SYSTEM ANALYSIS FOR SHOVEL-TRUCK PRODUCTIVITY

by

Aung Shein

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Department of Mineral Resources Engineering
Royal School of Mines
Imperial College of Science and Technology
London SW7 2BP

ABSTRACT

The selection of the proper shovel-truck combination, which will result in the lowest cost per ton of material being moved, is one of the major decisions in open-pit mine planning. The profitability of an open-pit mine is critically linked to the efficiency of the shovel and truck being used.

The research mainly deals with (1) the study of the methods used in predicting the productivity of an open-pit shovel-truck operation (2) the determination of optimum truck fleet size for a given shovel at a given production rate and (3) the selection of optimum shovel and truck combination for a given set of operating conditions.

The methods considered in the study are :

(1) Conventional Deterministic Estimating Procedure.
Where production estimates are determined by use of empirical formulae.

(2) Method applying Queueing Theory.
Where cost comparisons are made on the basis of truck waiting times and shovel idle times being considered.

(3) Computerized Simulation Method.
Where production estimates are determined by using Monte-Carlo Simulation Technique.

Computer Programs are developed for each method and the results are analysed and compared. The Programs can be used not only for analysis of needs for a new open-pit mine but for cost improvement in existing mines where changes in truck and shovel operations are contemplated.
Combinations of different capacities of shovel and truck sizes are considered in the study.

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1.0. INTRODUCTION

Truck haulage is widely used in open-pit and quarrying operations together with the loading machines. Loading and hauling are two major expenses for open-pit mine operator. Hauling, alone in some cases, can account for up to 30 to 50 % of the total mining costs (Mueller,1979). Truck-fleet productivity in open-pit mines has the lowest improvement rate among the three major unit operations: drilling, loading, and hauling (Michaelson,1974). In addition, trucks require individual labour, high maintenance and relatively frequent replacement making them sensitive to inflation. In 1981, a survey of several U.S. mines was conducted to determine vehicle component life and repair costs levels. The results showed that component life is directly proportional to hauling conditions and quality of preventive maintenance. The major operating cost areas turned out to be tyres, fuel oils, road repairs, and labour. Maintenance costs for trucks, as a percentage of the total investment costs, run about 14% as compared to conveyor belts and aerial rope ways which are around 2.4 % and 2.9 % respectively (Sheehan,1985).

Increasing depth of mining and consequent increase in stripping ratios require additional equipment to be used. As a result, management is faced with the problem of buying additional trucks or shovels if there is an improper balance of equipment in the mining operation. This problem usually results from inadequate use of haulage resourses.

Recent increases of the energy cost together with projected future increases will further increase truck-fleet capital and operating costs in the future. The impact of increasing fuel costs, combined with unprecedented high inflation has had a dramatic effect on truck operating costs. According to Burton (1981), a typical direct hourly operating cost for a 68 ton (75 st) truck had increased a staggering 150 % over the period 1973 to 1981 and also, fuel and lube component had almost trippled its share of the hourly cost from about 10 % to 29 % and it is noted that the cost of fuel has escalated faster than the overall rate of inflation.

All these factors have led mine operators to search for methods and ways to improve the effectiveness of the shovel-truck fleet, in order to lower costs and maintain a profitable operation in the face of declining markets and increasing worldwide competition.

Efforts to improve the truck efficiency are centered around engine improvements, improvements in the horse power to weight ratio, and various schemes aimed at optimizing truck utilization and minimization costs (Burton, 1981).

In this research, the methods, aimed at selecting the optimum combination of shovel-truck system with the lowest cost per ton of material moved, are examined. The selection of optimum shovel-truck combination is one of the major decisions in open-pit mine planning. Among the important questions that need to be answered prior to making an actual decision are the following:

(1) What size truck operates most efficiently in the

operating system for a given shovel ?

(2) How many trucks are required to minimize the waiting time of a given shovel ?

(3) What physical changes in the haulage layout are warrented to increase production ?

(4) What will be the total production and unit operating cost of a given fleet of trucks and shovels ?

(5) When new trucks and shovels are required, which ones should be selected to operate most efficiently ?

To aid mine management in answering the above questions, several computer models, based on different methods of predicting shovel-truck productivity, have been developed. The main objective of the research is to describe these methods and models and how they are used to evaluate the primary equipment requirements for a given set of mining conditions.

2.0. METHODS FOR PREDICTING SHOVEL-TRUCK PERFORMANCE

Various methods are used to determine shovel-truck productivity. Clearly, the productivity of a load-haul system is dynamic, that is, the system performance varies with time. Traditionally, time and motion studies are carried out and simple conventional method is applied to obtain the productivity performance in smaller fleets and simple situations. However, in complex systems, this approach makes it difficult to analyse the effect of interaction between the various system components. Computerized simulation models would best suit for such complex systems. The methods examined in the study are as follows:

(1) Conventional Deterministic Estimating Procedure -- Where production estimates are determined by using empirical formulae.

(2) Method applying Queueing Theory -- Where cost comparisons are made on the basis of truck waiting times and shovel idle times being considered.

(3) Computerized Simulation Technique -- Stochastic or Monte Carlo Simulation Technique is used to obtain production estimates.

2.1. BACKGROUND

2.1.1. CONVENTIONAL DETERMINISTIC ESTIMATING PROCEDURE

This method, as the name implies, is a simple method that can be carried out manually by using empirical formulae. It relies a great deal on past experience to modify the outcome so as to make it more realistic. The method starts with an arbitrary selection of shovel and truck sizes, and, from the manufacturer's truck performance curve, an average haul speeds both for loaded and empty are computed. Flachsenberg (1964) and Watkin (1965) used an average overall speed to calculate trip time rather than break the haul profile into various segments. A refinement to this method was introduced by Fitzpatrick (1967), Bishop (1968) and Morgan (1968). They used different speed factors for different segments to convert the maximum speed of the vehicle to an average speed for each segment of the haul profile which has been broken down into various segments. These haul roads are divided into segments corresponding to changes in grade or surface conditions. Once the average speeds are obtained, the travel times for each segment are calculated and then added to give an estimate for the total travel time.

The method uses the average values throughout the computation. Loading times are based on the loader's characteristics, the type and state of the material being loaded, the loader's target and etc.; again the plant manufacturers' performance handbook is usually consulted. Other considerations are the number of buckets per truck and the form of loading (e.g. whether single or double sided).

Manuever and dump times are similarly obtained from the plant manufacturers' performance handbook. Such values are usually given as fixed values for a given type of plant operating under specified conditions while the values themselves have typically been obtained from time studies in the field.

Having obtained these average values for most of the components of the cycle, experienced estimators will then modify these values through some judgemental input. Estimates are still required for waiting times but conventional deterministic estimation procedure gives little insight into this component. According to Bohnet and Janson

(1984) , that smaller fleets can use average values to compute the expected output per time unit, however the figures do not reveal the whole picture of the operation since the production capacity can be limited by either the loading function or hauling function.

In conventional deterministic eatimation procedure delays due to management, supervision and labour deficiencies, job and weather conditions, etc. , are taken into account as a job operational factor, whereas external delays such as delays caused by repair and maintenance, machine breakdowns, moving equipment, etc. , are accounted for in the method by use of an availability factor.

Douglas (1964), Morgan and Peterson (1968) and Deakin (1978) have stated that in this method, the system production is dependent on the relative output of the shovel versus the trucks, the lesser of the two governing the output of the of the system. They pointed out the effect of equipment mismatching on the production capacity of a shovel-truck system as illustrated in figure 2.1.

