.00	2500
63W106	4000	97.0	115.3	18.9%	4755		-1.33	1500
63W61	8410	106.4	117.8	10.7%	9311		1.25	1500
65C21	9250	75.1	67.5	-10.1%	8314		-1.90	2500
63W108	4670	88.9	96.3	8.3%	5059		3.85	2000
TOTAL TONS	37680				38179	1.0%		

materials moving with trackless equipment could be identified. Therefore, the next logical step was that of simulating all of the likely pieces of equipment that showed promise to the St. Joe management. The equipment that was simulated is as follows:

Terex	72-71	Loader
Caterpillar	980	Loader
Caterpillar	988	Loader
Caterpillar	992	Loader
Wagner	ST8	Loader
Wagner	ST11	Loader
Eimco	921	Loader
Eject-all	E621	Truck
Eject-all	E631	Truck
Eject-all	E641	Truck
St. Joe Type	Chute-Feeder	

For each of these pieces of equipment, there was an Equipment Data Sheet[21] developed similar to the example shown as Table I. Likewise, for each piece of equipment, an Ownership and Operating Cost sheet was developed[21] to show the hourly cost of the equipment, just as in Table II. All of this information was also entered into the files of the computer program.

One piece of equipment used in the study that has only been briefly mentioned is the vibrating feeder chute. Since it functions as a fixed point loader at the bottom of an ore pass, it's ownership and operating costs had to be tabulated and charged to the simulated cost when the chute system was simulated. Likewise, a time study had to be performed to determine the loading time of the truck. Loading 30 truck loads, averaging 27 tons each, took 1.17 ±.17 minutes with a vibrating feeder and 1.60 ± .25 minutes to load a 40 ton truck.

The next step was that of changing the existing computer program so that it would generate a data file within the computer that would store all of the elements of information needed when each optimization program was encountered. At the same time that these data files were compiled for storage, they were also printed. Though many manufacturers print simulated production tables for their equipment, to this researcher's knowledge, this approach and technique of using the stored array tables is probably unique in the materials-handling research field. There were several reasons for approaching the problem in this manner.

1. The only computer available within this Division of St. Joe Minerals at the time was an IBM 360, Model 25. By developing the program in sections relating to each piece of equipment, much of the program could be written and "debugged" locally.
2. The program that was written for each piece of equipment, generated the basic data and could be run on the local computer at very little expense.
3. This method generated Simulation Tables, for each type of equipment, that were made into a booklet for use by future researchers and mine operating people to solve simple

operating problems. This can now be done very easily without the use of a computer. Since the mines are in a rural Ozark area, the telephone lines are not yet of adequate quality to support data transmission. Therefore, this method freed the user from the need of direct access to the computer, but fit into the mode of "formula and electronic calculator."

4. In the future, costly "Central Processing Unit" time for solving complexed optimization problems would be minimized.

Also, a program was written at this point in the research for calculating the operating cost of each course.[(21)] The Flow Chart is similar to Figure 3. The Simulation Tables were designed to show the capabilities of each of the pieces of mobile equipment, on any course from 250 feet in length to 31690 feet (6 miles) in length. They also ranged from -9 to +17 percent grades in one percent increments. For each piece of equipment and for every given distance and grade, there were three pieces of data generated: the cycle time and the tons per hour were printed in the "Production Tables" and the cost per ton in the "Cost Tables." For the data that were printed in the array tables, the availability and efficiency factors were used as corrections. The computer programs to generate those tables, as well as a complete file of the Simulation

Tables, appear elsewhere.[21] However, a sample of one sheet of the Production Table is shown as Table V. It is interesting to note that the complete set of tables contain over 25,000 items of useable information.

D. Developing Concepts and Formulas for Methods of Moving Materials in a Trackless Haulage Mine

The Simulation Tables, as handy as they are, only give information on that single piece of equipment moving the material by itself. Therefore, the information as it appears, only applies to loaders operating as LHD units or trucks hauling material from a chute (HFC). Mathematical relationships had to be developed between the numbers printed in the tables, based on the logic of interactions between pieces of equipment as they moved the material from the stope to the ore pocket by different methods. Before approaching the formula development, the concepts of the methods as well as the approaches to optimization need to be discussed.

The concept of load-haul-dump has been expanded to include equipment that can either load-haul-dump or front end load. Loaders, with this flexibility, used either as a load-haul-dump unit, or as a front end loader, with trucks and/or a transfer raise with feeders, result in six practical ways to move the ore from the face to the shaft.

Method 1. Load Haul Dump (LHD): The loader fills its dipper and travels to the shaft.

TABLE V SIMULATED PRODUCTION TABLE FOR EJECT-ALL E621 TRUCK WITH 30 TON CAPACITY

Grade	0 Percent		1 Percent		2 Percent		3 Percent	
Haul Distance	Time*	TPH**	Time	TPH	Time	TPH	Time	TPH
250	2.31	581.26	2.33	577.97	2.34	575.00	2.37	566.56
500	2.71	435.30	2.76	487.71	2.79	481.47	2.85	472.02
750	3.09	387.70	3.17	424.52	3.23	415.92	3.32	404.65
1000	3.47	349.53	3.58	375.84	3.67	366.25	3.81	352.56
1250	3.85	318.20	4.00	335.77	4.11	327.17	4.29	313.57
1500	4.23	292.02	4.41	304.57	4.55	295.64	4.78	281.36
1750	4.60	269.82	4.83	278.67	4.99	269.65	5.25	255.96
2000	4.98	250.76	5.25	256.02	5.42	247.87	5.74	234.08
2250	5.36	234.22	5.66	237.47	5.86	229.34	6.22	216.23
2500	5.74	219.72	6.09	220.82	6.30	213.38	6.71	200.41
2750	6.12	206.91	6.50	206.88	6.74	199.51	7.18	187.18
3000	6.50	206.91	6.91	194.60	7.18	187.32	7.66	175.59

*Time - Minutes

**TPH - Tons Per Hour

Throughout the program, this system is referred to as Method "1".

Method 2. Load-Haul-Dump, Haul From Chute (LHD-HFC): The loaders, as LHD units, haul to a transfer raise and trucks, working on a lower level, draw from the transfer raise and haul to the shaft. Throughout the program, this system is referred to as Method "2".

Method 3. Front End Load (FEL): The loader fills the bed of one or more trucks in the heading and the trucks haul to the shaft while the loader remains at the face. Throughout the program, this system is referred to as Method "3".

Method 4. Front End Load - Load And Follow (FEL-LAF): A loader fills the bed of one truck as a FEL then fills its own dipper and follows the truck to the shaft. Throughout the program, this system is referred to as Method "4".

Method 5. Load At Mid-Point (LAM): One, or more, loaders fill their dippers and load one or more trucks at a mid-point, determined by balancing cycle times for the combination, and the trucks continue to the shaft.

Method 6. Load-Haul Dump-II (LHD-II): The loader fills its dipper and hauls to a transfer raise (without a feeder), another loader picks up the ore at the bottom of the raise and continues on to the shaft

In the research work covered in the report, two methods of materials handling have not been developed. They are "Load at Mid-Point" and "Load-Haul-Dump II." Eventually in future work, these two methods should be included.

When equipping a mine to utilize these six methods, one must not only consider the various brands and sizes of loaders and trucks, but also the number of each unit that will balance the desired production to minimize the cost. Thus far, the variables that have been considered and accounted for are:

The variable mine haulage conditions (grade, distance, road conditions, equipment availability, operators efficiency, hours per shift).

The various brands and sizes of equipment (in this case, seven loaders and three trucks).

The six methods of moving ore in a trackless mine with loaders, trucks and transfer raises.

But there are still other variables which must be understood before proceeding with the optimization problem. Namely, there are at least three ways that the problem can be approached to achieve the maximum usefulness, depending on the application. Each of these approaches will be described and the proper formula developed to calculate the production and cost data for four of the six methods of ore movement.

1. Production Planning

The objective of this approach is that of determining the amount of production and the method which will result in the optimum cost for the equipment specified.

The assumption here is that the individual (Mine Captain, Superintendent, Engineer, etc.) has already determined a particular mine condition (haulage distance and grade) and, knowing what equipment is, or may be, available, needs to determine the production capability and related cost of that equipment under those conditions. The answers can be calculated using the booklet of Simulation Tables and a desk calculator. The values given in the tables on trucks include a production rate, cost and time for the unit doing HFC work. The formulas given below corrects loading time and any mismatch of equipment which may result in idle time. The derivation of these formulas is found in Appendix B, and definitions of variable names are found in Figure 5, as well as in the Addendum in the front of this paper. Also found in Appendix B, are a few examples of the formulas worked out.

P_r = Production Rate—or—Required Tonnage (Tons/Hour)

L_c = Loader Capacity to LHD (Tons/Hour) (PT)

L_{co} = Loader Capacity to FEL (Tons/Hour) (PT)

L_t = Loader Cycle Time (Minutes) (PT) (Including Loading)

L_n = Number of Loaders

$L_{\$}$ = Loader Cost to LHD (\$/Ton) (CT)

$L_{\$o}$ = Loader Cost to LHD—Number of Units Rounded Up (\$/Ton)

D_c = Loader Dipper Capacity (Tons)

T_c = Truck Hauling Capacity (Tons/Hour) (PT)

T_n = Number of Trucks

$T_{\$}$ = Truck Hauling Cost (\$/Ton) (CT)

T_{bc} = Truck Bed Capacity (Tons)

T_t = Truck Cycle Time (Minutes) (PT) (Haul and Dump Only)

$S_{\$}$ = System Cost (\$/Ton)

$S_{\$a}$ = System Cost—Number of Units Rounded Up (\$/Ton)

S_c = System Production Capacity (Tons/Hour)

Source of information: Production Table (PT); Cost Table (CT)

1.90 = This is a correction factor used to account for the cost of operating a vibrating feeder-chute.

.04 = This is a correction factor used to account for the additional maintenance cost caused by loading ore into a truck in the stoping area, rather than by a chute.

Any additional subscripts of 1,2,3, or 4 implies that this formula applies only to that method.

Figure 5 Definitions of Variable Names Used In Production and Cost Formulas

a. LHD: Any number of loaders

$$P_{r-1} = L_c \times L_n \tag{3}$$

$$S_{\$-1} = L_{\$} \tag{4}$$

b. LHD-HFC: Any number of loaders and trucks

Use the smaller P_{4-2}

$$P_{r-2} = L_c \times L_n \quad P_{r-2} = \frac{1030 \quad T_{bc} \; T_n}{(23T_t + T_{bc})} \tag{5}$$

$$S_{\$-2} + L_{\$}\left[\frac{L_c \times L_n}{P_{r-2}}\right] + \frac{1030 \quad T_{\$} T_{bc} T_n}{P_{r-2}(23T_t + T_{bc})} + \frac{1.90}{P_{r-2}} \tag{6}$$

c. FEL: One loader and any number of trucks

$$P_{r-3} = \frac{T_c \times T_n}{\left[\frac{T_t + 1.16 \quad (T_{bc}/D_c)}{T_t}\right]} \tag{7}$$

$$P_{r-3} - L_{co}$$

$$S_{\$-3} = L_{\$o}\left[\frac{L_{co}}{P_{r-3}}\right] + T_{\$}\left[\frac{T_c \times T_n}{P_{r-3}}\right] + 0.04 \tag{8}$$

d. FEL-LAF: one loader and one truck

$$P_{r-4} = \left[\frac{44.82}{1.16 \; (T_{bc}/D_c) + L_t{*}}\right] (T_{bc} + D_c) \tag{9}$$

*Larger of L_t or T_t

$$S_{\$-4} = \left\{\frac{P_c L_{\$} + T_{bc}\left[\frac{T_{\$} T_c}{P_{r-4}} + L_{\$o}\right]}{T_{bc} + D_c}\right\} + 0.04 \tag{10}$$

2. Situation and Mine Planning

The objective of this approach is that of determining the amount and the type of equipment needed which will optimize the cost of the production specified. The assumption in this case is that the individual is faced with the problem of selecting equipment to be used in a new mine, or an expansion of an existing mine, knowing the required production rate and the haulage conditions of distance and grade which will prevail. The answers can be calculated using the simulator tables and the formulas given below. "Situation Planning" allows the use of fractional pieces of equipment, which might be possible to achieve when considering the expansion of an existing mine where equipment is available for part of the working shift, provided it is kept busy during the remainder of the shift elsewhere in the mine. "Mine Planning" only allows the use of whole pieces of equipment and adjusts the cost of hauling according to the idle time which will result from not having the equipment utilized to its full capacity.

a. LHD:

$$L_n = \frac{P_{r-1}}{L_c} \tag{11}$$

$$L_\$ = \text{Direct from table} \tag{12-S)*}$$

$$L_{\$a} = \frac{L_\$ L_c L_n}{P_{r-1}} \tag{12-M)*}$$

b. LHD-HFC:

$$L_n = \frac{P_{r-2}}{L_c} \tag{13}$$

$$T_n = \frac{P_r\ (23T_t + T_{bc})}{1030\ T_{bc}} \tag{14}$$

$$S_\$ = L_\$ + T_\$ + \frac{1.90}{P_{r-2}} \tag{15-S}$$

$$S_{\$a} = \frac{L_\$ L_c L_n}{P_{r-2}} + \frac{1030\ T_\$ T_{bc} T_n}{P_{r-2}\ (23T_t + T_{bc})} + \frac{1.90}{P_{r-2}} \tag{15-M}$$

c. FEL: Limited to one loader

$$T_n = \frac{P_r}{\left[\frac{44.82}{1.16\ (T_{bc}/D_c) + T_t} \times T_{bc}\right]} \tag{16}$$

$$T_{max} \leq \frac{1.16\ (T_{bc}/D_c) + T_t}{1.16\ (T_{bc}/D_c)} \tag{17}$$

$$S_\$ = \frac{L_{\$o} L_{co}}{P_{r-3}} + \frac{T_\$ T_c T_n}{P_{r-3}} + 0.04 \tag{18 S\&M}$$

*S - Situation Planning Formula; M - Mine Planning Formula

d. FEL-LAF: Limited to one loader and one truck

$$S_c = \frac{44.82}{S_t*} \quad (T_{bc} + D_c) \tag{19}$$

$$*St = 1.16\ (T_{bc}/D_c) + T_t \text{ or } 1.16\ (T_{bc}/Dc) + L_t$$

(Whichever is larger)

$$S_c \geq P_r$$

$$S_\$ = \frac{D_c L_\$ + T_{bc}\left[\frac{T_\$\ T_c}{S_c} + L_{\$o}\right]}{T_{bc} + D_c} + 0.04 \tag{20-S}$$

$$S_{\$a} = S_c \left\{ \frac{\dfrac{D_c L_\$ + T_{bc}\left[\dfrac{T_\$ T_c}{S_c} + L_{\$o}\right]}{(T_{bc} + D_c)}}{P_{r-4}} \right\} \tag{20-M}$$

Complete derivations of these formulas are found in Appendix B. All three of these planning methods have been programmed in Fortran IV for use on the IBM 360 model 50 at the University of Missouri-Rolla. The simulated production and cost tables for all the equipment considered to date are stored on tapes at UMR. The programs that were developed, compute and write out the desired information. The flow charts for this program are shown in Figures 6a thru 6d. If one is using the production planning program, then after specifying the type of equipment available, and the distance and grades for each of the four methods that one wishes to compare, then the program will develop the systems product in (TPH), cost ($/ton), and cycles per hour. It also prints the loader and truck data that it used in the analysis. If one is using either the Situation Planning or