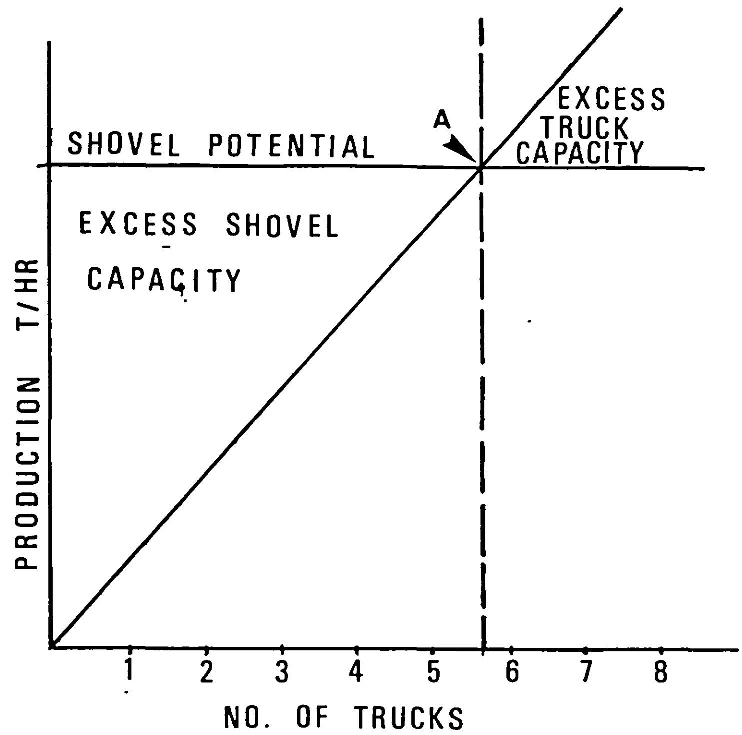

FIG. 2.1. PRODUCTION POTENTIALS VERSUS MISMATCHING EFFECT.

The figure shows the ideal situation, with point 'A' being the balance point or match point of the shovel and truck output. It will be observed that when the number of trucks is less than the number required to balance or match, the truck output appears to limit the system output. When the number of trucks is greater than the balance number, the shovel output is the apparent limit.

Deterministic procedure does not take into account bunching of haul units whereby queues develop and at other times the loader is not busy. Douglas (1964), Morgan and Peterson (1968), and Deakin (1978) state that there will be some additional loss in production due to the effect of bunching, because of variation in the haul unit's cycle times. What happens is more like the example shown in figure 2.2, where the shaded area indicates the loss of production due to bunching.

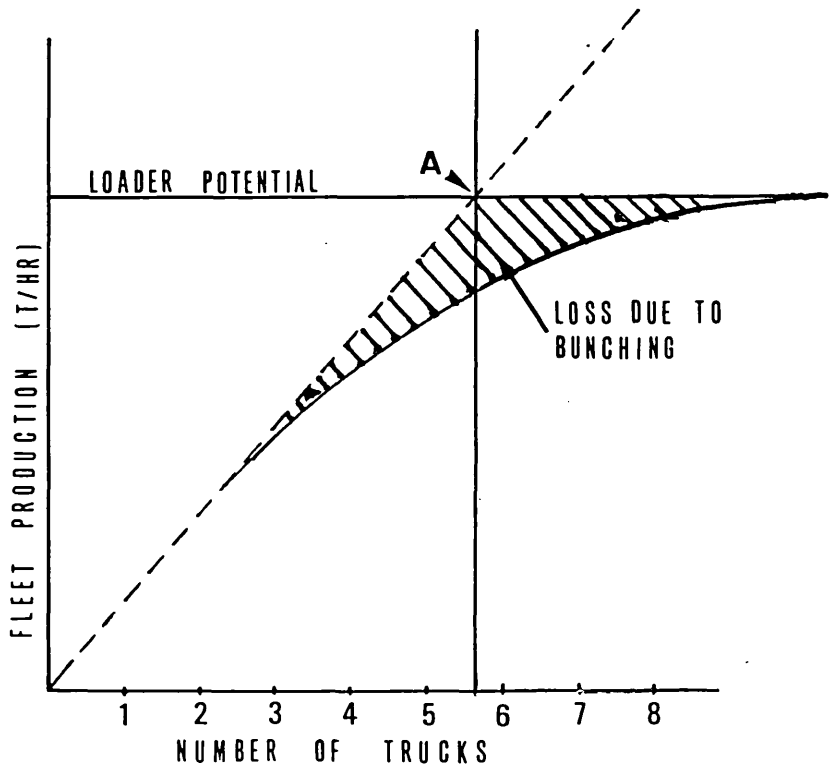

FIG. 2.2. THE EFFECT OF BUNCHING ON PRODUCTION SYSTEM

This bunching or irregular arrival of the hauling units at the loading area causes a further reduction in operating efficiency and the greater the bunching the greater the loss in efficiency.

Teicholz and Douglas (1963) and Morgan and Peterson (1968) have found that the most accurate way of determining the effect of bunching is to carry out a stochastic simulation of the operation in question. However, they have suggested methods whereby the conventional deterministic estimation procedure can be converted into a more realistic values. Both methods are similar and have been developed through the comparison of conventional results with simulation studies. Teicholz and Douglas (1963) show overestimates of production of from 3 % to 11 % using conventional estimates. Substantial overestimates occur, according to Douglas (1964), at a system configuration where the output of the hauling units is approximately balanced by the output of the shovel.

Brooks and Shaffer (1971) quote average overestimate errors of 12.5 % when using conventional deterministic estimation procedure when compared with site values. The site study involved a dragline and a fleet of 8 to 12 dump trucks.

2.1.2. METHOD APPLYING QUEUEING THEORY

The occurrence of queues is common place in construction and mining engineering. The queues occur because a demand for service exceeds the capacity to provide that service for part of the time that the engineering operation is in progress.

An example is the loading of a fleet of trucks by an excavator. The units (or) customers that require service are the trucks which queue at the load point. The excavator 'serves' the trucks by loading them; the loaded trucks then pass out of the 'system'. Schematically this is shown in figure 2.3.

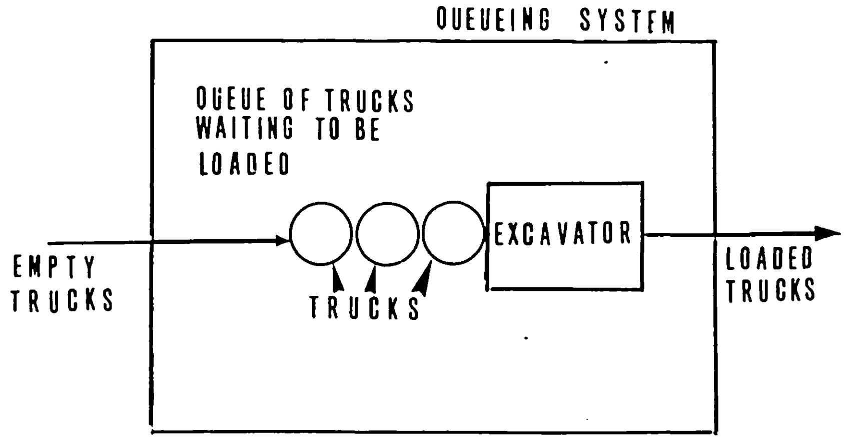

FIG.2.3. AN EXAMPLE OF QUEUEING SYSTEM

Generally speaking, there is some randomness associated with both the arrival of units to be serviced and the servicing of units. The interaction between the randomness of the interarrival times of the trucks and the shovel service time causes either a waiting line to form at the shovel, or leaves it idle. This situation suggests that a queueing approach may be of aid in analyzing how much time is lost in waiting and idling for both the trucks and shovel. Figures 2.4 and 2.5 show the various types of shovel and truck delays that one may encounter in a typical shovel-truck operation.

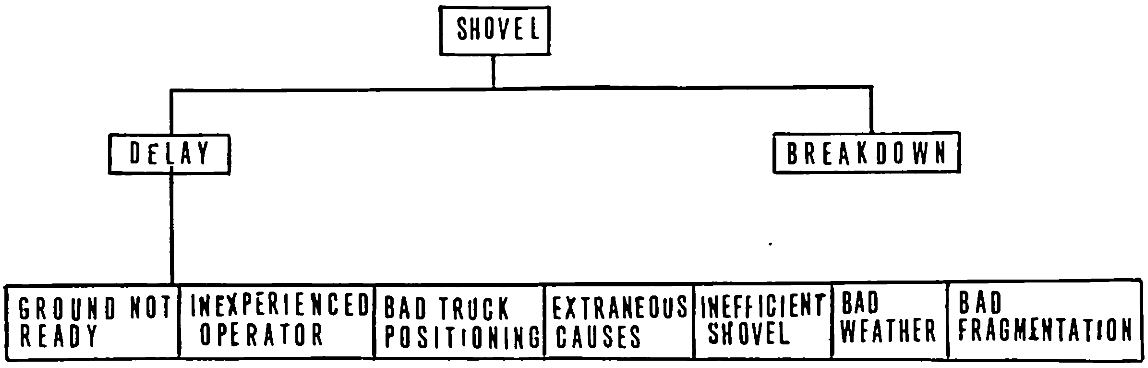

FIG. 2.4 TYPES OF SHOVEL DELAY (AFTER CHATTERJEE ET AL.,1971, THE QUARRY MANAGERS' JOURNAL)

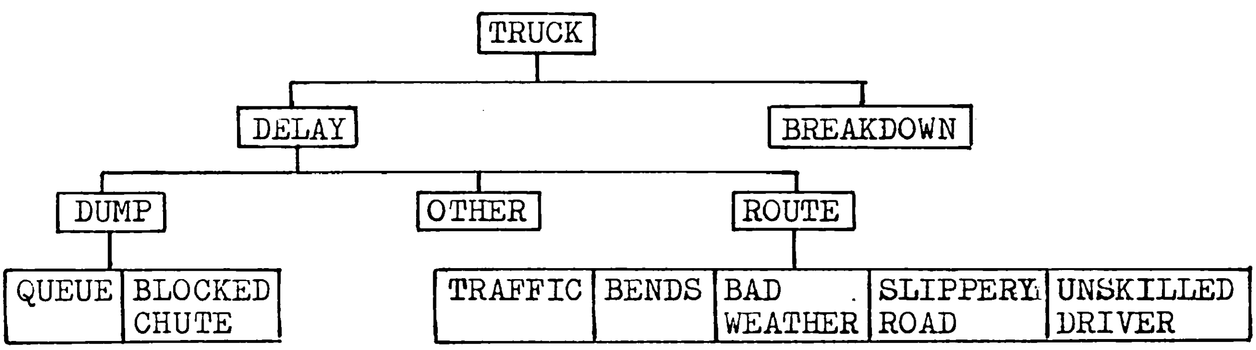

FIG.2.5. TYPES OF TRUCK DELAY (AFTER CHATTERJEE ET AL., 1971,THE QUARRY MANAGERS' JOURNAL)

The initial emphasis in queueing theory was in telephone applications. However in the 1940's work started to appear on applications in traffic and transportation and in machine repair/maintenance and the provision of spares (Carmichael, 1987). The first mining application appears to be by Koenigsberg in 1958 where also the idea of a cyclic queue was formalized (Carmichael, 1987).

Early applications of queueing theory to earthmoving, open-cut mining and quarrying operations were concerned with determining a production index (equivalently, server utilization) from which the production of the operation could be evaluated. Generally, attention centered on (i) the use of exponential distributions for both the loading and travelling and (ii) the single loader case (Carmichael, 1987). Morgan (1969) demonstrates the use of queueing theory to determine the average times lost through waiting in a queue. He also demonstrates an economic analysis of a shovel-truck fleet. Single server, finite calling population model was used in his study. Graff (1971) shows how queueuing theory was used to determine the shovel-truck waiting and idling times for a different number of trucks in the operating system. These waiting and idle times were used to compare the different economics of the fleet sizes. However, the approach used by Graff is highly questionable for one reason : the model he used is an infinite queue model for a single server finite queue system. For this reason, his results seem suspect.

Chelst, Tilles and Pipis (1981) apply queueing theory to analyse a coal train unloading system. The model was also employed to examine the relationships between the number of cars, coal throughput and queueing delays. The model was originally developed to dertermine the importance of adding a second unloader to the system. The model also served to identify several alternatives for increasing coal throughput and reducing queueing delays, in addition to satisfying its original purpose. A finite queueing model was used in their model formulation.

2.1.3. COMPUTERIZED SIMULATION TECHNIQUE

There are many factors which affect ore transportation in a surface mining load and haul operation. Many of these factors are very complex and the degree of their influence is very difficult to assess. However, some of the factors and their effects can be mathematically derived.These are often used to determine a theoretical value of vehicle performance. No matter how accurately a theoretical value of vehicle performance is derived, in actual practice it tends to fall short of the derived values. This is because the unavoidable and unpredictable elements in a load haul operation , and whose significance is considerable, tend to be disregarded in an empirical derivation of machine performance. These unavoidable and unpredictable elements contribute to what is known as 'queueing' and 'waiting time', which in turn lead to the variation of haul units cycle times and consequently the effect of bunching of haul units is occured.

The most accurate way of determining the effect of this bunching is to make a stochastic simulation of the operation in question (Morgan and Peterson, 1968). In such a simulation it is possible to determine the probable wait time for the shovel or truck as each truck approaches the loading area. By considering this waiting time for each individual case it is possible to determine the overall productivity of the fleet for a long time period.

Morgan and Peterson (1968), by using a stochastic simulation, discuss the problem of predicting travel times for new haul road conditions, a problem faced when trying to forecast the effect of modifications to an existing operation. They also state that how the effect of bunching can be handled by use of a stochastic simulation technique in the problem. Actual average travel times measured by Morgan and Peterson were up to 21 % greater than calculated average travel times. Cross and Williamson (1969) used a digital computer simulation to evaluate the effects of using dispatching in a mining system. The system was implemented at Pima Mining Company, Tucson with good results. It was also found that the simulation was able to identify the point of over-trucking on a shovel. This point determines the minimum size truck fleet needed to enable the mine to attain its operational goals.

Trafton and Kochanowsky (1969) describe how a simulation of a truck-shovel operation at a German limestone mine was used to find the optimal truck allocation for quality, quantity and cost restrictions.

O' Neil and Manula (1967) use a deterministic simulation to provide the cycle times which are then used as input data for a stochastic simulation. The simulator was originally designed to assign equipment optimally in an open-pit mine. However, they found that the simulator can also be used to (1) determine the best haulage route profile for a given mine and equipment (2) determine the best load weight for a given truck type on a given haulage profile such that maximum productivity is achieved and (3) evaluate quantitatively the effects on production of altering the rolling resistance of haul roads, thereby producing useful information to justify expenditures for better road maintenance.

Chatterjee and Hellewell (1971) use a stochastic simulation to show how the bunching effect due to the unavoidable and unpredictable elements can be handled. It was found that the results show that the theoretical tonnage computed is never achieved. The simulator was originally developed to allocate the load and haul resources to do the job in such a way that the overall efficiency is maximized and the total cost minimized.

Manula, Mohibatsela and Ramani (1980) have developed a simulation model to evaluate equipment performance on selected pit profiles to generate data required to choose coal and waste removal equipment for a mine site located in

Southern Anthracite field of Eastern Pennsylvania (1980). The simulator can also be used in applications such as : (1) the evaluation of changes in equipment sizes and haul road design on productivity (2) scheduling truck and shovel assignment to balance raw coal output (3) analysis of system performance resulting from the effects of blasting conditions on loading times and (4) evaluation of new and novel systems and equipment such as in-pit conveyor haulage. The simulator was developed as part deterministic, in which the truck's load capacity and performance characteristics are dealt and part stochastic, in which the loading cycle, spotting and dumping times are concerned.

Madge (1964) demonstrates a stochastic simulation of truck movements in an open-pit mining operation consisting of two pits symmetrically located with respect to a concentrator site. The simulator was designed to determine fleet requirements and to assist in the exploration of alternative ore removal procedures at the pit located in Northern Canada at Pine Point, Northwest Territories.

Gibbs, Gross and Pfleider (1967) have developed a series of three computer programs namely Program Least Square, Program Truck Haul and Program Haul Cost to carry out the comparative evaluation of various combinations of shovel-truck performances and costs. They have demonstrated how these programs permit the analysis of a multitude of variables through simulation and the determination of those factors which are most critical for a particular mining operation. The programs were developed originally to assist the management of a new mine in their equipment selection decision process.

The declining grades of surface deposits have reconfirmed the need for greater production rates. Large scale production requires increased investment in expensive and sophisticated equipment. The key to successful operations can depend entirely on proper equipment utilisation. According to Brake and Chatterjee (1979), the equipment utilisation figures in many current operations amount to 38 % for trucks , 68 % for shovels and these figures are of serious concern to mine officials and computerized shovel-truck control systems have been considered by some of these mines to improve equipment utilization.

Computer controlled truck dispatching systems, based on stocastic simulation have been developed for the purpose of

(1) improved utilization of equipment

(2) increased productivity and

(3) reduced waiting times

(Baron, 1977, Beaudoin, 1977, Crosson, Tonking and Moffat, 1977, Schlosser , 1955, Naplatanov, et al., 1976, Naplatanov, Sgurev, and Petrov, 1977, Mueller, 1977, Hobday, 1978, Brake, et al. 1979, Hauck, 1979, White, Arnold and Clevenger, 1982, Kim et al., 1981, Wilke, et al., 1981, Clevenger, 1983, Hucka et al., 1985)

2.2. APPLICATION OF THE METHODS

2.2.1. CONVENTIONAL DETERMINISTIC ESTIMATION

2.2.1.1. TRUCK PRODUCTIVITY

Equipment used in earthmoving, quarrying and open-pit mining operations perform regular cycles made up of several phases. For example, the phases for haul unit trucks used in these operations can be listed as follows :

- Loading
- Hauling (Loaded)
- Return (Empty)
- Dumping
- Spotting and maneuvering at loading point
- Spotting and maneuvering at dumping point

The cycle time is the time taken to complete one cycle or one circuit of these phases.