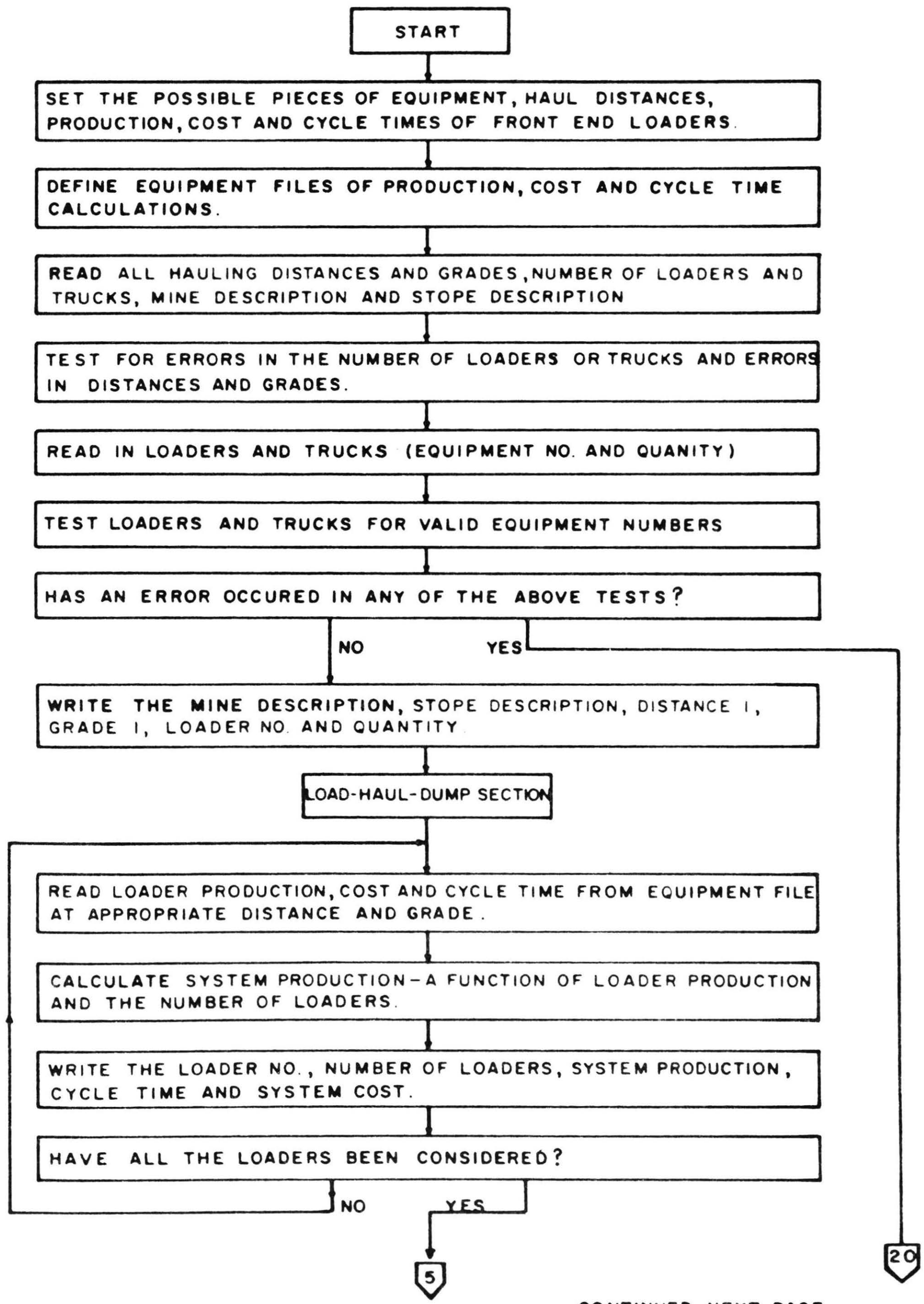

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FIGURE-6a FLOW CHART OF PRODUCTION PLANNING PROGRAM

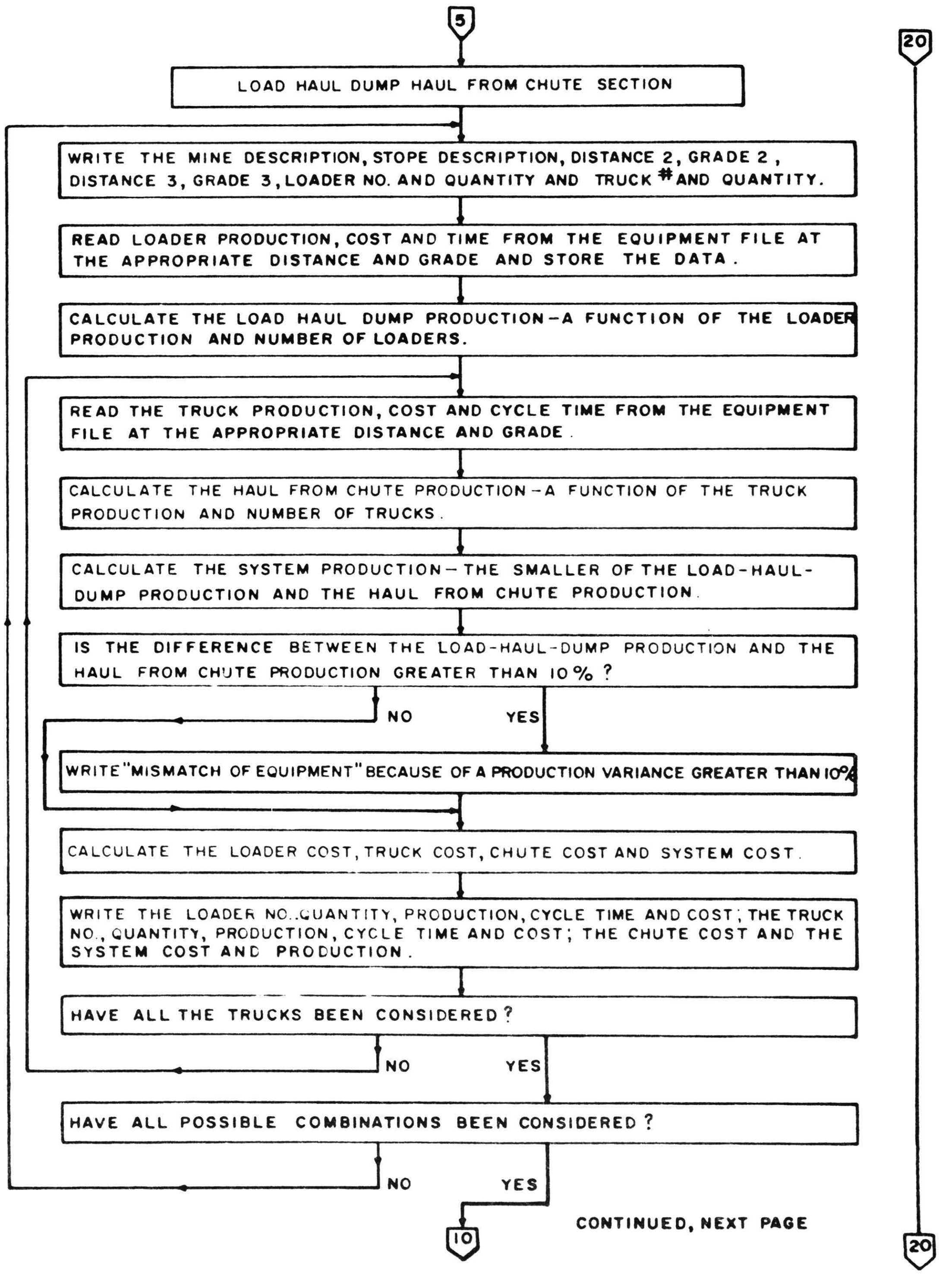
5
20
LOAD HAUL DUMP HAUL FROM CHUTE SECTION
WRITE THE MINE DESCRIPTION, STOPE DESCRIPTION, DISTANCE 2, GRADE 2, DISTANCE 3, GRADE 3, LOADER NO. AND QUANTITY AND TRUCK # AND QUANTITY.
READ LOADER PRODUCTION, COST AND TIME FROM THE EQUIPMENT FILE AT THE APPROPRIATE DISTANCE AND GRADE AND STORE THE DATA.
CALCULATE THE LOAD HAUL DUMP PRODUCTION – A FUNCTION OF THE LOADER PRODUCTION AND NUMBER OF LOADERS.
READ THE TRUCK PRODUCTION, COST AND CYCLE TIME FROM THE EQUIPMENT FILE AT THE APPROPRIATE DISTANCE AND GRADE.
CALCULATE THE HAUL FROM CHUTE PRODUCTION – A FUNCTION OF THE TRUCK PRODUCTION AND NUMBER OF TRUCKS.
CALCULATE THE SYSTEM PRODUCTION – THE SMALLER OF THE LOAD-HAUL-DUMP PRODUCTION AND THE HAUL FROM CHUTE PRODUCTION.
IS THE DIFFERENCE BETWEEN THE LOAD-HAUL-DUMP PRODUCTION AND THE HAUL FROM CHUTE PRODUCTION GREATER THAN 10% ?
NO
YES
WRITE "MISMATCH OF EQUIPMENT" BECAUSE OF A PRODUCTION VARIANCE GREATER THAN 10%
CALCULATE THE LOADER COST, TRUCK COST, CHUTE COST AND SYSTEM COST.
WRITE THE LOADER NO., QUANTITY, PRODUCTION, CYCLE TIME AND COST; THE TRUCK NO., QUANTITY, PRODUCTION, CYCLE TIME AND COST; THE CHUTE COST AND THE SYSTEM COST AND PRODUCTION.
HAVE ALL THE TRUCKS BEEN CONSIDERED ?
NO
YES
HAVE ALL POSSIBLE COMBINATIONS BEEN CONSIDERED ?
NO
YES
10
CONTINUED, NEXT PAGE
20

FIGURE-6b

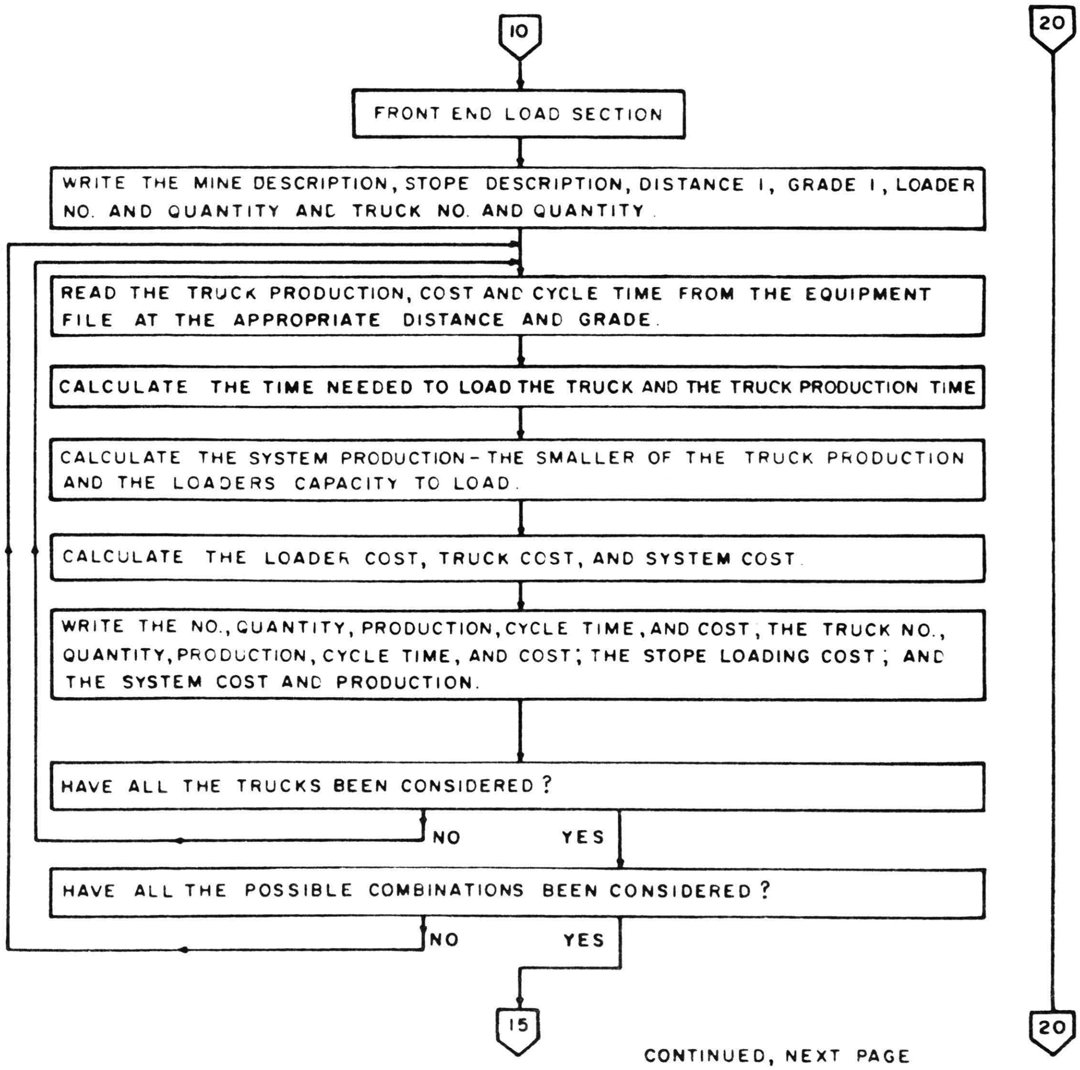
10
20
FRONT END LOAD SECTION
WRITE THE MINE DESCRIPTION, STOPE DESCRIPTION, DISTANCE I, GRADE I, LOADER NO. AND QUANTITY AND TRUCK NO. AND QUANTITY.
READ THE TRUCK PRODUCTION, COST AND CYCLE TIME FROM THE EQUIPMENT FILE AT THE APPROPRIATE DISTANCE AND GRADE.
CALCULATE THE TIME NEEDED TO LOAD THE TRUCK AND THE TRUCK PRODUCTION TIME
CALCULATE THE SYSTEM PRODUCTION - THE SMALLER OF THE TRUCK PRODUCTION AND THE LOADERS CAPACITY TO LOAD.
CALCULATE THE LOADER COST, TRUCK COST, AND SYSTEM COST.
WRITE THE NO., QUANTITY, PRODUCTION, CYCLE TIME, AND COST; THE TRUCK NO., QUANTITY, PRODUCTION, CYCLE TIME, AND COST; THE STOPE LOADING COST; AND THE SYSTEM COST AND PRODUCTION.
HAVE ALL THE TRUCKS BEEN CONSIDERED ?
NO
YES
HAVE ALL THE POSSIBLE COMBINATIONS BEEN CONSIDERED ?
NO
YES
15
20
CONTINUED, NEXT PAGE

FIGURE-6c

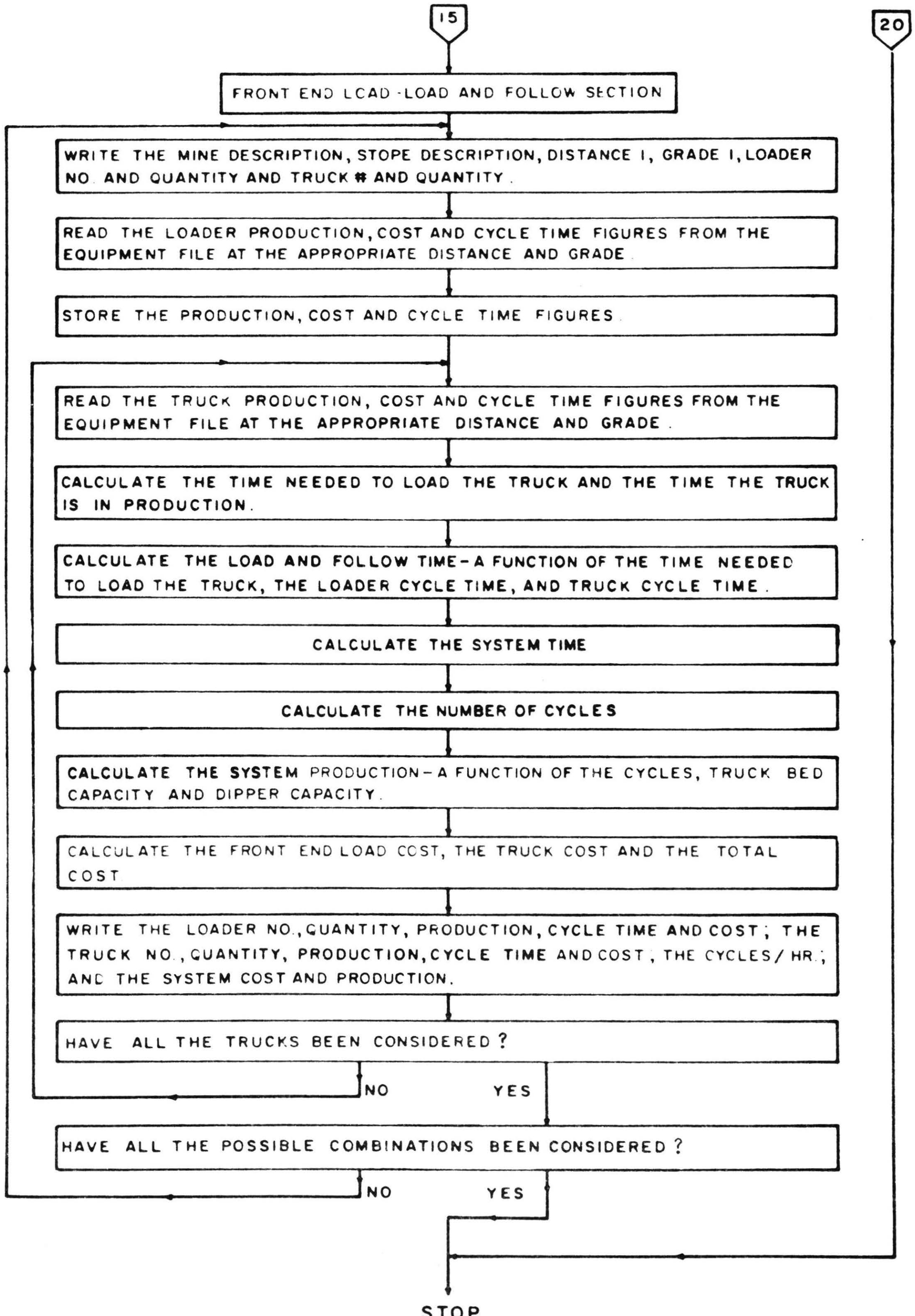
15
20
FRONT END LOAD -LOAD AND FOLLOW SECTION
WRITE THE MINE DESCRIPTION, STOPE DESCRIPTION, DISTANCE I, GRADE I, LOADER NO. AND QUANTITY AND TRUCK # AND QUANTITY.
READ THE LOADER PRODUCTION, COST AND CYCLE TIME FIGURES FROM THE EQUIPMENT FILE AT THE APPROPRIATE DISTANCE AND GRADE.
STORE THE PRODUCTION, COST AND CYCLE TIME FIGURES.
READ THE TRUCK PRODUCTION, COST AND CYCLE TIME FIGURES FROM THE EQUIPMENT FILE AT THE APPROPRIATE DISTANCE AND GRADE.
CALCULATE THE TIME NEEDED TO LOAD THE TRUCK AND THE TIME THE TRUCK IS IN PRODUCTION.
CALCULATE THE LOAD AND FOLLOW TIME-A FUNCTION OF THE TIME NEEDED TO LOAD THE TRUCK, THE LOADER CYCLE TIME, AND TRUCK CYCLE TIME.
CALCULATE THE SYSTEM TIME
CALCULATE THE NUMBER OF CYCLES
CALCULATE THE SYSTEM PRODUCTION-A FUNCTION OF THE CYCLES, TRUCK BED CAPACITY AND DIPPER CAPACITY.
CALCULATE THE FRONT END LOAD COST, THE TRUCK COST AND THE TOTAL COST
WRITE THE LOADER NO., QUANTITY, PRODUCTION, CYCLE TIME AND COST; THE TRUCK NO., QUANTITY, PRODUCTION, CYCLE TIME AND COST; THE CYCLES/HR.; AND THE SYSTEM COST AND PRODUCTION.
HAVE ALL THE TRUCKS BEEN CONSIDERED?
NO
YES
HAVE ALL THE POSSIBLE COMBINATIONS BEEN CONSIDERED?
NO
YES
STOP

FIGURE-6d

Mine Planning approaches to their problem, then after specifying the amount of production required, the grades and distance that will be traveled by the equipment in each of the four methods and which pieces of equipment should be considered. The program then computes the amount of equipment required for the types specified, and prints them as an explicit enumeration in the ascending order of least cost, including all four methods. The flow chart for this program appears in Figures 7a thru 7d. The computer programs for Production Planning, Situation Planning and Mine Planning appear elsewhere.[21] An example of a Mine Planning optimization problem is shown in Table VI. This particular comparison was conducted on only four pieces of equipment; a 10 ton or a six ton loader used with either a 30 ton or a 40 ton truck, (rated capacity).