In order to simplify estimating procedure for hauling units, the cycle time is broken down into two fundamental components :

(i) fixed time elements and

(ii) variable time elements.

Figure 2.6 below shows the main components of the cycle time. The fixed time components relate to loading, dumping, spotting and maneuvering and the variable time component relate to hauling and returning as the time to complete these phases depends on the speed of travel and distance of the haul, both of which are job dependent.

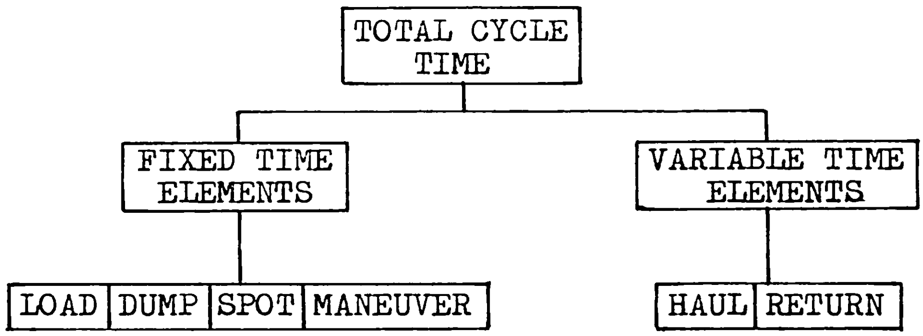

FIG.2.6. COMPONENT ELEMENTS OF TOTAL CYCLE TIME

(i) Fixed Time Components

(a) Loading Time

The loading time is dependent on the type of loading equipment in use, the properties of materials being loaded and the method of loading. The loading time can be obtained from the following expression:

$$TL = \frac{VHP * tc}{BC * 60} \quad ; \text{ min.}$$

Where TL = Loading time, min.

VHP = Truck Body capacity , yd^3

BC = Shovel dipper size , yd^3

tc = Shovel cycle time at 90 degree swing , sec.

(b) Dumping Time

Dumping time is primarily a function of the speed of body tipping and conditions of materials being dumped. The speed of tipping will vary between different types and manufacturers of dump trucks. Approximate figures are given in Table (1) of Appendix 1 assuming a constant tipping time.

(c) Spotting and Maneuvering Times

Spotting and maneuvering times can be considered as the times taken at the loading point and dumping point. These times will depend on the degree of accuracy of spotting required and the requirements for reversing or other maneuvering actions. General figures of the maneuvering and spotting times can be obtained from Tables 2 and 3 in Appendix 1.

(ii) Variable Time Components

The variable times of haul and return can be determined from manufacturers' published performance diagrams. For each hauling unit, a diagram can be used for both loaded and emptied conditions. Maximum attainable speed of the vehicle can be obtained from such a diagram for any particular total resistance (grade resistance plus rolling resistance) and weight of the vehicle.

For haul or return routes on steep downhill grades, it is advisable to use a retarder chart to determine a safe maximum speed that brakes can handle. For haul roads where total resistance is large, rimpull-speed-gradeability curve can be used to determine maximum attainable speed of the vehicle.

Such a typical performance curve (Rimpull-Speed-Gradeability) and retarder chart of a Caterpiller's Cat 777 off-highway truck are shown in figures 2.7 and 2.8.

Situations, where such a performance curve and retarder chart are not available, the following relationship can be used to calculate the maximum attainable speed of the truck:

$$VELM = \frac{HP * 375 * EFF}{RIMPULL}$$

Where VELM = Maximum attainable velocity of the truck, mph

HP = Rated horsepower of truck

EFF = Engine transmission efficiency

RIMPULL = Rimpull required to pull the truck, lbs

= EFFGR * 20 * Vehicle weight in short tons

Where EFFGR = Total effective grade in percent

= RR $\pm$ GR

RR = Rolling resistance of a given road section in percent

GR = Grade resistance of a given road section in percent (positive grade for uphills and negative grade for downhills).

Typical Rolling Resistance values for various road conditions and a range of rolling resistances for a particular road segment are shown in tables 4 and 5 in

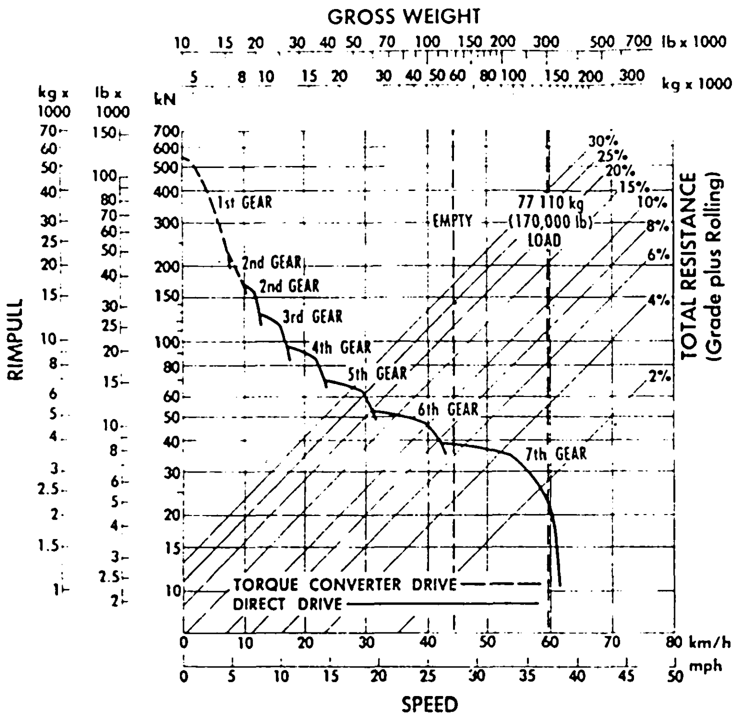

FIG. 2.7 RIMPULL-SPEED-GRADEABILITY CURVE FOR CAT 777 DUMP TRUCK

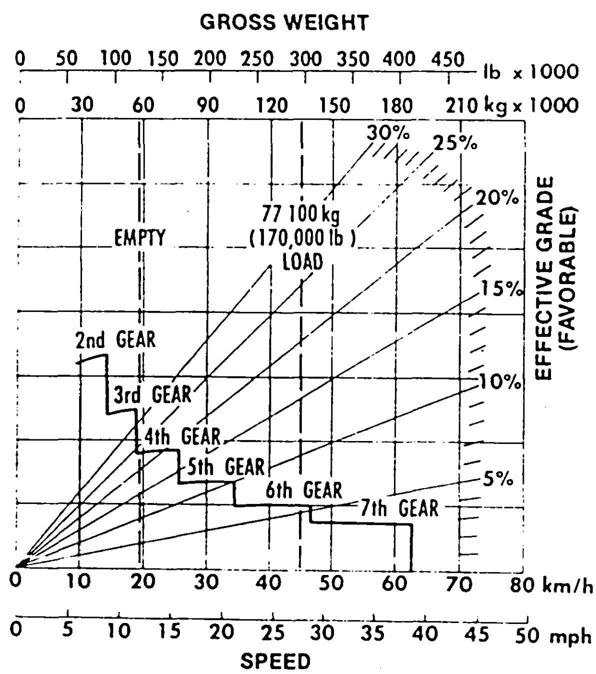

FIG. 2.8 RETARDER CURVE FOR CAT 777 DUMP TRUCK

Appendix 1.

Having obtained the maximum speed of the vehicle, it must be then corrected to give the practical average speed over the haul distance under consideration.

The determination of the practical average speed from the maximum attainable speed of the vehicle is dependent on many factors, some of which can be analytically determined while others for practical reasons must be assessed on an empirical basis.

Flachsenberg (1964) and WatKin use an average overall speed to calculate trip time rather than break the haul into various segments. A refinement to this method was introduced by Fitzpatrick (1967), Bishop (1968) and Morgan (1968). They use a term called 'speed factor' to convert the maximum speed of the vehicle into practical average speed for each segment of the haul profile which has been broken down into various segments depending on the changes in grade or surface conditions.