E. Analysis of Loading and Hauling Problem at Viburnum and Fletcher

Up to this point, all of the needed production and cost data had been generated, along with the formulas necessary for calculating loading and hauling capacities, by the four systems. The program was now ready to be used on real mine problems. At this point in time, management was preparing to order new loaders for the Viburnum Mines. Their existing development plans included the installation of chutes wherever possible when the LHD distance exceeded approximately 3,000 feet. For the Viburnum No. 28 and No. 29

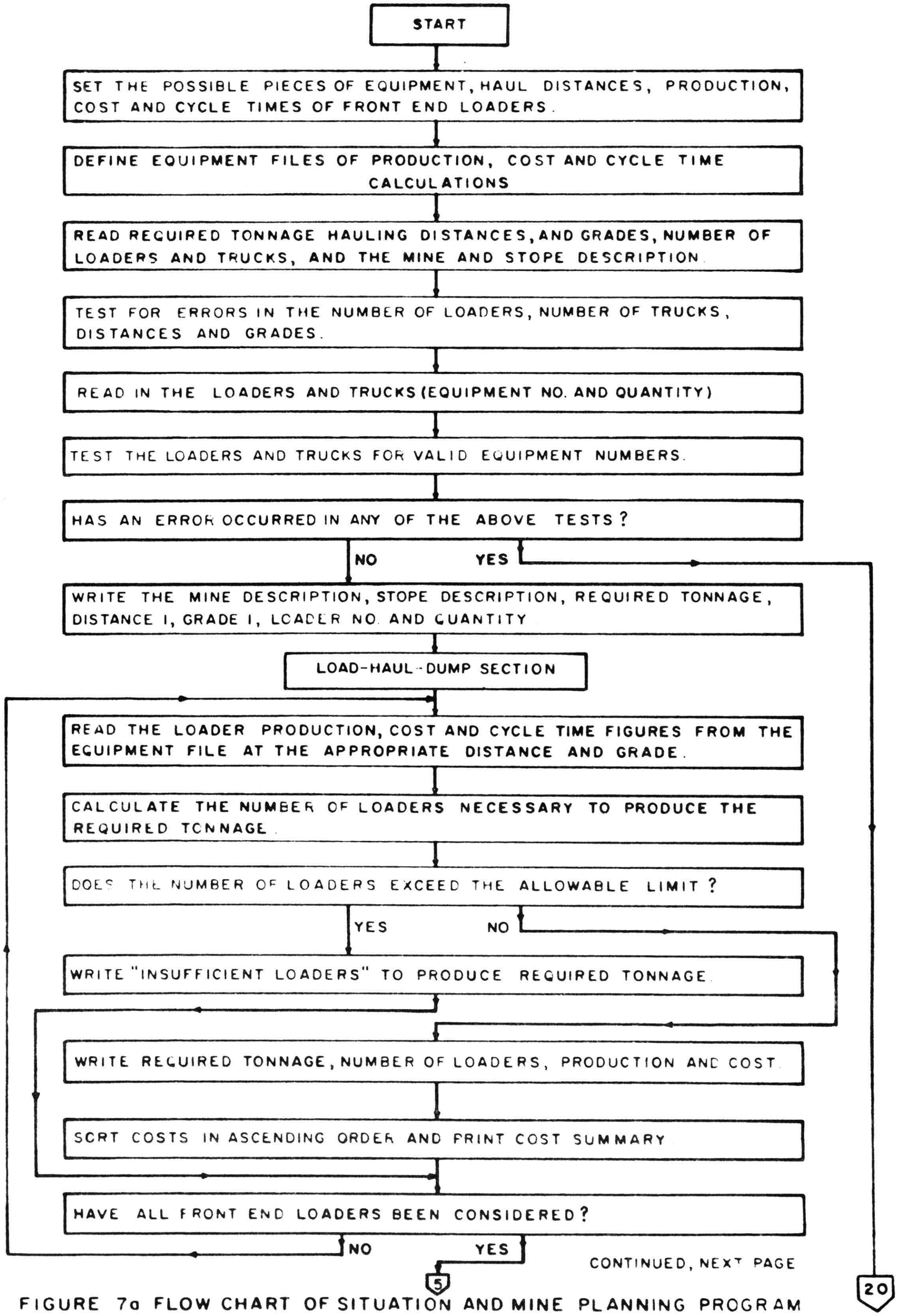

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FIGURE 7a FLOW CHART OF SITUATION AND MINE PLANNING PROGRAM

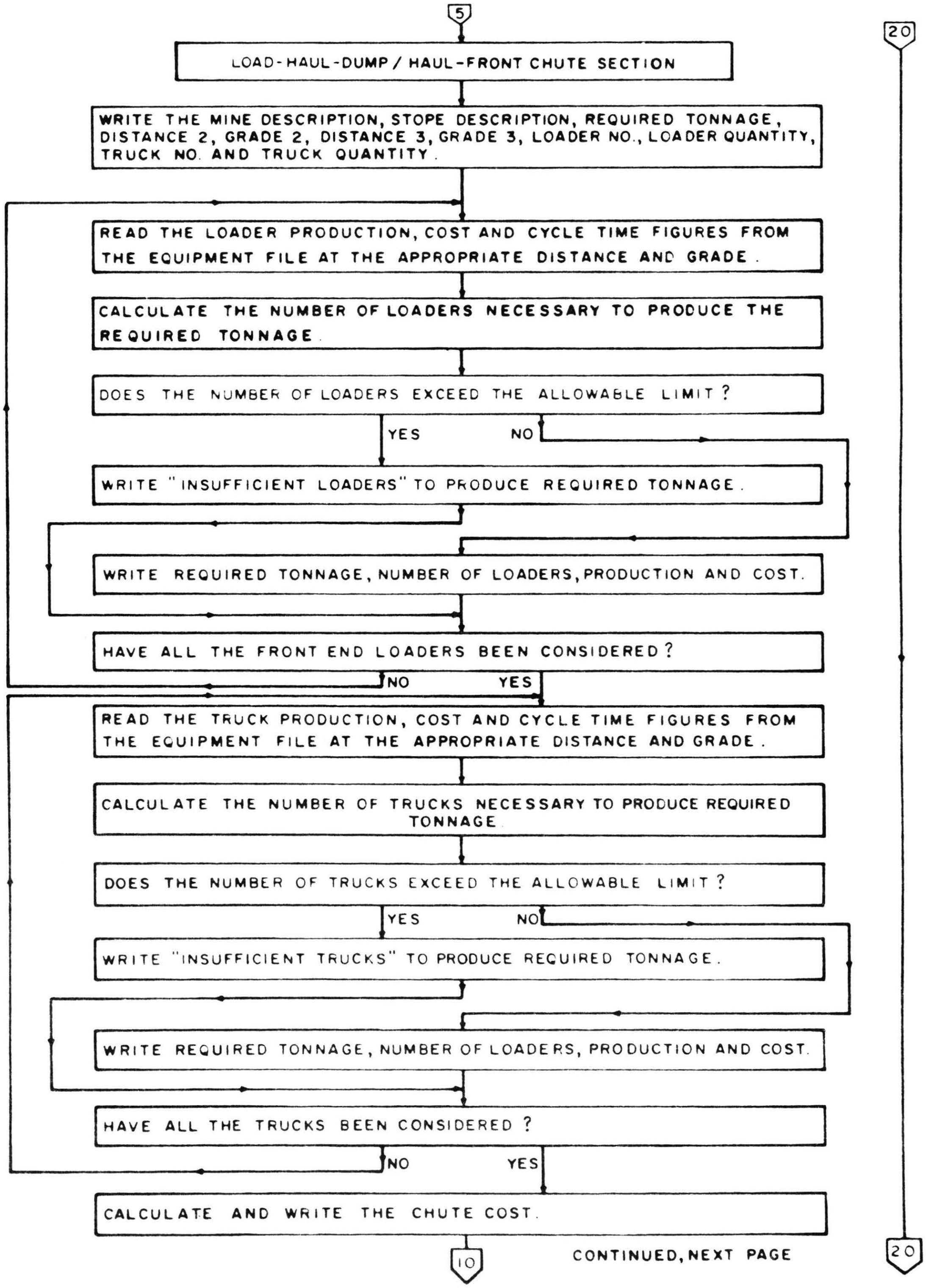
5
20
LOAD-HAUL-DUMP / HAUL-FRONT CHUTE SECTION
WRITE THE MINE DESCRIPTION, STOPE DESCRIPTION, REQUIRED TONNAGE, DISTANCE 2, GRADE 2, DISTANCE 3, GRADE 3, LOADER NO., LOADER QUANTITY, TRUCK NO. AND TRUCK QUANTITY.
READ THE LOADER PRODUCTION, COST AND CYCLE TIME FIGURES FROM THE EQUIPMENT FILE AT THE APPROPRIATE DISTANCE AND GRADE.
CALCULATE THE NUMBER OF LOADERS NECESSARY TO PRODUCE THE REQUIRED TONNAGE.
DOES THE NUMBER OF LOADERS EXCEED THE ALLOWABLE LIMIT ?
YES
NO
WRITE "INSUFFICIENT LOADERS" TO PRODUCE REQUIRED TONNAGE.
WRITE REQUIRED TONNAGE, NUMBER OF LOADERS, PRODUCTION AND COST.
HAVE ALL THE FRONT END LOADERS BEEN CONSIDERED ?
NO
YES
READ THE TRUCK PRODUCTION, COST AND CYCLE TIME FIGURES FROM THE EQUIPMENT FILE AT THE APPROPRIATE DISTANCE AND GRADE.
CALCULATE THE NUMBER OF TRUCKS NECESSARY TO PRODUCE REQUIRED TONNAGE
DOES THE NUMBER OF TRUCKS EXCEED THE ALLOWABLE LIMIT ?
YES
NO
WRITE "INSUFFICIENT TRUCKS" TO PRODUCE REQUIRED TONNAGE.
WRITE REQUIRED TONNAGE, NUMBER OF LOADERS, PRODUCTION AND COST.
HAVE ALL THE TRUCKS BEEN CONSIDERED ?
NO
YES
CALCULATE AND WRITE THE CHUTE COST.
10
CONTINUED, NEXT PAGE
20

FIGURE - 7b

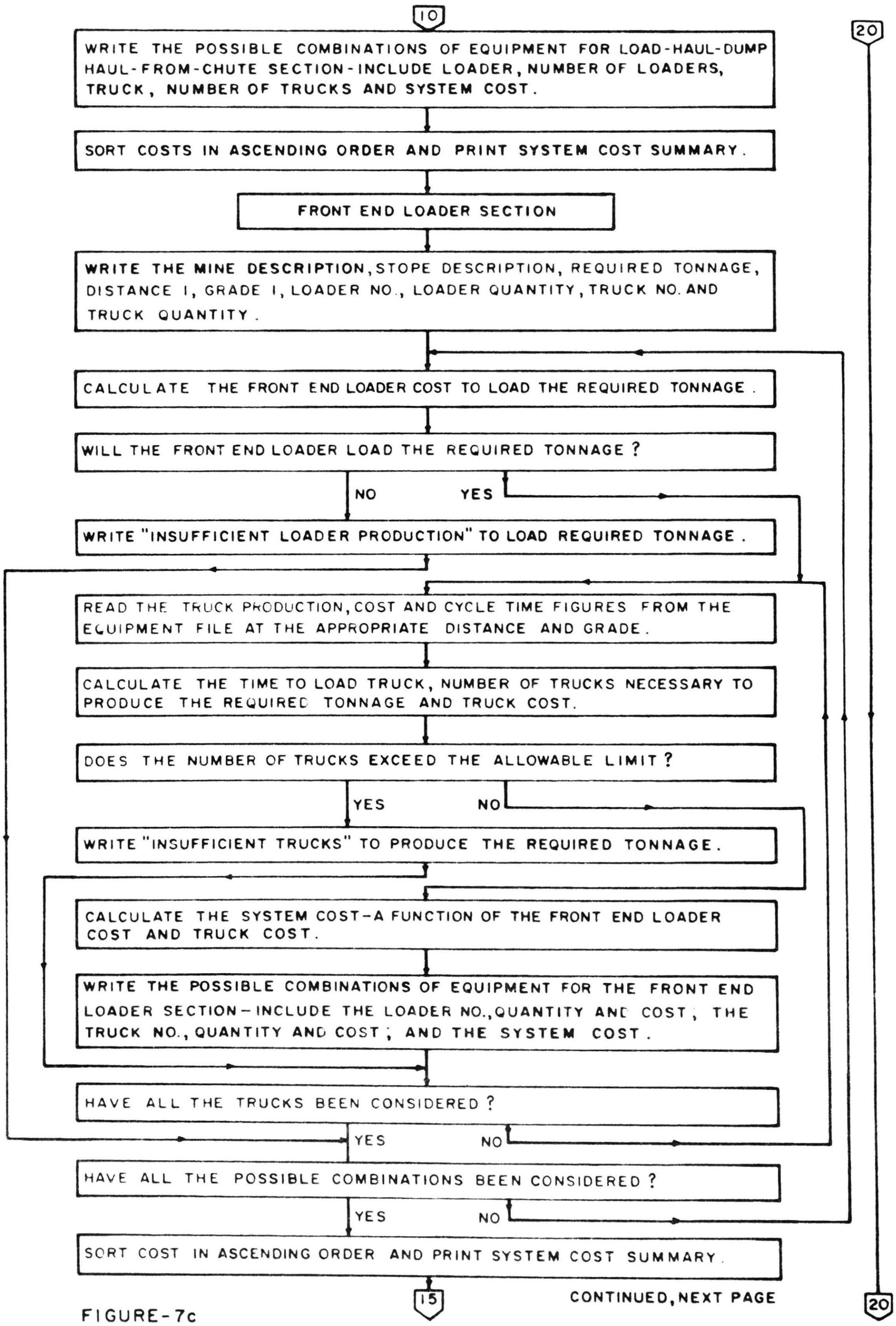

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FIGURE-7c

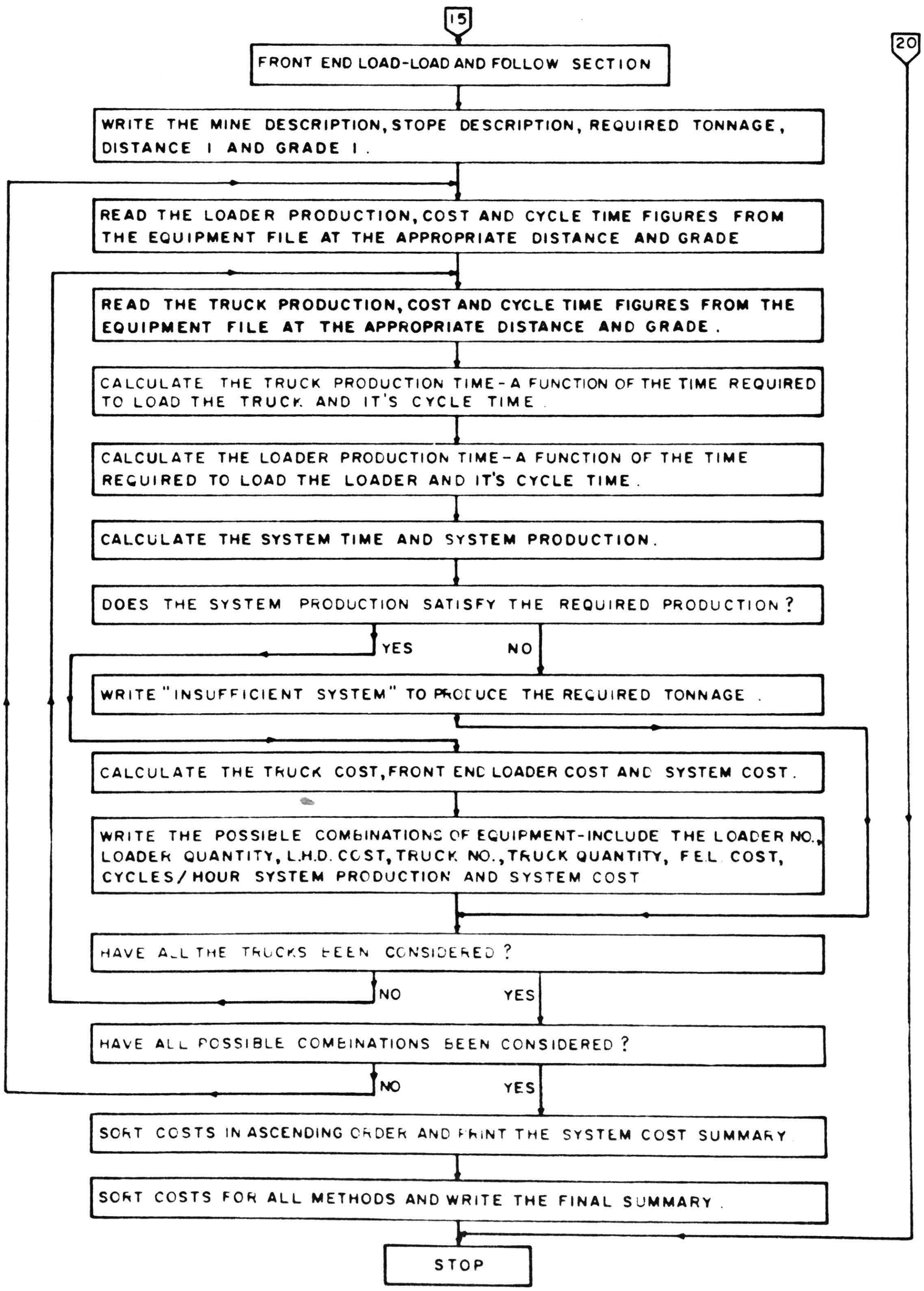

15
20
FRONT END LOAD-LOAD AND FOLLOW SECTION
WRITE THE MINE DESCRIPTION, STOPE DESCRIPTION, REQUIRED TONNAGE, DISTANCE I AND GRADE I.
READ THE LOADER PRODUCTION, COST AND CYCLE TIME FIGURES FROM THE EQUIPMENT FILE AT THE APPROPRIATE DISTANCE AND GRADE
READ THE TRUCK PRODUCTION, COST AND CYCLE TIME FIGURES FROM THE EQUIPMENT FILE AT THE APPROPRIATE DISTANCE AND GRADE.
CALCULATE THE TRUCK PRODUCTION TIME-A FUNCTION OF THE TIME REQUIRED TO LOAD THE TRUCK AND IT'S CYCLE TIME.
CALCULATE THE LOADER PRODUCTION TIME-A FUNCTION OF THE TIME REQUIRED TO LOAD THE LOADER AND IT'S CYCLE TIME.
CALCULATE THE SYSTEM TIME AND SYSTEM PRODUCTION.
DOES THE SYSTEM PRODUCTION SATISFY THE REQUIRED PRODUCTION?
YES
NO
WRITE "INSUFFICIENT SYSTEM" TO PRODUCE THE REQUIRED TONNAGE.
CALCULATE THE TRUCK COST, FRONT END LOADER COST AND SYSTEM COST.
WRITE THE POSSIBLE COMBINATIONS OF EQUIPMENT-INCLUDE THE LOADER NO., LOADER QUANTITY, L.H.D. COST, TRUCK NO., TRUCK QUANTITY, F.E.L. COST, CYCLES/HOUR SYSTEM PRODUCTION AND SYSTEM COST
HAVE ALL THE TRUCKS BEEN CONSIDERED?
NO
YES
HAVE ALL POSSIBLE COMBINATIONS BEEN CONSIDERED?
NO
YES
SORT COSTS IN ASCENDING ORDER AND PRINT THE SYSTEM COST SUMMARY
SORT COSTS FOR ALL METHODS AND WRITE THE FINAL SUMMARY.
STOP

FIGURE-7d

TABLE VI AN EXAMPLE OF THE ORE HAULING OPTIMIZATION, USING THE MINE PLANNING APPROACH. (THE COST FIGURES ARE FICTITIOUS, BUT ALL OTHER FIGURES ARE REAL.)

ORE HAULING OPTIMIZATION — MINE PLANNING — REQUIRED PRODUCTION 170.00 TONS/HOUR

METHOD 1	METHOD 2	METHOD 3
DISTANCE = 3000	DISTANCE = 1500	DISTANCE = 3000
GRADE = -1	GRADE = 0	GRADE = -1

NUM	COSTS	LOADER	QUAN	LPROD	LCOST	LTIME	TRUCK	QUAN	TPROD	TCOST	TTIME	METHOD
1	0.35	988	1.00	386.38	0.90	3.480	621	1.00	207.92	0.15	9.950	4
2	0.45	988	1.00	386.38	0.95	4.640	631	1.00	277.13	0.20	11.110	4
3	0.50	980	1.00	231.83	0.20	5.800	621	2.00	207.92	0.30	12.270	3
4	0.55	988	1.00	386.38	0.25	3.480	621	2.00	207.92	0.30	9.950	3
5	0.65	988	2.00	107.92	0.50	4.150	621	1.00	207.92	0.12	6.470	2
6	0.66	980	1.00	231.83	0.20	7.733	631	2.00	277.13	0.45	14.203	3
7	0.70	988	1.00	386.38	0.25	4.640	631	2.00	277.13	0.45	11.110	3
8	0.73	988	2.00	107.92	0.50	4.150	631	1.00	277.13	0.20	6.470	2
9	0.75	988	3.00	69.43	0.75	6.680	0	0.0	0.0	0.0	0.9	1
10	0.80	980	3.00	68.90	0.65	3.900	621	1.00	207.92	0.12	6.470	2
11	0.85	980	3.00	68.90	0.65	3.900	631	1.00	277.13	0.20	6.470	2
12	0.90	980	4.00	43.56	0.90	6.170	0	0.0	0.0	0.0	0.0	1

Mines and for the Fletcher No. 30 Mine, all of the present mining conditions were analyzed as well as those to be developed within the next three years. The various conditions of distance and grade were all listed and then summarized into 24 separate courses for each of the four methods. For each course the distance and grade for a chute system was included. Though sometimes it was obvious that a chute could not be installed, in which case the program ran the information for a "dummy chute." The program considered seven different loaders and three different trucks for all four systems of moving the ore. Since the program was being used to plan equipment and compare cost for different methods of loading and hauling, "situation planning" and "mine planning" programs were used. If the reader will recall, in "situation planning" the equipment needs are left at fractional parts (real number) and assumes that the equipment will be moved to a different location when it becomes idle. In "mine planning," the equipment needs are rounded up to the next whole number (integer number) and the program assumes that the equipment will sit idle when it runs out of the specified ore it is to produce. Management chose to put more confidence in the "mine planning" approach, and used it as the better basis for making equipment purchasing decisions. A sample of three of the 24 courses that were tested is found in Table VII. The cost figures have been altered in value, but relative to each other, they are correct. In effect, they

TABLE VII EXAMPLES OF COST ANALYSIS, GENERATED BY THE SIMULATION PROGRAM
(The numbers under Cost/Ton are relative indexes.)

ORIGIN OF ROCK (Stope, Area or Chute)	TONS/ HOUR REQ'D	DUMP POINT	HAUL DIST (Ft)	AVE GRADE %	COST/TON FOR EQUIPMENT CONSIDERED							
					LOADER						TRUCK	METHOD OF MOVING ORE
					A	B	C	D	E	F		
No. 17 Stope	85	Shaft	6000	-3	1.468	1.209	1.107	1.756	1.238	1.102	NONE	LHD
	85	Chute	2750	-4	0.929	0.810	0.758	0.846	0.808	0.754	621	LHD-HFC
		Shaft	3250	-1	0.948	0.829	0.777	0.865	0.827	0.773	631	
	85	Shaft	6000	-3	0.705	0.741	0.773	0.780	0.808	-	621	FEL
					0.789	0.804	0.832	0.841	0.863	-	631	
	85	Shaft	6000	-3	0.566	0.532	0.515	0.727	0.589	-	621	FEL-LAF
					0.599	0.540	0.518	0.720	0.583	-	631	
No. 29 Stope	85	Shaft	1500	-2	0.534	0.409	0.374	0.422	0.400	0.370	NONE	LHD
					-	-	-	-	-	-	621	DUMMY LHD-HFC INFO
					-	-	-	-	-	-	631	
	85	Shaft	1500	-2	0.595	0.631	0.664	0.670	0.698	-	621	FEL
					0.661	0.675	0.703	0.712	0.734	-	631	
	85	Shaft	1500	-2	0.334	0.257	0.239	0.262	0.247	-	621	FEL-LAF
					0.380	0.280	0.258	0.283	0.260	-	631	
No. 25 Stope	85	Shaft	4750	-3	1.209	0.991	0.906	1.387	1.009	0.903	NONE	LHD
	85	Chute	1500	-1	0.661	0.521	0.490	0.537	0.522	0.486	621	LHD-HFC
		Shaft	3000	-1	0.679	0.539	0.508	0.555	0.540	0.504	631	
	85	Shaft	4750	-3	0.675	0.711	0.744	0.750	0.779	-	621	FEL
					0.753	0.768	0.795	0.805	0.826	-	631	
	85	Shaft	4750	-3	0.502	0.458	0.440	0.599	0.496	-	621	FEL-LAF
					0.538	0.469	0.446	0.599	0.494	-	631	

appear as relative indexes. The largest loader and the largest truck were left out of the examples since they were, in fact, too large to be taken underground and were not even considered. Loader "F" was so configured that it was excellent for LHD, but management did not believe that it was practical for loading trucks. Therefore, it was not considered for methods "3" or "4".

The analysis of these programs was somewhat startling to all those people involved. While it was recognized that there would be differences in the cost between the various systems of moving the ore, the magnitude of those differences was not appreciated until the analysis was complete. It was extremely significant that for nearly all of the conditions specified, the FEL-LAF system was the cheapest method for moving the ore. Sometimes it was as much as one-half that of the LHD-HFC and as much as one-third that of LHD. Only when the cycle time is extremely short or extremely long, do the other systems look more favorable. For the LDH-HFC to be the most favorable, the cycle time of the LHD must be very short. At the same time, the differences in the equipment production cost, used within a given system, were less significant than the differences with the system. This caused management to reappraise other factors of equipment purchasing (i.e., present inventory, operating and maintenance knowledge, etc.). Furthermore, they have since begun to re-evaluate the total concept of mine development, taking into consideration the idea of

FEL-LAF. The original development plan for two mines has been changed as a result of this study, while the original plans of a third mine have been conferred as being the best approach.

F. Development of the Optimum Trackless Method, Materials Moving Charts

At this point, it became obvious that when the individual was faced with the problem of equipment selection and application for many working areas, the computer programs could generate such a tremendous amount of information that it would be virtually impossible to comprehend it completely. To present the information, a program was written to generate production charts of the least cost equipment and method for each condition. The charts cover distances from 250 to 10,500 feet and grades from -5 to +10%. Production requirements were chosen to be 85 and 170 tons per hour (representing the normal production from one and two jumbos respectively). Ore body dimensions were set at 500 feet (narrow) and 1,500 feet (wide). Figures 8a-9e show example production charts for Caterpillar 980 and 988 front-end loaders, and Eject-All E621 and E631 trucks calculated by the mine planning program for wide conditions at the low production rate. The first plot shows the least cost combination of equipment for all conditions. The next four plots show the least cost method for using each particular combination of equipment for all conditions.

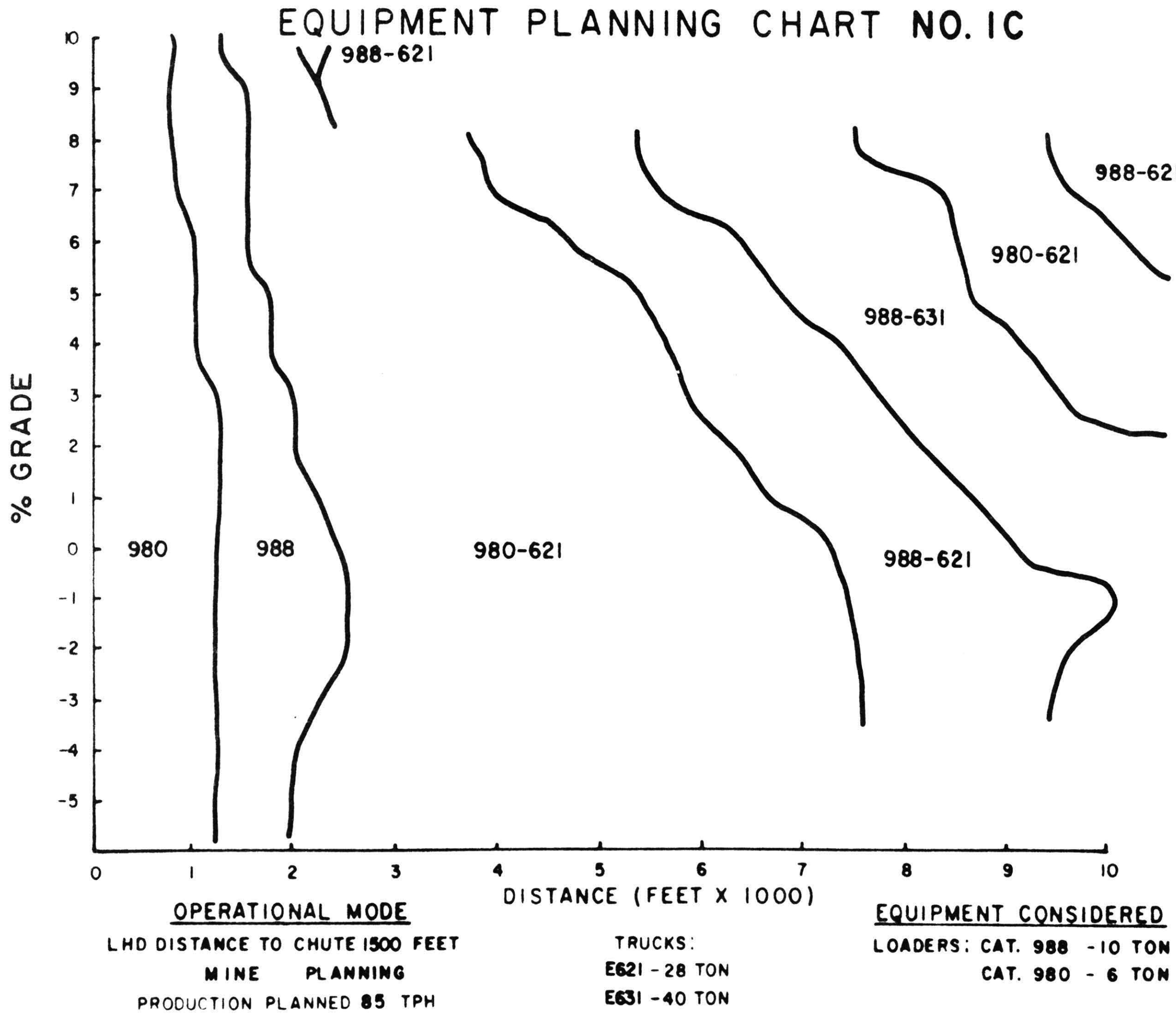
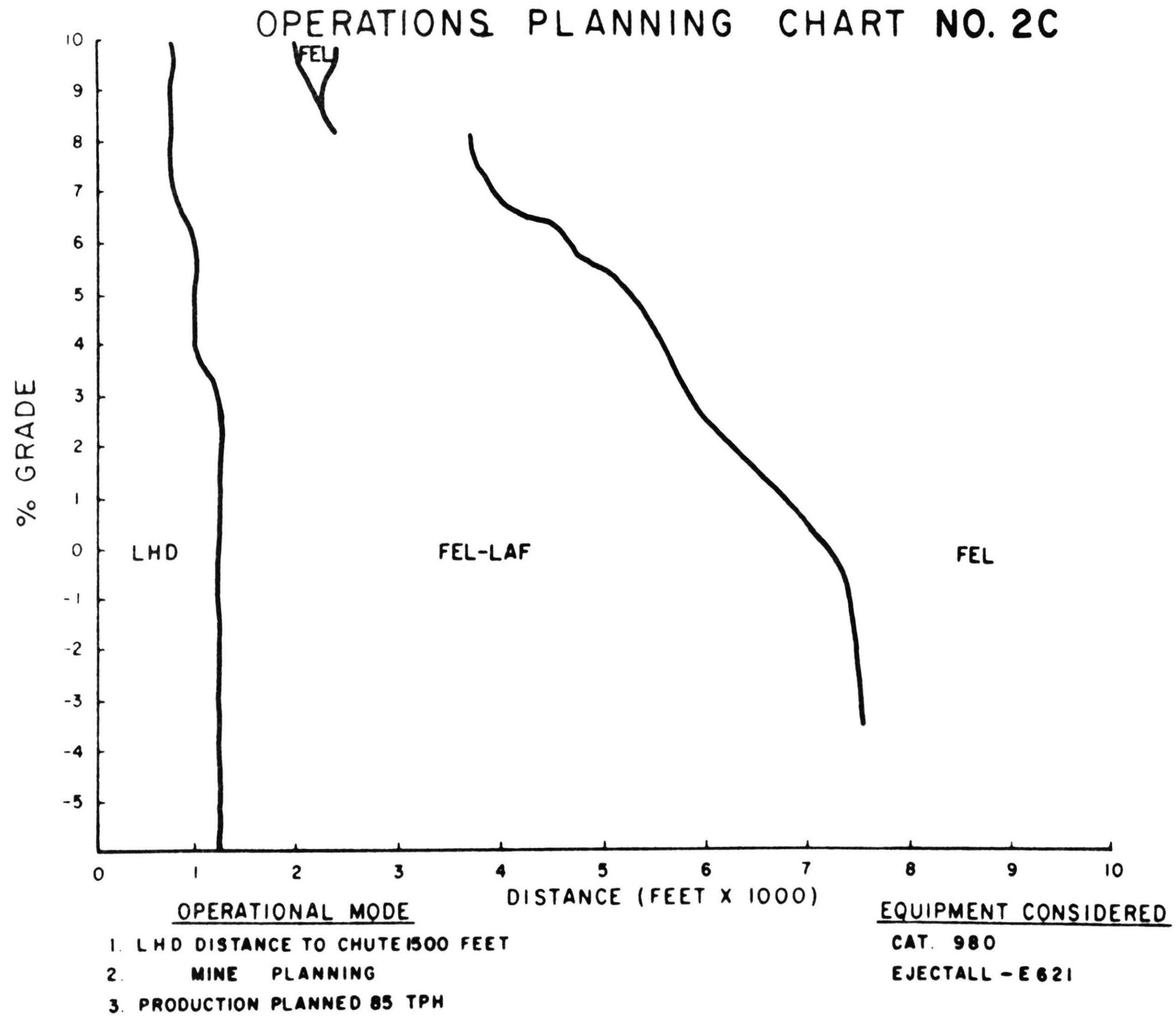