Fitzpatrick (1967) uses two different speed factors depending on the truck's movement :

(1) Truck travelling from rest

$$\text{Speed Factor (SFR)} = 0.9 - \frac{720}{(D + 1200)}$$

(2) Truck entering road section with momentum

$$\text{Speed Factor (SFM)} = 0.95 - \frac{460}{(D + 1200)}$$

Where D= the length of the haul section, in feet.

Morgan (1968) uses different derating factors to reduce the maximum attainable speed of the vehicle to the average speed. These derating factors depend on such conditions as :

- Length of haul road
- Speed of entry into haul section
- Speed of exit from haul section
- Grade effect on acceleration and inertia
- Power reserve on upgrade working.

Morgan uses the following formulae to convert the maximum speed to an average speed :

$$V = V_{max.} (1 - K)$$

$$K = K_l * K_e * K_f * K_g * K_p$$

$$K_l = \frac{1000}{(L + 1000)}$$

$$K_e = 1 - \frac{V_e}{V_{max}} \left(\frac{L + 500}{8L} \right)$$

$$K_f = 1.1 - \frac{V_f}{10 V_{max}}$$

$$K_g = 1 + \frac{G}{100}$$

$$K_p = \frac{WVW}{350 * BHP}$$

Where V = Average speed of the vehicle over length L, Ft/min

V_{max} = Maximum speed over each segment, Ft/min

K = Speed correction factor

K_1 = Haul length factor

L = Length of haul section in ft.

K_e = Entry speed factor

V_e = Entry speed in mph.

K_f = Exit speed factor

V_f = Exit speed in mph.

K_g = Grade factor

G = Total grade in percent.

K_p = Power reserve factor

WVW = Working vehicle weight in pounds

BHP = Engine brake horsepower

Travel time is then computed from the following expression :

$$\text{Travel time (min)} = \frac{\text{Length of section (feet)}}{\text{Speed factor} * 88 * \text{Max. Speed (mph)}}$$

The travel time is calculated for each segment of the haul road with the truck loaded and returning empty. The total travel time for the entire haul route is the sum of the segment travel times.

The total travel time is then added to the fixed time for loading, dumping, spotting, and manuevering to obtain the total productive cycle time :

Total Productive Cycle Time = Fixed Time + Variable Time

The production rate of a truck can be considered as a function of two variables :

The performance characteristics of the truck and the pit conditions where the truck will be operating.
There exists a large body of literature and publications devoted to the process of estimating the production rates (for example, Peurifoy (1979), Antill (1981), Caterpillar Tractor Co. (various editions), Terex Corporation (1981), and among others).
Generally, the calculations that are carried out to obtain these production estimates are done for plant in isolation; there is no means whereby queues or waiting times which come about through equipment interaction, may be estimated. This tends to give values which underestimate cycle times and hence overestimate production rate.
To arrive at a practical production rate, some allowance for delays, due to management, supervision and labour defficiencies, job conditions, climate, must be made so that the estimates more closely fit job and local site conditions.
Generally speaking, (as mentioned in chap. 2.1.1), in conventional deterministic estimation procedure, delays due to management, supervision and labour defficiencies, job and weather conditions, etc., are considered as a job operational factor.
Extenal delays such as delays caused by repair and maintenance, machine breakdowns, moving equipment, etc., are considered as an availability factor.
The product job operational factor and availability factor

is known as the operating efficiency, and the production rate of a truck is given by :

$$QTH = \frac{60 * VHP * TRUAV * OPJF * DM}{TCT * 2000} \quad ; \quad Tons/hr$$

Where QTH = Hourly truck production, Tons/hr.

VHP = Vessel capacity of a truck, cu.yd.

TRUAV = Availability of a truck

OPJF = Job operational factor

DM = Density of the material, lbs/cu.yd.

TCT = Truck productive cycle time, Min.

Values for TRUAV and OPJF can be obtained from field records by industrial engineering methods and time studies. Where no experience is available to enable TRUAV and OPJF to be determined, their product, known as 'operating efficiency', may used.

$$OPEFT = TRUAV * OPJF$$

Where OPEFT = Operating efficiency of the truck.

The values for OPEFT are given in Table 6 in Appendix 1. Fitzpatrick (1967) mention a more realistic method in which delays are divided into two categories :

(1) Fixed shift delays -- lunch time, shift change, equipment refueling and servicing. This type of delay can be handled quite satisfactorily by using a 50 minute hour or 7 hour shift.

(2) Variable, system delays -- waiting to load or dump.

These delays may be considered as a fixed quantity for each cycle and applied on this basis. The effect of these delays will be more pronounced on short hauls, which more truely reflects operating conditions.

According to his method, total productive cycle can be converted into total truck cycle time as follows :

$$\text{Adjusted truck cycle time} = (CT + D) \frac{60}{(60 - FD)}$$

Where CT = Productive cycle time, min.

D = Cycle delays, min.

FD = Fixed delays, min.

The production rate of a truck is then given by the following expression :

$$QTH = \frac{60 * VHP * DM}{(\text{Adj. Total cycle time}) * 2000} \quad ; \text{Tons/hr}$$

Morgan (1968) classified the factors, which influence the total haul time and production rate, as random and constant factors. Random factors will include :

- Delays on single track haul road while awaiting the passing of a truck in the opposition direction.
- Delays at railway or road crossings.
- Delays due to obstacles or natural features of the haul road.

He also mentions that no mathematical expression can be produced for these effects and it is necessary to consider individually and add to relative section time or slow down time. The actual time to be added will depend on many factors and is best determined on the basis of experience of similar operations.

Constant factors will include the effects of :

(a) Statutory speed restriction

(b) Natural speed restriction due to irregularities in road surface or curvature of road.

(c) Changing rolling resistance due to brake up of the haul road surface and/or weather conditions.

(d) low operator skill and experience.

He stated that the effects of the factors (c) and (d) can be assessed on the basis of experience; however, the effect of speed restrictions (a) and (b) can be closely assessed by use of the speed correction factor.

The other factors to be considered in estimating truck productivity are rolling resistance, grade resistance, and the profile of of haulage road sections .

(i) Rolling Resistance

Rolling resistance is one of the major factors affecting truck performance and consequently a realistic assessment should be made of its value. It is a measure of the force that must be overcome to roll or pull a wheel of a machine over the ground. It tends to stop a truck from moving and it is made up of three components :

- Internal friction of the vehicle
- Tyre flexing and
- Tyre penetration.

Although roll-ing resistance cannot be eliminated, it can be minimized. Internal friction can be controlled by good lubrication and maintenance practices. Tyre flexing can be controlled by avoiding overloads and by maintaining proper tyre inflation pressures and even roads. Well packed roads made with good material reduce tyre penetration. Some typical values of rolling resistance for various roads conditions and a range of rolling resistance values for a particular road segment are given in Tables 4 and 5 in Appendix 1.

(ii) Grade Resistance

Grade resistance is a measure of the force that must be overcome to move a vehicle over unfavourable grades (uphills.) Grade assistance is a measure of the force that assists machine's movement on favourable grades (downhills).Grade resistance is usually expressed as a positive percentage for uphills and negative percentage for downhills.

The combined effect of rolling resistance and grade resistance is known as 'Total Resistance or Effective Grade' and it is a useful for estimating maximum speed of the vehicle when working with Rimpull-Speed-Gradeability Curves and Retarder Curves.

$$EFFGR = RR\ (\%) \pm GR\ (\%)$$

This total effective grade EFFGR can then be converted into total resistance in pounds or rimpull required in pounds by use of the following expression :

$$\text{Rimpull Required} = EFFGR\ (\%) * 20\ \text{lb/ton} * WVW\ \text{in tons}$$

Where WVW = Working vehicle weight in tons.

(iii) Haulage Profile

In predicting the performance of a haulage truck, the optimum haulage profile should be specified, if it is possible. The profile may then be split into sections where a change in characteristics occurs and the average haul speed, and hence the time of each section for a particular truck, may be assessed. Consequently a time for hauling over the entire road may be estimated.

2.2.1.2. SHOVEL PRODUCTIVITY

Generally speaking, the productivity of a power shovel depends on many factors such as :

- The characteristics of the material being loaded
- The working parameters of the shovel
- Nominal shovel cycle time
- Body capacity of a dump truck
- Other operating conditions such as climate, weather, working face, and management and supervision conditions.