Figure 8a Equipment and Operations Planning Charts
(Continued)

Figure 8b Equipment and Operations Planning Charts (Continued)



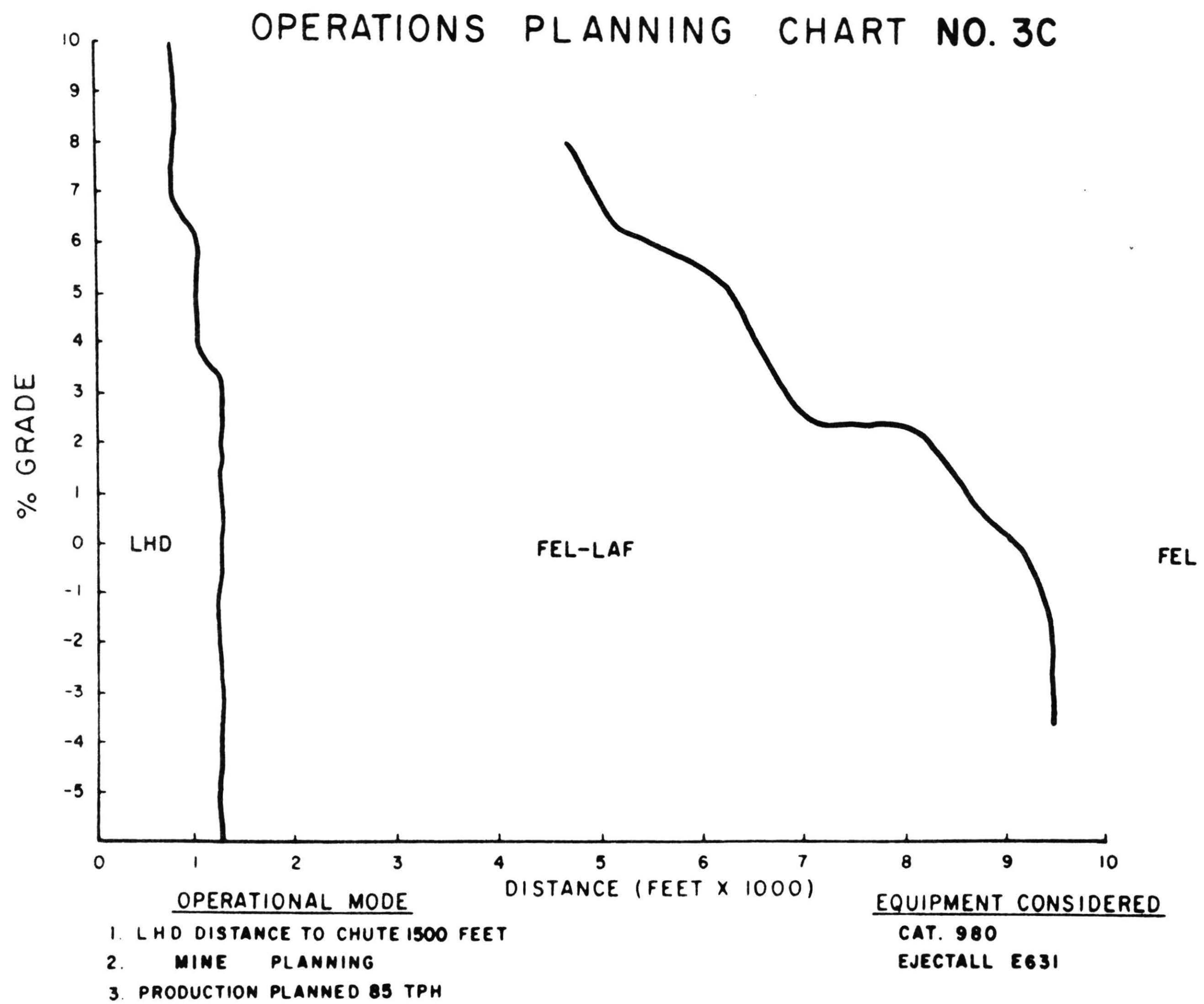


Figure 8c Equipment and Operations Planning Charts (Continued)

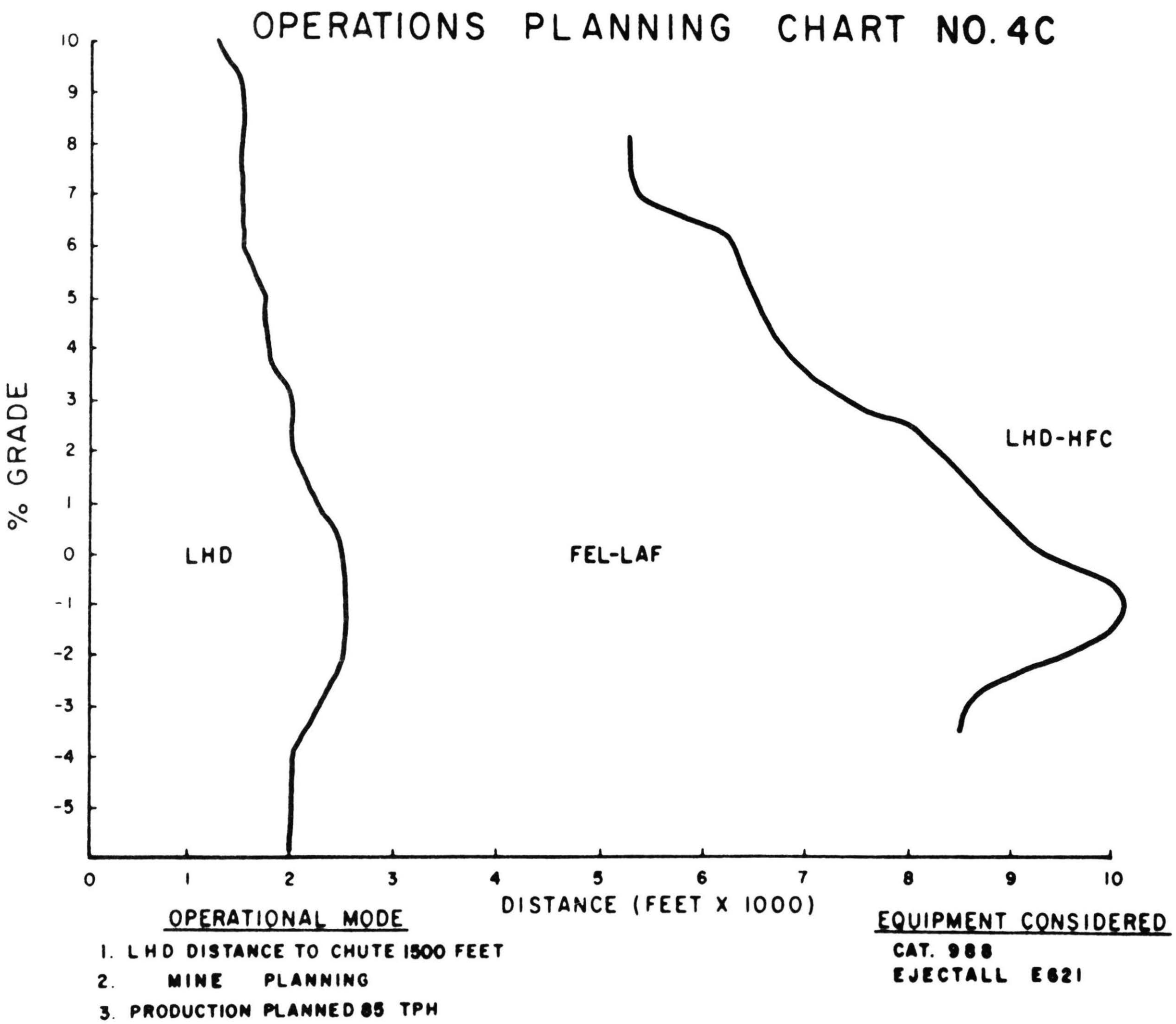


Figure 8d Equipment and Operations Planning Charts (Continued)

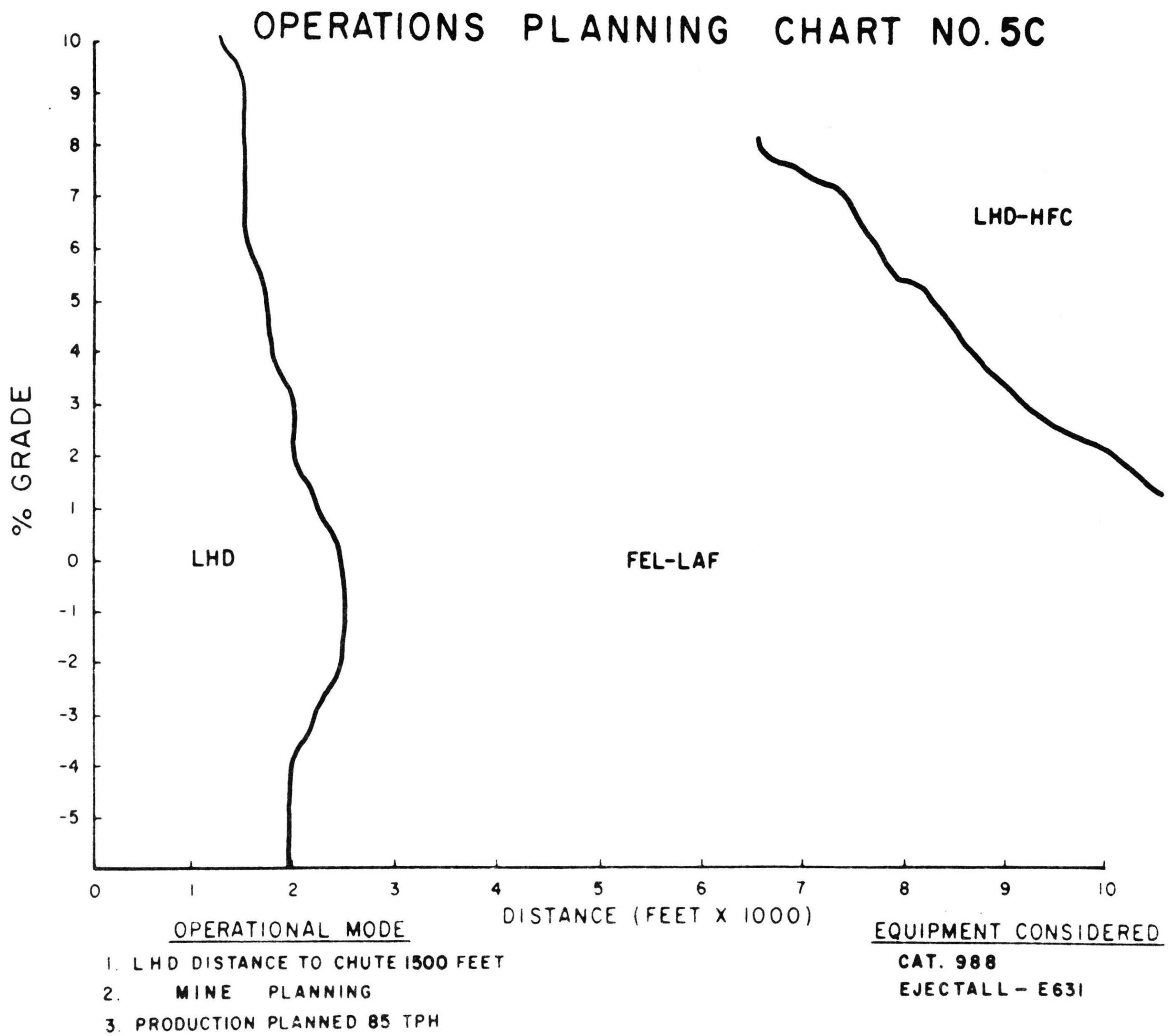


Figure 8e Equipment and Operations Planning Charts (Continued)

In the original group, there are 40 charts,[21] of which 5 are shown here as an example. Using the charts is a very simple procedure. First, find the mine condition that exists, then read the optimum equipment combination from the chart. Then by looking at the chart in that set which contains that equipment, the optimum method can be determined. Even though no cost figures can be obtained from the charts, the user would know that it would be the least cost method.

G. Optimum Chute Distance Problem

A separate problem was brought to this research project by the management of St. Joe. The object was to solve the problem of knowing how often the vibrating feeder chutes should be moved if the ore body is very long and narrow. For those who might not be as familiar with this system of mining, a chute is established along the strike length of the ore trend. LHD units haul from all directions into the chute. The distance hauled across the width of the ore body is limited by the ore body width, but the distance along the strike should be governed by economics. Large trucks on a lower level pull the ore from the chutes and haul it in one direction, towards the shaft. One can realize that probably one-half of the rock will be hauled at least partially in the wrong direction, compared to the direction of the final haulage. Therefore, the more often the chute is moved, the less wasted haulage, (which amounts to operating cost) but the greater will be the fixed cost (which in this case is ownership cost).

1. A study of cost of handling ore by the LHD-HFC method was done on the basis of the following assumptions:
   a. Mine development in two principal directions along the strike of a very long ore body.
   b. 5000 tons per day required output from the mine; or 2500 tons per shift; or 1250 tons per shift in each direction.
   c. Assuming 200 tons per day will come from development, 200 tons per hour, per chute will be needed from production stopes for 6 hours per shift.
   d. To yield this production, one must assume approximately 500 tons per jumbo shift, an ore body 400 feet wide and 80 percent extraction.
   e. Vibrating feeder and structural steel can be moved as needed and will have a serviceable life of 18 years.
2. Chute costs were estimated as follows:
   a. Time period in hours is

6 Hrs/Shift x 2 Shifts x 255/Year x 18 Years = 55080 Hours.

   b. Center line spacing; move interval

      1 Year = 500 feet,

      2 Years = 1000 feet,

      3 Years = 1500 feet.

c. Purchase and Installation

(All Costs Hypotehtical)	Move Interval (Years)		
	1	2	3
Steel for Structure	$ 4,500	$ 4,500	$ 4,500
Vibrating Feeder	13,500	13,500	13,500
Installation Cost @$5000/Move	90,000	45,000	30,000
Development Cost @$7000/Move	126,000	63,000	42,000
Repair Cost	10,000	10,000	10,000
TOTAL	$244,000	$136,000	$100,000
Cost/Hour	$4.43	$2.47	$1.82
Interest	.20	.11	.08
Total Cost/Hour	$4.63	$2.58	$1.90

d. Equipment Used in Study-Caterpillar 980, Eject-all E621

This information was used in the situation planning program to project the annual costs of moving ore, for each move interval, over a 12-year period. The results are shown in Figure 9. Interestingly, the cost trends for all three move intervals are virtually identical. On this basis, the problem becomes one of convenience and the three-year move interval becomes the most desirable. The sawtooth effect on the graph is the result of fluctuating LHD costs in moving the ore to the chute.