Conventionally, the production rate of the shovel can be expressed by assigning values for a few simple variables and making a quick arithmatic calculation.

$$SHOCAP = \frac{3600 * BC * OPEFS * DM}{tc * 2000} \; ; \quad tons/hr$$

Where SHOCAP = Shovel production capacity, tons/hr.

BC = Dipper capacity of a shovel, cu.yd.

OPEFS = Operating efficiency of a shovel (Table 6, Appendix 1

DM = Density of the material, Lbs/cu.yd.

tc = Nominal shovel cycle time, sec (Table 7, Appendix 1)

Norminal shovel cycle times for various shovel dipper sizes and digging conditions are given in table 7 in Appendix 1. Alternatively, the shovel production rate can be estimated by use of the following formula from which the previous formula can be derived. The derivation is shown in Appendix 2.

$$SHOCAP = XMU * TPT \quad \text{in tons/hr.}$$

Where XMU = Loading rate of the shovel,trucks/hr.

TPT = Tons carried per trip per truck, tons.

This formula can only be used when XMU and TPT have already been obtained in an operating shovel-truck system. For preliminary estimation the previous formula should be used.

2.2.2. METHODS APPLYING QUEUEING THEORY

Queueing theory can be applied in open-cut load-haul-dump operations according to the nature of the problem associated. One example is the loading of a fleet of trucks by an excavator as already mentioned in chapter 2.1.2 and figure 2.3. Another example with a different nature is the problem associated with the repair and maintenance of construction and mining plant or machinery. As machinery or equipment breaks down, generally at irregular intervals, the machinery is serviced by a repair crew(s) or repairman or operator in order to restore the machinery to its original running order. The service also takes irregular lengths of time to perform. This type of queueing system is shown schematically in figure 2.9 below.

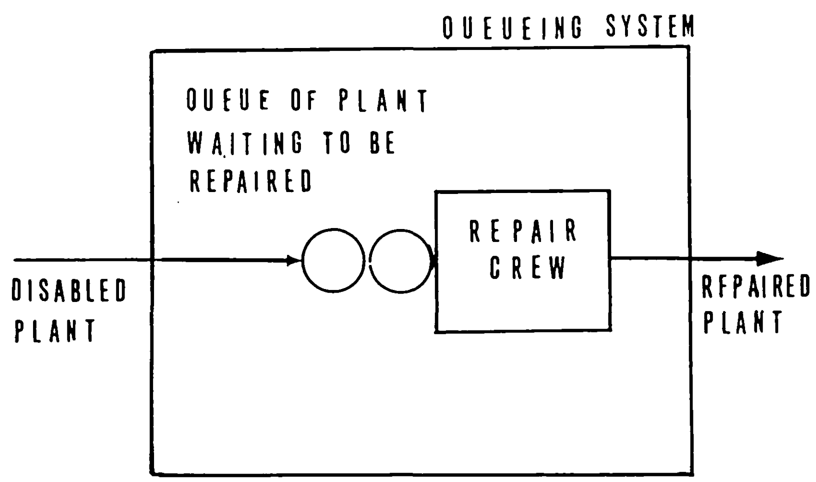

FIG. 2.9. AN EXAMPLE OF QUEUEING SYSTEM (REPAIRMAN MODEL) (After Carmichael, 1987, Engineering Queues in Construction and Mining).

Generally speaking, a queueing situation involves units arriving for service, waiting or queueing for service or being served if there is no waiting line, and then units leaving the system after being served.

The study of such queueing situations is of relevance to engineering management as decisions have to be made as to the type and size of service capacity required as well as to the demand itself. For example in the truck-excavator case, management decisions are required related to the size and number of trucks, the size and number of excavator, haul road length and surface conditions, etc. For the machine repair case, engineering management decisions are required related to the type and number of machines, number of repair crews, regular maintenance versus repair-on-breakdown philosophies, etc. There is some optimum solution for every problem and this generally involves a trade-off between providing extra service capacity at cost and providing less service capacity. The latter is also at cost as there is a cost associated with units waiting for service.

In applying queuing theory to a materials handling problems, the system under investigation may be characterised according to :

(a) the system input source

(b) the queue characteristics

(c) the service discipline

(d) the service mechanism.

This is illustrated in figure 2.10, where units (customers) flow through the queueing process.

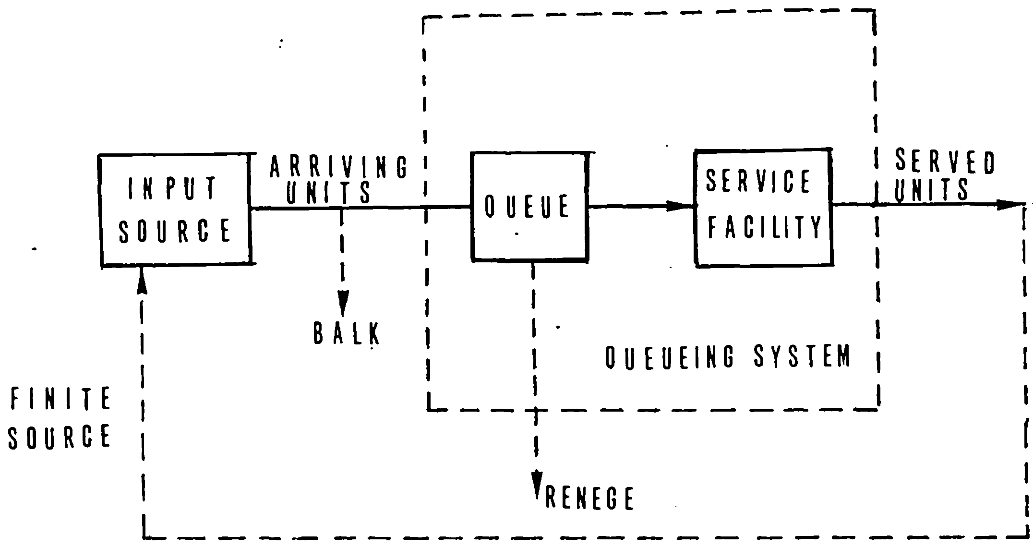

FIG.2.10. FUNDAMENTAL QUEUEING PROCESS (After Carmichael,1987, Engineering Queues in Construction and Mining).

(a) Input source

The input source of the queueing system can be characterized by means of (i) customer population and (ii) arrival pattern of the units.

(i) Customer Population

Units entering the system derive from some calling population which is taken to be either finite or infinite. Customer population represents the units that will at some time require service. In an earthmoving operation, as an example, the fleet of trucks will be finite in number and hence the customer population will be finite.

(ii) Arrival Pattern

The input source is also characterized by its statistical qualities, either in terms of a probability distribution or statistical moments such as expected value and variance. The arrival pattern is the manner in which the trucks arrive and become a part of the queue. In the majority of cases in load-haul operations the problem of a number of dumptrucks requiring service by a loading machine can be considered as being of the random distribution type. The behaviour of random arrivals can be specified by the Poisson Distribution :

$$P_n(t) = \frac{(\lambda t)^n e^{-\lambda t}}{n!}$$

Where t = Arrival times of units

n = Number of customers (trucks) in the process

λ = Mean arrival rate of the units

$P_n(t)$ = Probability of n customers (trucks) arriving during time t.

Equivalently this may be rephrased to state that the time between unit arrivals (interarrival time) follows an exponential distribution.

(b) Queue Characteristics

The queue of units waiting to be serviced is taken as either finite (restricted) or infinite (unrestricted). The former occurs where there is insufficient space for an unlimited length queue to form. In such a case, units arriving when the queue has reached its maximum length are turned away. The occurance of such finite queues is not common in construction and mining operations, there generally being enough space provided for any anticipated queue size.

(c) Service Discipline or Queue Discipline

The service or queue discipline refers to the manner in which units or customers are serviced or processed. Typically this is on a first-come, first-serve (FCFS) basis or less often on a last-come, first-serve (LCFS) basis. Equivalently these two service disciplines may be described as FIFO (first-in, first-out) and LIFO (last-in,first-out) respectively. The case of a fleet of trucks being loaded by an excavator working at the mine face is usually according to an FCFS discipline and no priorities exist.

(d) Service Mechanism

Service mechanism can be expressed by means of two service characteristics : (i) service facility and
(ii) service pattern.