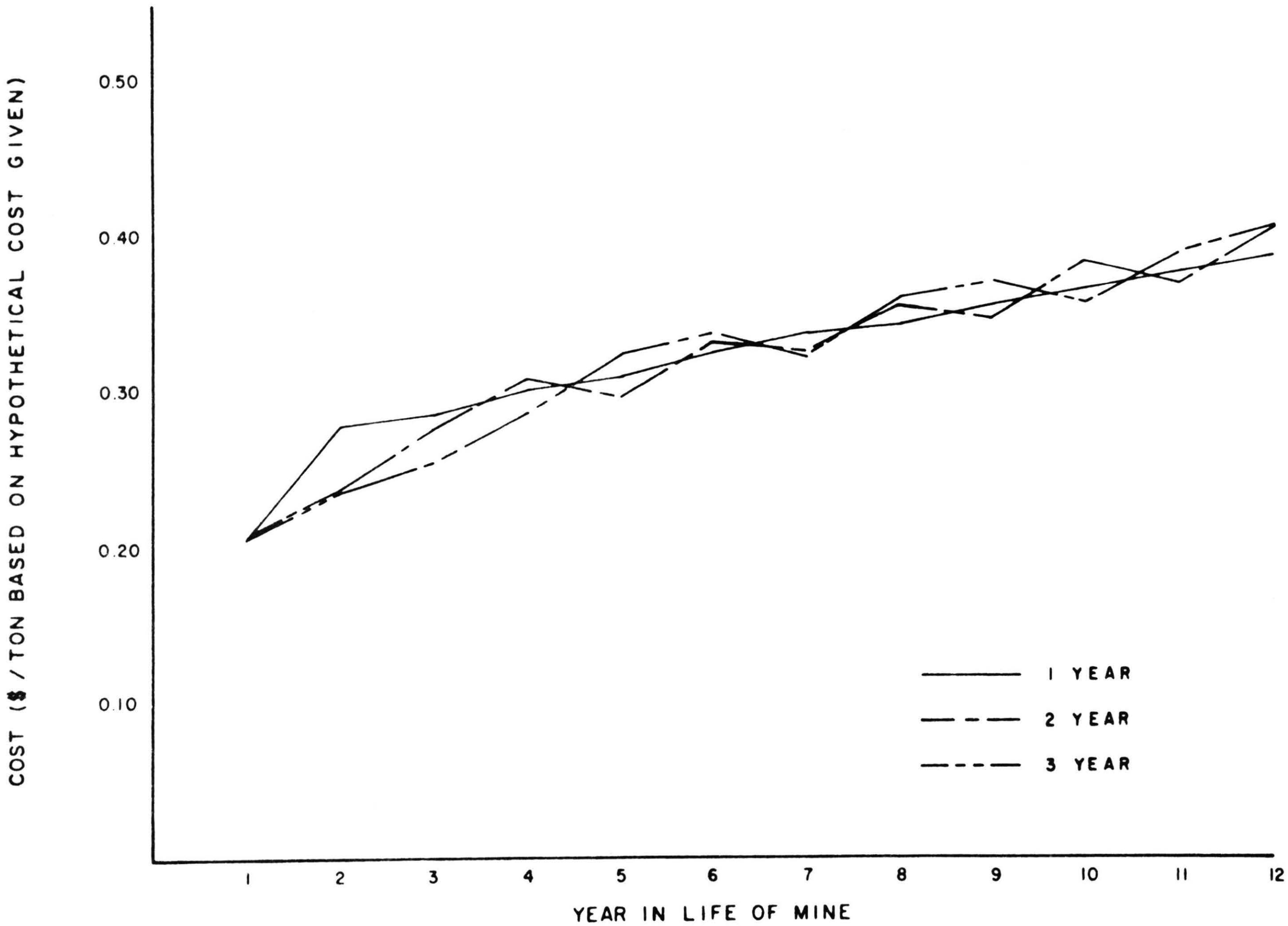


FIGURE-9 CHUTE SPACING OPTIMIZATION, USING SITUATION PLANNING

Periods longer than three years were not considered in this analysis but would undoubtedly yield a different answer, particularly if the distance along the strike becomes very much greater than the haul distance across the width of the ore body.

H. Validation of "Load and Follow" and "Front End Load"

At this point in the study, new loaders had been acquired for the Viburnum mines and they were beginning to use the new loaders in conjunction with the truck they already had. Also, the Fletcher Mine had acquired an E631 (40 ton) truck, which would eventually be used for the HFC method. The St. Joe Management suggested that a study be conducted to see if what had been predicted by the simulation, was being achieved for both LAF and FEL. It is worth mentioning that this new concept of "Load and Follow" was not met with favor by some of the operators of the loaders or the line supervisors. Another problem was that there was not an incentive system that had been established for this method of moving ore. It takes time to establish a basis for a bonus schedule and it is unlikely that produc-ion expectationsor performance records are ever achieved during the study period. However, the study was made and the results are shown in Table VIII. As in previous validations, the individual predictions are not extremely

TABLE VIII VALIDATION OF SIMULATION BY OBSERVATIONS

Stope Number	Tons Ore Loaded and Hauled	Method Used	TPH Rate Observed	TPH Rate Simulated	Simulated Tons of Ore Loaded and Hauled	% Error For Simulation	Course Description Ave R_G	Course Description Distance (Ft)
63W18	1981	LAF	220.1	226.1	2035	+2.7	-2	2160
63W17	1661	LAF	184.6	211.8	1906	+14.8	-2	2500
65C21	1311**	LAF	145.7	215.8	(1942)* 1295**	(+48.1)* -1.2**	-2	2410
67C42	1479	LAF	164.3	162.3	1461	-1.2	+8	2600
67V28	1160	LAF	128.9	134.5	1210	+4.3	-1	4750
67G69	1916	LAF	212.9	186.3	1677	12.4	0	2500
Total LAF	(9508)* 8197**	LAF	182.2**	184.2**	8289**	(+7.6)* +1.1**	-	-
64V28	1008	FEL	112.0	109.8	988	-2.0	-1	4750
66V25	1505	FEL	167.2	140.3	1263	-16.1	-2	3000
Total FEL	2513	FEL	139.6	121.1	2251	-10.4	-	-
GRAND TOTAL	(12021)* 10710**	-	167.0**	168.2**	(12482)* 10540**	(+3.8)* -1.6**	-	-

* The rock piles for this stope loading were extremely small. Therefore, there is an adjustment that is made which is explained in the Interpretation of Results, LAF and FEL Validations.

** The adjusted figures.

accurate. Yet, considering the circumstances under which they were taken, the results are actually better than were expected. Of course, when the total rock of all of the stopes is considered for the method observed then the tonnage predicted is fairly accurate. But again, compensating errors account for the unwarranted accuracy. Further analysis of this and other data, will be reserved for results and conclusions.

I. Analysis and Simulation for the Goose Creek Mine Haulage System

The Goose Creek Mine is one of St. Joe's smallest mines in daily output and was not developed with the same concepts of trackless haulage as were most of the other mines in the New Lead Belt. As a result, the roadways are more narrow and crooked, the dumping stations are in smaller rooms, and most important, the broken rock piles are usually much smaller than in the other mines. At the time of the study, the quota for production for this mine was only 1400 tons per day or 117 tons per hour for two six hour shifts. The equipment used to move the ore by LHD were two Caterpillar 980 loaders. St. Joe management requested that this situation be studied to see if the analysis would reveal any possible way to improve on the existing practices. The present practices were averaging 60 TPH per loader, from various stopes throughout the mine, but loading with two of them at once in separate stopes. All of the data on the courses for each stope was obtained and the haulage was simulated for methods 1, 2, 3 and 4, using mine planning for 60 TPH. The simulated production cost revealed that the least cost method was LHD and it came within 1.4% of the year to date cost for the loading and hauling of that mine. However, when the entire mine production of 120 TPH was used in the mine planning approach, the least cost method became LAF and showed a potential cost savings of 22%.

Using LAF with a Caterpillar 980 loader and an Eject-all E621 truck, the system should be sufficient to clean up all of the stopes during each shift. Even though the margin of savings predicted was not very spectacular, on the basis of the information presented, the management put an E621 truck into the mine. Several physical changes must take place in the mine before it will accommodate the system to yield the predicted tonnage. When these changes are made, a validation of the predicted tonnages should be made.

J. Repeat on Fletcher LHD Cost Validation

One of the first tasks of this materials moving study was that of trying to validate the predicted cost, with the actual cost very early in the study. It was so successful that no adjustment of any of the variables was made at that time. Near the end of this reported research, and before the practice of LHD at Fletcher Mine was replaced with other methods, a second analysis was made to determine if the predicted cost corresponded to the actual cost of the LHD method. However, the technique was changed somewhat in this validation. The tons for each stope were gathered from company records for the month. Then the simulated production rate was divided into the stope tonnage for each stope, to yield the calculated hours. The calculated hours were then used to weight the simulated cost (cost index for this report). The weighted costs were then totaled to reveal what the mine's loading and hauling cost would be. In this report (Table IX) the index of that cost is 0.7328 per ton. An index of actual months cost was 0.7245. The predicted cost was only 1.15% in error. This confirmed the first cost validation. It also illustrates that after eight months of production, the simulation of the many variables was reasonably stable, or the compensating errors were varing in the same proportions.

K. Other Validations

1. There was an attempt to validate the LHD-HFC installation at the Fletcher Mine. The installation had just been completed when the study was made. The study for the simulation of the chute loading facilities was originally done on the Viburnum No. 28 Mine chute that had been operating very efficiently for years. This new chute at Fletcher had many problems that made the loading of trucks considerably different than what had been simulated. For example, without going into the details of the problems that existed, the truck loading time

TABLE IX SECOND LHD COST VALIDATION

STOPE NO.	TONS ORE REPORTED MOVED (LHD)	SIMULATED PRODUCTION RATE (TPH)	CALCULATED HOURS FROM RATE SIMULATED	WEIGHTING FACTOR	INDEX OF SIMULATED COST*	WEIGHTED AVERAGE INDEXED COST/TON
63W61	15727	95.32	165	.1521	.6542	.0995
63W14	3876	94.54	41	.0378	.5616	.0212
60W192	6968	81.98	85	.0783	.5662	.0443
63W18	10946	86.19	127	.1171	.6695	.0784
65C21	9318	76.15	124	.1122	.7449	.0836
68C17	11411	83.29	137	.1263	1.0829	.1368
67C42	10360	75.07	138	.1272	1.0920	.1389
63W114	9079	82.54	110	.1014	.5395	.0547
60126	14677	91.73	160	.1475	.5109	.0754
TOTAL	92362		1085	1.0000		.7328

*This is not the actual simulated cost. It is an index of the cost, since actual cost are proprietory information.

was 2.54 minutes, compared to that of the simulation (and the No. 28 time study), of 1.17 minutes. Another problem, the truck was only being loaded approximately 2/3 of its capacity. Thus, instead of a 40 ton load being hauled, less than 30 tons were being hauled. Also, the chute would often run out of rock for short periods of time, thus causing needless and unsimulated delays. However, the time study, and the HFC simulation did assist management in locating where the problems existed that needed to be corrected. In spite of all of the problems, if one used a loading time of 1.17 (23 TPH for 27 tons) then the simulated production for a 27 ton pay load results in 238.8 tons per hour. As a matter of fact, their average cycle time for the time study was 6.77 minutes which calculates to 239.3 tons per hour if the full 60 minutes is used instead of the 44.82 as used in the simulation. From observation, the men were working at the job full time, but the physical restrictions simply would not allow the expected productivity. Since this study was made, most of the problems have been corrected and the reported productivity has increased. The LHD-HFC production from this chute should run at least 320 tons per hour as predicted.

2. After the previous time study revealed several problems which were directed back to the management to be corrected, a study was done by one of the mine engineers at Viburnum on the No. 28 Mine chute. They have installed recording charts which keep a record of the cycle of each vehicle that dumps ore into the pocket. The cycle time actually recorded throughout several shifts was 10.34 minutes for the truck pulling ore from the chute. Applying the .83 and .90 correction factor to this, it becomes 7.72 minutes. The simulated cycle time is only 6.47 minutes, and the simulated tonnage would have been 158.2 TPH. The actual tonnage that was reported by the engineer was 146.9 TPH. Even though the reported cycle time was in error by 16.1%, the reported tonnage was only 7.7% off.

IV Interpretation of Results

When one tries to use mathematical functions to model physical situations, involving prediction of human behavior and their interaction with mechanical equipment, there will undoubtedly be errors. If the model is structured correctly, the errors will not be those of logic. If such internal errors exist in the model, during validation, these errors will produce illogical results and should be obvious. But there can also be errors caused by assigning improper values to the data that is to be used in the program to generate the results. These errors are typical of simulation studies involving men and machines. The approach that is used by many researchers is that of assigning the probability distribution of each variable to a random number generator. In this manner, combining all of the variables randomly generated, dozens or even hundreds of times, produces output that has an expected confidence level. However, such accuracy must be consistent with the objective of the simulation. In the simulation work involved in this study, the objective stated earlier was that "of accurately determining the optimum [trackless] method of materials moving for the four" systems studied. The "accuracy" of this study must be good enough to meet this objective. There are many applications to which the simulation programs and the techniques used here will probably require a greater degree of accuracy than now exists. But because of the degree of familiarity that this researcher felt for the systems being simulated, and in an effort to keep this very complex system as simple as possible, the deterministic approach, rather than stochastic approach was applied. This researcher has helped develop other industrial computer programs where the Monte Carlo method of generating random numbers is used. But invariably, where operating people can either ignore or bypass the probability aspect of a variable, they will do so and end up with an explicit number for that variable. Therefore, it was felt that a reliable time study of the "fixed time" resulting in an explicit value was the best approach that could be used and understood by operating supervisors.

A. Summary of the Simulation Validation

1. The Original LHD Simulation

The original "Travel Time and Earth Moving Production Program" furnished by Caterpillar, when modified slightly, simulated the underground LHD application of the 988 loaders very closely. Time studies of that portion of the cycle not simulated, revealed that the loading time plus all of the delays at that end of the cycle equaled 1.16 minutes, but had a standard deviation of ± 1.09. The dump time portion of the fixed time, considering all of the delays on that end of the cycle amounted to 0.36 ± 0.19 minutes. In spite of the wide deviation of the loading times using a total fixed time of 1.52 minutes, an 83 percent equipment availability and a 90 percent man efficiency, resulted in a simulation production of 38,179 thousand tons, which was only +1.3 percent in error (see Table X). However, errors on individual courses varied from a +18.9 percent to a -10.1 percent. This first study involved 428 hours of LHD operation. The predictions of the cost of moving ore by LHD with 988 loaders resulted in comparable accuracy, and amounted to slightly less than 2¢ per ton error.

2. The LAF and FEL Validations

These studies were taken in two different mines and involved 988 loaders and both E621 and E631 trucks. As was expected, there was considerable operator resistance in some cases, and where they had the support of the line supervisor,the productivity was most certainly not up to the expected or simulated tonnage. This was the situation characterizing 64V28 stope on the LAF study. Yet in spite of this, the predicted production was only off +8.8%. Still worse conditions prevailed in 68C21. To get to the 68C17 stope with the new truck, the truck had to leave the roadway, and travel a considerable distance through the "old mines," including several hundred feet of an abandoned sump where the truck maneuvered axle deep in mud. This was

TABLE X SUMMARY OF SIMULATION VALIDATIONS

Method Validated	Range of Distance (Feet)	Range of Grade (Percent)	Total Tons Moved	Total Hours Simulated	Composite Error of Production by that Method	Range of Course Errors (Percent)
1st LHD	1500 to 2500	-1.9 to 3.9	37,680	428	+1.3	-10.1 to 18.9
LAF	2160 to 4750	-1.0 to 8.0	8,197	54	+1.1	-12.4 to 14.8
FEL	3000 to 4750	-2.0 to -1.0	2,513	18	-10.4	-16.1 to -2.0
2nd LHD	1800 to 3000	-2.0 to -3.8	95,362	1085	+1.2	---
HFC	3000	-1	3,526	36	+7.7	---

enough to account for the +14.8% error of the simulation prediction. The situation of 65C21 stope was a still different problem. The individual headings usually contained only 150 to 200 tons of rock. In the original time study work for the 988, which established the "fixed time," the time of 1.16 minutes included both good and bad loading conditions. But it did not include the time required if all of the rock was loaded from small rock piles. The loader will normally have to use considerable amounts of time giving a stope heading the final "cleaning." It must be clean enough that the drillers can drill the next round of holes with only minor amounts of hand shoveling. But if instead of 600 tons being in each heading, there is only 150 to 200 tons, then the operator must spend three times as long just scrapping up the heading. From the original time studies it was determined that a loader operator will load the first 80% of a 600 ton rock pile in approximately 50% of the total time it takes to load all of the rock and clean the heading. If this information is applied to the problem of small rock piles in each heading, then the "load time" to clean 600 tons in four headings is 2 hours and 40 minutes, not the 1 hour and 10 minutes expected in the simulation.

Since there was 1311 tons moved from 65C21, and the suggested error was as much as 1.5 hours per 600 tons, then the total time lost was at least three hours. If the simulated time is reduced by three hours, then the total tons produced would have been 1295, which is only in error by -1.2% of what actually was produced. For this reason, the data from 65C21 should not be considered, except to illustrate how costly it is to both the company and the loader operator to tolerate small rock piles scattered in several headings. In fact, the problem was corrected after the study was made. If the 65C21 data is not included, then the error of the simulated tonnage is only +1.1% for the LAF method.

The simulated production for the FEL study is somewhat lower than that which was actually produced. The operator resistance found in the LAF method vanished when the FEL method was applied. The operators were "proving the point" by getting the maximum production possible for this method. Thus the composite error for the simulated FEL method was a -10.4%, the highest of all the validation composite errors for any method (see Table X).

3. The Second LHD Cost Validation

Little can be said for this study except that the results illustrate that for the second time, the LHD cost was predicted with just over 1% error. Since the approach taken can not justify such accuracy, again compensating errors must be given the credit.

4. The HFC Validations

While there were so many physical problems which kept the newly installed Fletcher Mine chute from operating efficiently for the HFC method, it is interesting that when the correct loading time was applied and the tonnage calculated by the simulation formulas and based on the rest of the cycle time, then the predicted tons per hour matched that which they were achieving. It really only validates the dumping and hauling portion of the cycle.

5. Other Validations

The HFC study at the Viburnum No. 28 Mine, which was done by a mine engineer assigned to that mine, used recording charts rather than standard time study techniques. Thus it is not possible to determine just where the errors in cycle time were taking place. But the fact that the predicted tonnage was only off by +7.7% illustrates that overall, the considerations of the .83 and the .90 factors must be very close to the conditions that actually existed at the mine during the study.

6. A Comparison of Methods

   In the validation work that was done, only in one stope was the comparison actually made between the more conventional (and well excepted) FEL system and the new concept of LAF. But in this one comparison, even with the operators trying to bias the study, the LAF method out-produced the FEL system by 15% (i.e. 112.0 to 128.9 TPH). The simulated difference was 22% (i.e. 109.8 to 134.5 TPH).

B. Analysis of Operating Conditions Leading to Optimum Equipment Selection

The objective of this study was to be able to determine the optimum method of moving the ore in the trackless mines. The unexpected results revealed by the study and the wide range of costs predicted were a surprise to this researcher and to the St. Joe management. It clearly indicates that when personal experience and judgement are the only criteria for equipment and method selection, then less than optimum results are apt to be achieved.

The 24 individual cases, where the hauling conditions were described for the three mines and all of the available equipment was tested, resulted in computer "print-outs" similar to Table VI, but listing all six loaders with the two trucks using all four methods. The method of explicit enumeration revealed the optimum method - equipment combination for each case. To aid management further by condensing the data, tables were prepared similar to Table VII which showed the operating cost per ton for each method - equipment combination, with the optimum system in each case underlined. Looking at all of the tables at one time, it was very easy to see the least cost brands of equipment and methods of moving the ore. As mentioned earlier, there were far greater differences in the costs between methods than there were between brands of equipment using the same method. All 24 situations are summarized in Table XI. There are several points which need comment concerning the results of this work.

1. The least difference between the "Best" and the "Worst" combination was 36¢ per ton. The greatest difference was $1.24 per ton. It is true that in this worst combination, that the LHD method is obviously not practical at 6000 feet. Yet for a short term, development situation, one might be tempted to try it and not realize how much it would cost.
2. Since LAF was the "Best" method, 18 out of the 24 situation studied, one would conclude that for the period of time involved most of the tonnage will be moved by this method if it is to be done with the least cost.
3. The LHD method never appears as the least cost method, even at distances as low as 1500 feet.

TABLE XI OPTIMUM METHOD AND EQUIPMENT STUDY FOR FUTURE MINE HAULAGE FOR THREE OF ST. JOE MINERAL CORPORATION'S MINES

FROM STOPE TO SHAFT		OPTIMUM METHOD AND EQUIPMENT						COST DIFFERENCE
DISTANCE (Feet)	GRADE (%) (Approx)	BEST COMBINATION			WORST COMBINATION			BETWEEN BEST AND WORST $/TON
		METHOD	LOADER BRAND	TRUCK (SIZE)	METHOD	LOADER	TRUCK (SIZE)	
6000	-3	LAF	C	30	LHD	D	-	1.24
1500	-2	LAF	C	30	FEL	E	40	0.50
4750	-3	LAF	C	30	LHD	D	-	0.95
1750	-4	LAF	C	30	FEL	E	30	0.44
6000	-2	LAF	C	30	LHD	D	-	1.05
5000	-2	FEL	B	30	LHD	D	-	0.81
3500	-2	LAF	B	40	LHD	A	-	0.80
2250	-1	LAF	C	30	FEL	E	40	0.48
3750	-2	LAF	C	30	FEL	E	40	0.41
4500	-1	LAF	C	30	LHD	A	-	0.76
4000	-2	LAF	C	30	LHD	A	-	0.68
4250	-2	LAF	C	40	LHD	D	-	0.68
6000	-1	LAF	C	30	LHD	A	-	1.00
9000	-1	FEL	B	30	LHD	F	-	0.83
4750	-2	LAF	C	40	LHD	D	-	0.78
6000	-2	FEL	B	30	LHD	E	-	0.69
3000	-4	LAF	C	30	LHD	A	-	0.50
3250	-2	LAF	C	30	LHD	A	-	0.56
2000*	+3	LAF	C	30	LHD	A	-	0.36
2750	+3	LAF	C	30	LHD	A	-	0.47
3000*	+2	HFC	C	30	LHD	A	-	0.51
3750	+2	LAF	C	40	LHD	A	-	0.66
4000*	+1	HFC	C	30	LHD	A	-	0.73
4750	+1	FEL	B	30	LHD	B	-	0.50

*Only in these three cases was the LHD distance to the chute specified at 500 feet, signifying a long narrow ore body. All other distance to the chute was 1250 feet or greater

4. For the HFC method to become the "Best" method, a haul to the chute of only 500 feet was used.
5. The FEL became the optimum method in four cases, at distances between 4750 and 9000 feet. Yet, LAF was the optimum method for six cases in the range from 4500 to 6000 feet. The difference was the required tonnage. If one expected to move only 85 TPH from such distances, then LAF could do it best. If, however, one required 170 TPH to be moved from that distance the FEL system usually was optimal.
6. Concerning the loaders that appeared to give the least cost for these situations, Brand C was usually the best, except when FEL was optimal, then Brand B was the best. Brand A, D or E usually showed up the worst for the conditions simulated.
7. In the case of the 30 to 40 ton truck selection, one might suspect that there is an internal error of logic in the basic program to cause the 30 ton truck to usually show the best cost. But in this case, there was such a drastic difference at that time, between the first cost of the 30 ton and the first cost of the 40 ton trucks, that it made the difference that is observed. The cost per ton for the live load

was actually greater in the 40 ton truck. The manufacturer has corrected this situation since then by raising the price of the 30 ton up equivalent to the 40 ton. Therefore, the same results would not be repeated now.

To further aid management in making the optimum selection of equipment and method, 40 charts were made up which cover all conditions that will be encountered in the next several years of trackless mining in the New Lead Belt mines. If they are used, they should be of considerable aid.

Concerning the Optimum Chute Spacing problem, it is a matter of trading slightly greater LHD cost for somewhat higher cost of moving the chute. The results were surprising to everyone concerned and yet perfectly logical. The cost trend for the 1,2, or 3 year move interval is nearly identical. So for operator convenience the three year moves are recommended.

V. Conclusions

It was the intent of this research effort to identify the areas of possible optimization of both equipment and materials handling methods. The simulation study illustrates the versatility, the flexibility, speed and tremendous power to today's mining equipment that can be used in many possible ways to efficiently move ore in a modern trackless mine.

In this research, six methods of moving ore in a trackless underground mine were identified. For four of these methods, formulas were developed which will yield the production and cost per ton when used in conjunction with the Simulated Cost and Production Tables generated in this research. In addition to the four methods of moving the ore, three operating points of view, or approaches, were developed to aid the manager in the decision making process. For this particular study, seven loaders made by four different manufactuers were used in the simulation study along with three different size trucks made by one manufacturer. In addition, a vibrating feeder type of chute was also simulated since it is an intergal part of the materials handling system in some of the St. Joe mines.

There were five official validations of the simulation for the four methods, each having a composite error as follows: LHD: +1.3%, +1.2%; HFC: +7.7%; FEL: -10.4%; LAF: +1.1%.

The results of this effort yielded optimum methods of moving the ore for all of the conditions studied. It also shows that <u>any</u> condition which alters the cycle time appreciably, may effect the optimum solution to the point that it may no longer appear optimal. Most investigations of this type usually result in identifying the optimal system as a function of distance. This is a questionable practice, unless all of the other conditions, especially production requirements and haulage grade are held constant. Nevertheless, a few qualified general conclusions will be made:

1. In most of the conditions now being mined or that will be mined in the period studied, in the New Lead Belt of Missouri, the "Load-And-Follow" method was the least cost system. Providing that the system will produce the desired tonnage specified, the LAF system was optimal to a distance of 6000 feet, at grades of -5 to +2%. For the equipment shown, the Optimum Method charts show the LAF method as the least cost for the Mine Planning approach for some situations for distances slightly over 9000 feet.
2. For all of the specific situations studied, the haulage distance was already beyond that where LHD is the least cost for the tonnage specified. Yet, the Optimum Method charts show for the

equipment simulated and the lower tonnage, that LHD can be optimal at moderate grades out to 2300 feet.

3. For the situations simulated, there were only two of the 24 which produced the HFC as being the least cost method. In both of these cases the ore body, where the haulage was being simulated is long and narrow, thus the haulage to the chute was specified as 500 feet and the haul from the chute as 2500 and 3500 feet. But when only 1500 feet at a +3% was given as the truck haulage from the chute, the LAF became the optimal method. It can be determined from the Optimum Method charts that if either the grade had been increased to +7%, or the chute haulage distance increased to 2200 feet and all other factors remained the same, then the best method would have been HFC.
4. The three cases out of the 24 that were simulated, where the least cost method was the FEL system, were all cases of Mine Planning, where a large tonnage was required. This needs some qualifications: in the program for FEL, for both Mine Planning or Situation Planning, the number of trucks necessary to produce a given tonnage is determined, and the

cost is adjusted accordingly. But for LAF, the equipment is limited to only one loader and one truck and if it will not produce the required tonnage, it is not considered in the optimum determination. This may or may not be justified. If the cycle time is sufficiently long for two sets of LAF vehicles to work the stope with out queuing, then they should be considered and the solution reached here might be in error. But if the cycle time was so short, that the two sets would interfere, then in all likelihood the correction solution was identified here. This is a problem for future research. In the Optimum Method charts this is illustrated very well, since FEL was the optimum method at 85 TPH only at a distance to the shaft of between 8000 and 10000 feet, and where the haul to a possible chute as 1500 feet. But for the set of charts that show 170 TPH required, and a 1500 foot haul to a possible chute, FEL is shown to be optimal for all the equipment combinations used for all distances over 4000 feet. For some suboptimum combinations, FEL is the least cost method for all distances over 1200 feet, at all grades. Of course, in this latter case the best equipment for the task is not being simulated.

The final conclusion of this report is that a method of determining the least cost, or optimal method and combination of sizes or brands of equipment, has been developed for the Southeast Missouri mines of St. Joe Minerals Corporation.

This is not to say that the techniques developed here are not just as applicable to any trackless, or partially trackless mine. However, one must be very careful that the values assigned to the program variables do correspond to the mine, the equipment and the operating conditions being simulated. The value of repeated validations cannot be overstressed. It is the only way that the researcher will know if he has successfully built a model of the real mine system. It is also the only way that credibility will link the researcher with the management that must make the decisions. And finally, it is the only defense that the researcher or the decision makers will have if the new method of equipment is tried in an environment different than what was modeled and validated. It will give greater impetus for correcting a problem of operating environment, so that the expected productivity and cost can be achieved.

Suggested Future Research

Limiting the suggestions only to the continuation of the research contained here, the following work is suggested:

1. Develop accurate cost indexes from actual wage increases, bonus earnings and new equipment prices that can be applied externally to the Simulated Cost Tables, to update them on a regular period basis. This will take a minor amount of investigation but considerable amount of validation work.
2. Develop production and cost tables for many of the pieces of equipment not included in this particular study. These should include: Wagner ST 2A, 4A and 5A loaders and their entire range of truck sizes; Emico 911, 912 and 915 loaders; Joy Front End Loader; the entire line of Michigan and Hough loaders as well as Athely trucks.
3. It would be well to restructure the program to the stochastic approach to see if the standard deviation of the validation errors decreases. This would entail determining the probability value for each variable used.
4. The program needs to be tried simulating trackless haulage methods in several different mining methods, such as sublevel stoping or

sublevel caving. It is felt that this would be an extremely valuable tool not only to try to optimize equipment or method, but to optimize LHD haulage distances against development cost.

5. Another area of simulated information which might yield a good return would be that of illustrating the difference in cost and production between different policies of maintenance of equipment and the resulting down time.
6. Develop the other two methods of moving the ore, Load at Midpoint (LAM) and Load Haul Dump to and from an ore pass (LHD II), to the same extent that the other methods have been developed, and validate the work in the underground trackless mines.

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VITA

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He was awarded a Bachelor of Science degree in Mining Engineering from the University of Missouri-Rolla in 1951. After working a short time for the New Jersey Zinc Company, he was inducted into the United States Army. In 1955, he earned a Master of Science degree in Mining Engineering from the University of Missouri-Rolla. He then joined St. Joe Minerals Corporation as a research engineer. Over the past twenty years has held all supervisory positions in St. Joe's mining organization, and as their Director of Mining Research. Recently, he was appointed Corporate Director - Mine Development and Research for St. Joe.

He has been a licensed Professional Engineer of the State of Missouri and a member of the American Institute of Mining, Metallurgical and Petroleum Engineers for 24 years. He is also a member of the American Mining Congress, the Canadian Institute of Mining and Metallurgical Engineers, the International Society of Rock Mechanics and the Colorado Mining Association. He serves as an Executive Reservist to the Department of the Interior's Emergency Minerals Administration. On the University of Missouri-Rolla campus, he serves as a Director of the Gamma Xi House Corporation. He resides with his family in Viburnum, Missouri.

Appendix A

Equipment Data Sheets

(As Printed By Data Processing)

980 4 SPEED POWER SHIFT 23.50 x 25 6 TON BUCKET

PAYLOAD = 12000. EMPTY WEIGHT = 48000. SHIFT TIME = 0.0

RIMPULL VELOCITY CURVE

VELOCITY	RIMPULL
0.50	52000.
1.00	46500.
1.50	38500.
2.00	31500.
2.50	26000.
3.00	20000.
3.50	18000.
4.00	16200.
5.00	12500.
5.30	11300.
6.00	10500.
7.00	9200.
8.00	8200.
9.00	7200.
10.30	5800.
11.00	5500.
12.00	5200.
13.00	4900.
14.00	4600.
15.00	4400.
16.00	4100.
17.00	3900.
18.00	3700.
19.00	3500.
20.00	3000.
21.00	2800.
22.00	2400.
23.00	1900.
24.00	1100.
24.50	700.
25.00	500.
26.00	0.

SHIFTING SPEED	ROTATING MASS CON
3.00	0.34
5.30	0.16
10.20	0.07
26.00	0.05
0.0	0.0
0.0	0.0
0.0	0.0
0.0	0.0

988 3 SPEED POWER SHIFT 29.50 x 29 10 TON ST. JOE. BUCKET

PAYLOAD = 20000. EMPTY WEIGHT = 72000. SHIFT TIME = 0.0

RIMPULL VELOCITY CURVE

VELOCITY	RIMPULL
0.50	53500.
1.00	48000.
1.20	45000.
1.50	38500.
2.00	31000.
2.50	24000.
2.90	19200.
3.20	18300.
4.00	15300.
4.50	13700.
5.00	12200.
6.00	9800.
7.00	8000.
7.50	7600.
8.00	7300.
9.00	6500.
10.00	6000.
11.00	5400.
12.00	4900.
13.00	4500.
14.00	4200.
15.00	3800.
16.00	3500.
17.00	3300.
18.00	2900.
19.00	2500.
20.00	1600.
21.00	900.
22.40	0.
0.0	0.
0.0	0.
0.0	0.

SHIFTING SPEED	ROTATING MASS CON
2.90	0.37
7.00	0.12
22.40	0.05
0.0	0.0
0.0	0.0
0.0	0.0
0.0	0.0
0.0	0.0

992 3 SPEED POWER SHIFT 33.25 x 35 15 TON BUCKET

PAYLOAD = 30000. EMPTY WEIGHT = 127200. SHIFT TIME = 0.0

RIMPULL VELOCITY CURVE

VELOCITY	RIMPULL
0.50	82500.
1.00	77500.
1.50	69000.
2.00	59000.
2.50	50000.
3.00	42000.
3.50	33500.
3.70	29000.
4.00	28000.
5.00	24500.
6.00	21200.
7.00	18100.
8.00	15500.
8.50	12500.
9.00	12100.
10.00	11900.
11.00	10500.
12.00	10100.
13.00	9500.
14.00	9000.
15.00	8500.
16.00	8000.
17.00	7700.
18.00	7100.
19.00	6700.
20.00	5900.
21.00	4900.
22.00	4000.
23.00	2700.
24.00	1400.
25.00	700.
25.80	0.

SHIFTING SPEED	ROTATING MASS CON
3.70	0.25
8.50	0.09
25.80	0.05
0.0	0.0
0.0	0.0
0.0	0.0
0.0	0.0
0.0	0.0

TEREX 72-71 3 SPEED PS 29.5 x 29 GM 8V-71T DIESEL

PAYLOAD = 23500. EMPTY WEIGHT = 84325. SHIFT TIME = 0.0

RIMPULL VELOCITY CURVE

VELOCITY	RIMPULL
0.0	51249.
0.52	46859.
1.04	41782.
1.30	39939.
1.57	36964.
2.09	31506.
2.61	26923.
3.14	22335.
3.41	20009.
3.68	17573.
4.21	15649.
5.27	13373.
6.33	11094.
6.87	9939.
7.42	8728.
8.38	7876.
10.47	6730.
12.59	5583.
13.66	5002.
14.74	4393.
16.06	3660.
16.86	3139.
17.57	2638.
18.00	2300.
18.30	2115.
18.75	1750.
18.94	1637.
19.25	1350.
19.54	1312.
19.75	1175.
20.04	1036.
20.80	0.

SHIFTING SPEED	ROTATING MASS CON
3.68	0.24
7.42	0.11
20.80	0.06
0.0	0.0
0.0	0.0
0.0	0.0
0.0	0.0
0.0	0.0

EIMCO 920 4 SPEED PS 29.5 x 29 DETROIT DIESEL 12V71N55

PAYLOAD = 26000. EMPTY WEIGHT = 87000. SHIFT TIME = 0.0

RIMPULL VELOCITY CURVE

VELOCITY	RIMPULL
0.0	72547.
0.39	68833.
0.76	62192.
0.90	59000.
1.13	54659.
1.48	47109.
1.85	40407.
2.00	38000.
2.23	35094.
2.63	30104.
3.03	24452.
3.33	22437.
4.01	19487.
4.50	17500.
4.73	16716.
5.47	13578.
5.84	12807.
6.90	11250.
7.02	11123.
8.28	9541.
9.57	7750.
10.61	7049.
12.76	6122.
15.00	5750.
15.05	5251.
17.40	4266.
18.00	4000.
18.50	3379.
19.16	2544.
20.56	1213.
22.06	470.
24.03	0.