(i) Service Facility

The manner of servicing the units may be characterized according to the number of servers. In cases, with simple situations where the fixed number of trucks are allocated to a shovel, the system is considered a closed-loop system, and hence is called single-channel (server) model. However, there are also multi-channel (server) models in which shovels working in different working faces or levels performing the same service.

Other queue types include cyclic and network queues. Cyclic and finite queues are the most common queues used in the earthmoving operation involving loading, hauling, dumping and returning per cycle. Both queue interpretations lead to the same result (Carmichael, 1987).

(ii) Service Pattern

The service mechanism is also characterized by the service pattern. That is, by its statistical qualities, in a similar fashion to the input source. As with the input source, the commonly adopted distribution is the exponential distribution describing the service times. The Erlang and constant distributions, among others, are also adopted.

2.2.2.1. QUEUEING SYSTEM INFORMATION

Having applied a queuing model to a given engineering operation, what information would be of use as a decision or management tool ? Generally, the information of interest to the engineer is that principally related to :

- Waiting times for units or customers in the system and in the queue
- Queue length, number of customers in the system

- Server idle time
- Customer waiting time in the system
- Number of idle servers
- Productivity, production or output of the operation.

The term "Measures of Effectiveness" is commonly used to describe some of this system information.

The optimal performance of the operation will relate to the costs associated with the above quantities. Generally, some trade-off or balancing of costs would need to be carried out in order to determine results of the operation.

2.2.3. COMPUTER SIMULATION TECHNIQUE

The activities of materials loading and transporting lends itself to constant change. A means must be provided to indicate the effects of these changes. It is not possible to perform this by straight forward empirical analysis, and in large open pit mine the truck haulage system becomes so complex that quantitative results are difficult to obtain analytically from queueing theory. Computer simulation is probably the only practical method for predicting the performance of a truck haulage system.

Computer simulation and in particular, that termed Monte Carlo simulation, can handle the overall effect of the interactions and interrelationships of the variables which are an indigenous part of the real system configuration, and which all too often, cannot be estimated in any other way.

Monte Carlo Simulation is basically a probabilistic method and employs a random sampling technique. There are two unique features of this method, viz.

(i) A distribution of the data must be known. If it is not known then a distribution must be assumed.

(ii) Simulation can be conducted by creating an artificial sample of the population that is required to be investigated.

In its strict form, Monte Carlo Analysis is used to evaluate a deterministic problem by converting it to a probability model. The random sampling technique is really used to determine the solution to an analytical or deterministic problem.

In a truck haulage system the Monte Carlo Simulation is used where significant variation in the system is likely to occur due to random changes in the input variables These input variables include loading time, dumping time, travel time (haul and return) , delay times,load weights. Each variable would have a separate distribution defining it.

There are two possible methods of assigning values to the variables in a random analysis. One uses a random number to fix the value of a random variable through the use of a probability distribution function and the other uses a cumulative relative frequency polygon plot of the particular element (Chatterjee, 1970).

If the equipment is in already in operation and working under similar conditions to that being modelled, then time studies will give all the values required. It will also give an indication of the best distribution to use or allow the use of empirical distributions. In the case of feasibility study or where the effect of replacement equipment on an existing operation is being studied, manufacturers' figures or some sort of estimation would be used.

2.2.3.1 DATA REQUIRED FOR MONTE CARLO SIMULATION

Production and operating data needed for simulation study are as follows:

(a) Haulage road profiles and characteristics, i.e. distances, grades, rolling resistance, speed limits, right-of-way rules.

(b) Equipment characteristics and availabilities, i.e. speed-rimpull curves, motor-current curves, mechanical availabilities, empty vehicle weights.

(c) Field observations of shovel's loading time, truck's dumping times, and payload weights.

(d) Pit configuration, equipment configuration and associated production during the time simulated.

The next step is to analyse the collected data for their respective distribution.

(1) Loading Time

The loading time observation starts when an empty truck begins backing up to the shovel and ends when the same truck starts on its way to an unloading point. A shovel loads a given size truck in a certain time which is stochastic.

For estimation purposes, O' Neil and Manula (1967) use a normal distribution while Deshmukh (1970) and Teicholz (1963) prefer a lognormal one. The lognormal view is further strengthened by Mutmansky (1970) as well as by Kim and Ibarra (1981).

(2) Truck Dumping Time

Similarly, the dumping is also stochastic and its observation starts when a loaded truck begins backing up to the dumping point and ends when the empty starts the return trip to a shovel.

For estimation purposes, there are a number of different opinions. O' Neil and Manula as well as Mutmansky use an exponential distribution while Morgan and Peterson (1968) use a normal distribution. On the other hand, the lognormal view is supported by Douglas (1963), Deshmukh (1970), and Kim and Ibarra (1981).

(3) Travel Time

The time required to complete the haul (loaded) and return (empty) will depend on the distances, grades, rolling resistances and equipment performance character. This phase is also stochastic, but if the delays are isolated from it, the time required to complete the phase will be essentially constant for every cycle, and thus can be treated as deterministic.

This view is supported by most authors without added delays in the travel time (for example O' Niel and Manula (1967), Gibbs, Gross and Pflieder (1967), Chatterjee and Hellewell (1971), Manula Mohibatsela and Ramani (1980), Kim and Ibarra (1981) and Tu and Hucka (1985).

The travel time is treated as deterministic process and is determined by using the manufacturer's equipment performance characteristic curves in their programs.

However, where the performance curves are not available, an approximate formula can be used which utilizes a relationship of engine horsepower and maximum speed to develop rimpull. This is already stated in chapter 2.2.1, and need not be detailed here.

(4) Delay time

During the load-haul- and dump phases, two types of delay can occur, namely induced and external.
Induced delays are the waiting period before loading and dumping and the travelling time behind a slower truck.
External delays consist of lubrication of trucks, inefficiency of operators, and adverse weather conditions.
Both kinds of delays are stochastic and those authors that advocate this approach - Douglas (1963), Deshmukh (1970) and Mitmansky (1970) - all agree that for estimate purpose, it should be treated as lognormal.

(5) Repair Time.

Besides the stochastic performance of shovel and truck, unscheduled down time of shovel and trucks had to be taken into account, when the realty of an openpit operation is aimed at. Tu and Hucka (1985) use an exponential distributions both for shovel and truck repair times.

(6) Load Weights

This would seem logically to follow a normal distribution and this in fact is the only part of the system that all the authors can or will admit to agreement on.

2.2.3.2. SELECTING THE APPROPRIATE DISTRIBUTION

Having obtained the required data, the next step is to select the appropriate distribution to represent the system and subsystem under study. To do this, it is necessary to plot the sample data to see if the resulting distribution looks like a known distribution function. If one or more known distributions look like they would do an adequate job of representing the distribution of the data, then statistical tests should be applied to the data to determine which distribution would be the most appropriate to use. One could also plot the sample cumulative distribution before making any statistical tests. Of course, the results of any statistical test are based on the basic assumptions that the sample data used in the test are representative of the population from which they came and that the sample size is sufficiently large enough to indicate such.

Chi-square Goodness-of-Fit Test

This test is appropriate for testing the hypothesis that a given set of data came from a certain distribution with all parameters specified. The general procedure for Chi-square goodness-of-fit test is as follows :

(a) Hypothesize that the sample data came from a certain distribution with all parameters specified or estimated.

(b) Divide the range of the hypothesized distribution into 'm' subintervals such that the expected number of values, E_j, in each subinterval is at least five for j = 1,2,3 ____ m.

(c) Let O_j be the number of sample data points in the j^{th} subinterval for j = 1,2,3 ____ m.

(d) The quantity

$$\text{calculated } \chi^2 = \sum_{j=1}^{m} \frac{(O_j - E_j)^2}{E_j}$$

approaches the distribution with (m - k - 1) degrees of freedom as 'm' becomes large, where k is the number of parameters that must be estimated from the sample data in order to calculate the expected number of values, E_j for j = 1,2, ____ m .

(e) If calculated χ^2 value is greater than the critical value that might be expected under the null hypothesis assumption, using (m - k - 1) degree of freedom and a 100(1- α) percent significant level, reject the hypothesis that the sample data came from the hypothesized distribution; other wise, consider some other distributions before accepting the hypothesized distribution as being adequate. Alternatively speaking, the null hypothesis is rejected if calculated χ^2 value falls in the rejection region of the sampling distribution as shown in figure 2.11 below.