SHIFTING SPEED	ROTATING MASS CON
3.03	0.32
5.47	0.16
9.57	0.07
24.03	0.04
0.0	0.0
0.0	0.0
0.0	0.0
0.0	0.0

ST-8 4 SPEED PS 26.5 x 25 DEUTZ F10L-714 ENGINE

PAYLOAD = 21000. EMPTY WEIGHT = 60780. SHIFT TIME = 0.0

RIMPULL VELOCITY CURVE

VELOCITY	RIMPULL
0.53	54689.
0.74	50212.
0.95	44674.
1.48	33090.
2.01	24460.
2.66	18031.
3.07	16014.
3.48	14235.
4.10	11862.
4.51	10344.
5.21	9213.
5.61	8691.
6.01	8182.
6.41	7722.
7.21	6849.
7.62	6425.
8.02	6061.
8.45	5661.
9.35	5018.
10.13	4738.
10.91	4470.
11.68	4208.
12.47	3971.
14.03	3522.
15.58	3117.
17.14	2718.
17.92	2525.
18.70	2307.
20.26	1484.
21.82	655.
23.38	125.
24.52	0.

SHIFTING SPEED	ROTATING MASS CON
2.40	0.59
4.60	0.19
8.80	0.09
24.52	0.05
0.0	0.0
0.0	0.0
0.0	0.0
0.0	0.0

ST-11 4 SPEED PS 29.5 x 29 DEUTZ F12L-714 ENGINE

PAYLOAD = 27500. EMPTY WEIGHT = 95250. SHIFT TIME = 0.0

RIMPULL VELOCITY CURVE

VELOCITY	RIMPULL
0.53	65500.
0.74	58812.
0.98	51260.
1.49	37530.
2.02	27232.
2.48	21763.
3.09	18234.
3.51	16117.
3.92	13999.
4.44	11842.
4.85	11120.
5.66	9858.
6.06	9317.
6.46	8776.
7.27	7670.
8.08	6672.
8.48	6227.
8.88	5891.
9.43	5718.
10.21	5366.
10.99	5069.
11.78	4791.
13.35	4235.
14.14	3944.
14.93	3678.
16.49	3202.
17.28	3029.
18.07	2844.
19.64	2300.
21.21	1360.
22.78	606.
24.79	0.

SHIFTING SPEED	ROTATING MASS CON
2.25	0.50
4.40	0.20
9.25	0.07
24.79	0.04
0.0	0.0
0.0	0.0
0.0	0.0
0.0	0.0

621 300 HP 8 SPEED PS 26.50 x 29 EJECT-ALL WAGON

PAYLOAD = 60000. EMPTY WEIGHT = 58370 SHIFT TIME = 0.0

RIMPULL VELOCITY CURVE

VELOCITY	RIMPULL
0.50	38700.
1.50	34700.
2.90	25750.
4.40	17200.
4.50	16600.
4.80	16600.
5.10	16400.
6.40	14300.
6.70	13400.
7.00	12100.
7.70	11650.
8.60	10600.
9.10	10000.
9.90	8870.
10.40	8650.
11.00	8320.
12.20	7330.
12.60	6710.
13.40	6560.
14.90	6150.
15.60	5860.
16.30	5560.
16.50	5430.
17.00	4970.
20.00	4560.
22.00	4120.
22.20	4040.
23.00	3650.
27.00	3380.
30.00	3020.
31.00	1560.
32.30	0.

SHIFTING SPEED	ROTATING MASS CON
4.50	0.20
6.70	0.67
9.10	0.44
12.20	0.26
16.50	0.17
22.20	0.11
32.30	0.07
0.0	0.0

631C 415 HP 8 SPEED PS 29.50 x 35 EJECT-ALL WAGON

PAYLOAD = 80000. EMPTY WEIGHT = 76800. SHIFT TIME = 0.0

RIMPULL VELOCITY CURVE

VELOCITY	RIMPULL
0.30	83900.
0.80	80300.
1.70	60800.
2.30	44700.
2.60	37050.
3.10	33600.
4.20	24650.
4.60	21400.
5.40	20800.
6.10	19750.
6.40	19200.
6.70	15650.
6.80	15600.
7.70	15000.
8.70	14200.
9.10	11650.
9.20	11600.
10.40	11200.
11.60	10600.
12.30	8600.
13.20	8500.
14.10	8300.
15.70	7800.
16.60	6400.
16.70	6350.
19.00	6250.
22.00	5950.
22.40	4700.
22.60	4700.
25.60	4550.
28.70	4350.
33.40	0.

SHIFTING SPEED	ROTATING MASS CON
2.60	0.44
4.60	0.19
6.80	0.64
9.10	0.43
12.30	0.25
16.60	0.16
22.40	0.11
33.40	0.06

641B 550 HP 8 SPEED PS 33.50 x 39 EJECT-ALL WAGON

PAYLOAD = 100000. EMPTY WEIGHT = 98800. SHIFT TIME = 0.0

RIMPULL VELOCITY CURVE

VELOCITY	RIMPULL
0.30	140400.
1.00	114450.
1.90	75550.
2.40	57700.
2.70	50500.
2.90	47600.
3.90	36100.
4.70	29700.
4.90	29600.
6.20	26250.
6.50	21900.
6.60	21850.
7.50	20900.
8.40	19400.
8.80	16300.
8.90	16300.
11.30	14500.
11.80	12050.
12.00	12050.
13.60	11500.
15.30	10700.
16.00	9000.
16.10	9000.
18.30	8600.
20.40	8000.
21.40	6600.
21.80	6600.
23.30	6502.
24.70	6300.
27.70	5900.
30.50	3850.
31.90	0.

SHIFTING SPEED	ROTATING MASS CON
2.70	0.46
4.70	0.20
6.50	0.72
8.80	0.45
11.80	0.30
16.00	0.17
21.40	0.12
31.90	0.07

Appendix B

Derivation of Production and Cost Formulas

A. Production Planning Formulae

1. LHD: Since the tables of cost and production were sent up for LHD, this information can be read directly from the tables and applied to any number of loaders:

$$P_{r-1} = L_c \times L_n \tag{3}$$

$$S_{\$-1} = L_{\$} \tag{4}$$

2. LHD-HFC: As above, the loader production and cost can be obtained directly from the tables. Let "P_L" be the loader production, then:

$$P_L = L_c \times L_n$$

The production and cost tables were set up for the trucks moving and dumping ore, but a time and cost factor must be added for the loading with a chute. The chutes that are used by St. Joe load at the rate of 23 tons per minute. This figure was developed by time study of the No. 28 mine chute, loading a 27 ton truck (Rated capacity, 28 tons). Actually, the loading time average was 1.17 $\pm$ .17 minutes for 30 loads. Considering the 83 percent availability and 90 percent efficiency, there are 44.82 working minutes in each hour. Therefore, using "P_T" as the trucks production moving rock from the chute, it became:

$$P_T = T_n \; T_{BC} \left[\frac{44.82}{T_T + \frac{T_{bc}}{23}}\right]$$

$$= \frac{1030 \; T_{bc} \; T_n}{23T_t + T_{bc}}$$

P_{r-2} is the smaller of P_T or P_L (5)

In all probability, a mismatch will exist between the two types of equipment in the system which will result in lost time due to either loaders waiting for dumping room or trucks waiting for ore to haul. The cost for the system must be adjusted to reflect this loss of time and production by increasing cost in the ratio of potential production to system production. Use of this ratio maintains the cost per hour, as determined by ownership and operating cost, at a constant level. The system cost must also include the contribution made by the chute to the overall production cost. This chute cost is fully described in the section on chute spacing optimization. Considering these factors, the system cost becomes:

$$S_\$ = L_\$ \left[\frac{L_c \times L_n}{P_{r-2}}\right] + \frac{1030 \; T_\$ \; T_{bc} \; T_n}{P_{r-2} \; (23T_T + T_{bc})} + \frac{1.90}{P_{r-2}} \qquad (6)$$

3. FEL: This system assumed the use of only one loader, and one or more trucks. From time study

information, the average time to load one dipper into a truck was 1.16 minutes (including time for laying out boulders, scraping up, etc.). Therefore, the time to load one truck is:

$$L_{tr} = \frac{T_{bc}}{D_c} \times 1.16$$

combine this time with the truck's haul time, taken directly from the table, the truck cycle time becomes:

$$T_{ct} = L_{tr} + T_t$$

The truck production, as indicated in the table, will be reduced according to the table cycle time. On this basis, the system production will be:

$$P_{r-3} = \frac{T_n \times T_c}{\left[\frac{T_t + 1.16\ (T_{bc}/D_c)}{T_t}\right]} \tag{7}$$

This rate may be any rate up to the loader's zero distance capacity, as given in the tables, L_{co}. Therefore:

$$P_{r-3} \leq L_{co}$$

The zero distance loading cost given in the tables assumes full production and must be adjusted according to the ratio of zero distance loading capacity to system production. Likewise, the truck cost must reflect an increase in the ratio of truck production, to system production. These adjustments account for any mismatch of equipment which may exist, and maintains a constant cost per hour for each piece of equipment. In addition to these adjustments, \$.04 per ton has been added to the systems cost to account for increased truck abuse, increased ventilation requirements, and truck driver bonus. Considering these factors, the system cost becomes:

$$S_{\$-3} = L_{\$o} \left[\frac{L_{co}}{P_{r-3}} \right] + T_{\$} \left[\frac{T_c \times T_n}{P_{5-3}} \right] + 0.04 \qquad (8)$$

4. FEL-LAF: This system assumes only one loader and one truck working together. The time to load one truck is the same as the FEL system:

$$L_{tr} = 1.16 \times \frac{T_{bc}}{D_c}$$

The truck cycle time and the loader cycle time can be taken directly from the tables. The system time is the time to load the truck plus the larger of the loader cycle time or the truck cycle time:

$$S_t = L_{tr} + L_t \text{ or } L_{tr} + T_t$$

Considering the 0.83 availability and 0.90 efficiency, there are 44.82 working minutes in each hour. Therefore, the system production is:

$$P_{r-4} = \left[\frac{44.82}{1.16(T_{bc}/D_c) + (L_t \text{ or } T_t)}\right] (T_{bc} + D_c) \qquad (9)$$

The truck costs must be weighted according to the ratio of potential production and then both costs must be balanced according to the cost capacity of the truck and the cost capacity of the loader. On this basis, the system cost is:

$$S_{\$-4} = \left\{\frac{D_c L_\$ + T_{bc}\left[\frac{T_c T_\$}{P_{r-4}} + L_{\$o}\right]}{T_{bc} + D_c}\right\} + 0.04 \qquad (10)$$

As in FEL, the factor of \$0.04/ton has been added to compensate for taking the truck into the stope.

B. Mine and Situation Planning Formula

These equations are intended for use to select equipment on the basis of its production potential and a required tonnage over a planned distance and grade of haulage road. Situation planning gives the equipment required in fractional parts and its exact cost. (Noted by the subscript S). Mine planning rounds up to the next whole piece of equipment and adjusts cost according to the mismatch which results from idle time. (Noted by the subscript M.)

1. LHD: Take the loader capacity directly from the production table at the specified distance and grade.

$$L_n = P_r/L_c \tag{11}$$

For situation planning, read the loader cost directly from the cost table.

$$L_\$ = \text{Direct from Table} \qquad (12\text{-S})^*$$

For mine planning, modify the loader cost in the ration of potential production to required tonnage:

$$L_{\$a} = L_\$ \left[\frac{L_c L_n}{P_r}\right] \qquad (12\text{-M})^*$$

2. LHD-HFC: Take the loader capacity directly from the production tables at the specified distances and grades. The truck capacity must be adjusted to account for the chute loading time of 23 tons per minute.

$$L_n = \frac{P_r}{L_c} \tag{13}$$

$$T_n = \frac{P_r}{T_{bc} \left\{\dfrac{44.82}{T_t + \dfrac{T_{bc}}{23}}\right\}}$$

*S = Situation Planning Formula

M = Mine Planning Formula

$$T_n = \frac{P_r\ (23T_t + T_{bc})}{1030\ T_{bc}} \qquad (14)$$

For situation planning, read the loader and truck costs directly from the cost tables. Include \$1.90 per hour for the ownership and operation of the chute and feeder:

$$S_\$ = L_\$ + T_\$ + \frac{1.90}{P_r} \qquad (15\text{-S})$$

For mine planning, modify the loader and truck costs in the ratio of their potential productions to the required tonnage:

$$S_{\$a} = L_\$ \left[\frac{L_c L_n}{P_r}\right] + \frac{T_\$ T_{bc} T_n \left\{\dfrac{44.82}{T_t + \dfrac{T_{bc}}{23}}\right\}}{P_r} +$$

$$\frac{1.90}{P_r} \qquad (15\text{-M})$$

$$S_{\$a} = L_\$ \left[\frac{L_c L_n}{P_r}\right] + \frac{1030\ T_\$ T_{bc} T_n}{P_r (23\ T_T + T_{bc})} +$$

$$\frac{1.90}{P_r}$$

3. FEL: This method is limited to one loader working with as many trucks as required. The time to load one truck (L_{tr}) is:

$$L_{tr} = \frac{T_{bc}}{D_c} \times 1.16$$

Find the truck cycle time in the production table at the specified distance and grade. Based on 0.83 availability and 0.90 efficiency, there are 44.82 working minutes per hour. The number of required trucks becomes:

$$T_n = \frac{P_r}{\left[\dfrac{44.82}{1.16(T_{bc}/D_c) + T_t} \times T_{bc}\right]} \tag{16}$$

The maximum number of trucks one loader can keep busy is:

$$T_{max} = \frac{1.16(T_{bc}/D_c) + T_t}{1.16(T_{bc}/D_c)} \tag{17}$$

and

$$T_n \leq T_{max}$$

The zero distance loading cost given in the tables assumes full production and must be adjusted according to the ratio of zero distance loading capacity to required tonnage. Likewise, the truck cost must be increased in the ratio of truck production to required tonnage. To compensate for taking the truck into the stope, \$0.04 per ton is added to obtain the system cost. This cost calculation is the same for situation and mine planning.

$$S_\$ = L_{\$o} \left[\frac{L_{co}}{P_r} \right] + T_\$ \left[\frac{T_c T_n}{P_r} \right] + 0.04 \qquad (18\text{-S\&M})$$

4. FEL-LAF: This system assumes only one loader and one truck working together. The time to load one truck is the same as the FEL method:

$$L_{tr} = 1.16 \times \frac{T_{bc}}{D_c}$$

The truck cycle time and the loader cycle time can be taken directly from the tables. The system time is the larger of the loader cycle time or the truck cycle time added to the time to load the truck:

$$S_t = 1.16(T_{bc}/D_c) + (\text{The larger of } T_t \text{ or } L_t)$$

With 44.82 working minutes per hour, the system production is:

$$S_c = \frac{44.82}{S_t} \quad (T_{bc} + D_c) \qquad (19)$$

For the system to be acceptable:

$$S_c \geq P_r$$

The truck cost must be weighted according to the ratio of potential production to required production. Both loader and truck costs must be balanced according to the cost capacity of the loader and the cost capacity of the truck. On this basis, the system cost for situation planning:

$$S_{\$} = \frac{D_c L_{\$} + T_{bc}\left[\dfrac{T_{\$} T_c}{S_c} + L_{\$o}\right]}{T_{bc} + D_c} + 0.04 \qquad (20\text{-S})$$

For mine planning, this cost must be modified in the ratio of system production to required production to compensate for the mismatch between the capability of the equipment and the requirements of the applications:

$$S_{\$a} = \frac{S_c \left\{\dfrac{D_c L_{\$} + T_{bc}\left[\dfrac{T_{\$} T_c}{S_c} + L_{\$o}\right]}{(T_{bc} + D_c)}\right\}}{P_r} \qquad (20\text{-M})$$

C. Examples of the Use of Production and Cost Formulas

Situation: The Mine Captain wishes to use a Caterpillar 988 Loader and an E621 truck to move ore from a stope 2625 feet, at a +3.5% grade. If he wished to put in an ore pass and feeder chute, the LHD to the chute will average 1500 at +4% grade and the haul from the chute to the shaft will be 1500 at +1%. What method would optimize production?

Variables needed to be identified:

L_c = 62.13* TPH (Assumed 10 ton capacity)

L_{c_2} = 86.78* TPH

L_n = 1

L_t = 7.53 Minutes*

T_n = 1

T_{bc} = 30 tons (Assumed)

T_t = 7.09

T_c = 189.9

Solution: Production Planning

1. LHD:

$$P_{r-1} = L_{c_1} \times L_n = 62.13 \times 1 = 62.13 \text{ TPH}$$

2. LHD-HFC:

$$P_{r-1} = \frac{1030\ T_{bc}\ T_n}{23\ T_t + T_{bc}}$$

$$= \frac{1030 \times 30 \times 1}{23 \times 7.09 + 30} = 160.1 \text{ TPH/Truck}$$

$$P_{L_2} = L_{c_2} \times L_N$$

$$= 86.78 \times 1 = 86.78$$

$$P_{r-2} = 86.78 \text{ TPH/Loader}$$

*Interpolated between 2500 and 2750 feet distance and between 3.0% and 4.0% grade from Product on Table. For T_t, T_c see Table I.

3. FEL:

$$P_{r-3} = \frac{T_n \times T_c}{T_t + 1.16(T_{bc}/D_c)}$$

$$= \frac{\frac{1 \times 189.9}{7.09 + 1.16 \quad (30/10)}}{7.09} = 127.4 \text{ TPH}$$

4. LAF:

$$P_{r-4} = \frac{4482}{1.16 \ ({}^{T}bc/D_c) + (L_T \text{ or } L_L)} \qquad (T_{bc} + D_c)$$

$$= \frac{44.82}{1.16 \ (30/10) + 7.53} \ (30 + 10)$$

$$= 4.07 \times 40 = 162.83 \text{ TPH}$$

Therefore, the maximum production would come from the LAF combinations. If he wanted to determine the least cost method it would be found in the same manner, only looking up $L_{\$}$, $L_{\$_o}$, and $T_{\$}$. It is interesting to note, that LAF would even produce greater TPH than two 988's LHD-HFC with the truck capacity of 160.1 TPH being the limiting factor.