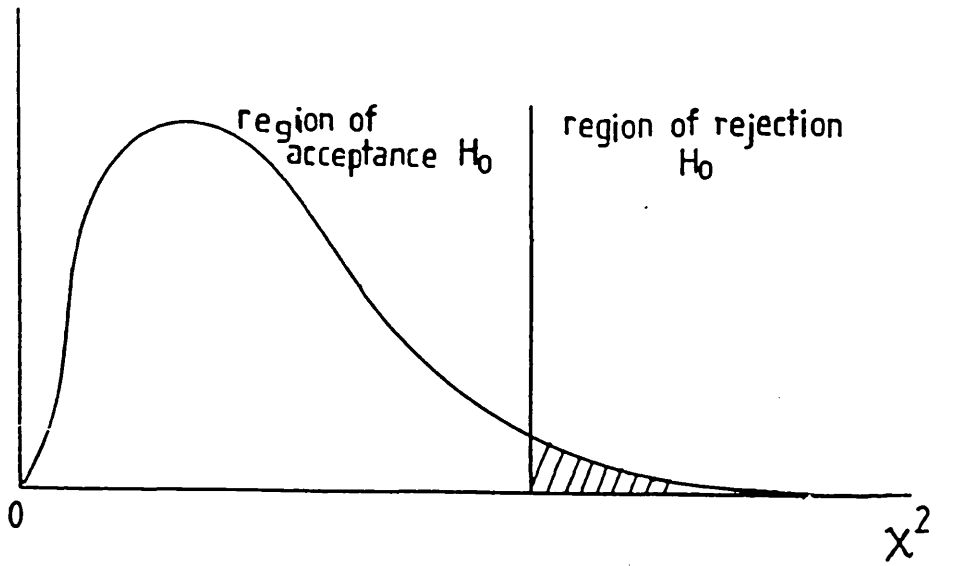

FIG. 2.11 CHI SQUARE DISTRIBUTION

In this study, distributions for all stochastic variables have been assumed to have some form of distribution . General discussion has been made in the previous chapter (2.2.3.1) and the assumed distributions for particular type of stochastic variable will be dealt in chapter 4.3.1 .

3.0. DETERMINATION OF OPTIMUM SHOVEL TRUCK COMBINATION

The determination of the optimum shovel-truck combination mainly depends on the cost criterion of the shovel-truck fleet. It is necessary to establish the total costs per ton of material being moved. To establish the total costs of hauling material, the cost of trucking operation cannot be considered in isolation. It is obvious that, as the performance of the truck fleet is dependent on the method of loading, the optimum overall costs are a function of the economic relationship between the loading and hauling equipment. It is now clear to carry out the economic comparisons of the various combinations of shovel-truck fleet so as to obtain the optimum combination of shovel and truck as well as the size of the fleet.

3.1. METHODS FOR ECONOMIC COMPARISONS

The method of economic analysis chosen will depend on the type of operation under consideration. If the operation is such that all equipment will be kept in service for its full service life, a system expressing the total costs in terms of an annual average can be used. Where the study is made to determine a replacement policy for equipment, the analysis must take into account the time shape of expenditure (i.e. low maintenance cost in the early years which increase with hours of operation).

Economic decisions with revenue and expenses spread over a span of time are evaluated by comparing the time value of cash flow. The most commonly used methods in comparing the alternatives are as follows :

(1) Equivalent Annual-Cost Method.

(2) Present-Worth Method.

(3) Discounted Cash Flow (DCF) Yield or Internal Rate of Return Method.

(4) Net Present Value Method.

The first two methods compare costs on the basis of a stipulated minimum acceptable return on invested capital. The alternative with the lowest "equivalent annual expense" or lowest "equivalent present cost" is the economic choice. However, these two methods are related to schemes whereby only the expenditure is considered and are confined to determining whether one course of action is more economic than another course of action.

The Internal Rate of Return or Discounted Cash Flow Method compares alternatives on the basis of the percentage return on increments of investment. The amount then an alternative exceeds a minimum standard return within available investment sources is the criterion for selection.

Net Present Value Method is used to determine whether a proposed project or alternative yields at least the minimum return specified by the company. If the N.P.V. is positive it follows that the yield is above minimum and the project or alternative is worthy of further consideration. If the N.P.V. is negative then the yield is less than the minimum and the project or alternative can be rejected without further analysis. For general use in the off-highway haulage, the discounted cash flow and net present value methods would prove unsuitable except for large projects and contractors, on account of initial difficulties with familiarization and application. For the majority load-haul operations, equivalent annual cost method would give sufficiently accurate results.

3.2. EQUIVALENT ANNUAL COST METHOD

With the equivalent annual cost method, it is practicable to separate the cost figures into the cost of investment on the equipment and annual running or operating cost. The equivalent annual cost of capital recovery can be expressrd by the following equation :

$$ACCR = (P - S) * \left[\frac{i(1 + i)^n}{(1 + i)^n - 1} \right] + Si + OC$$

Where ACCR = Annual Cost Of Capital Recovery

P = installed cost or purchase price of equipment

S = Expected net salvage value of equipment after 'n' years.

n = Service life of equipment.

i = Interest rate on investment.

OC = Annual running or operating costs.

The expression $[\frac{i(1+i)^n}{(1+i)^n-1}]$ is termed 'the capital recovery factor' and it takes into account the following factors :

- Service life of facility
- Salvage value of proposed facility
- Depreciation
- Installed cost
- Investment expenses (included in P)
- Interest rate on Capital.

The operating costs are those incurred while the equipment is operating and include the following :

- Tyre costs
- Fuel costs
- Costs for lubricating oil, hydraulic oils, etc.
- Repair and Maintenance Costs
- Operator costs.

The procedure for estimating operating costs items is given in Appendix 3.

3.3. TAX CONSIDERATIONS

Taxes are a major factor in any profit seeking venture. They affect net returns for both individuals and corporations. In many situations, before-tax analyses provide adequate solutions. When the alternatives being compared are to satisfy a required function and are affected identically by taxes, the before-tax comparison yields the proper preference. Evaluation of public projects rarely include tax effects and are conducted as before-tax analysis.

The main objection to the use of a before-tax rate of return requirement is that the method makes no allowance for the rate of write-off (Gerald, 1973). The other additional objections given by the same author to the before-tax approach include the following :

(1) Because the before-tax approach makes no allowance for the rate of write-off, it fails to provide any guidelines for choice of a depreciation method nor does it give a proper basis for the accept-reject decision where liberalized depreciation is applied.

(2) Even with straight-line depreciation the approach is inexact. Its use results in understatement of prospective rate of return or overstatement of revenue requirements for properties which are depreciable.

(3) In allocating funds to projects (capital budgeting) the after-tax approach may be the only alternative when comparing projects subject to depletion against those subject to depreciation. In any case, the ranking of projects may be different on a before-tax basis than on an after-tax basis.

(4) As usually treated in practice, the before-tax approach does not take into account the firm's financial structure.

For these reasons (objections), it is wise to perform economic analysis on an after-tax basis.

There are numerous kinds of taxes and the most commonly known taxes are : (a) Income Taxes

(b) Property Taxes

(c) Sales Taxes and

(d) Excise Taxes.

(a) Income Taxes

Income taxes are usually the only significant taxes to be considered in an economic analysis. They are assessed as a function of net income or profit minus certain allowable deductions and exemptions.

(b) Property Taxes

They are assessed as a function of the 'value' of real estate, business and personal property. Hence, they are independent of the income or profit of an individual or business. They are normally treated as annual disbursements.

(c) Sales Taxes

They are assessed as a function of purchase of goods and/or services, and are thus independent of the net income or profits.

(d) Excise Taxes

These taxes are federal taxes assessed as a function of the scale of certain goods or services often considered "nonnecessities" and hence are independent of the income or profit of an individual or business. While they are usually charged to the manufacturer or original provider of the goods or service, the cost is passed on to the consumer.

It is now obvious that the income taxes are the only significant taxes to be considered in an economic evaluation. They are levied by the federal, most state, and occasionally municipal, government, and these regulations are extremely complex and changed rather frequently, it is not intended to be a comprehensive treatment of income taxes, in this chapter. However, it will deal with certain basic concepts in considering the after-tax economic analysis.

3.3.1. GENERAL PROCEDURE FOR MAKING AFTER-TAX ECONOMIC ANALYSIS

A tabular approach is convenient for modifying the before-tax cash flow to show the effects of taxes. The number of entries in the table depends on the number of tax considerations involved: the most common are depreciation and interest deductions. Table headings based on these tax effects are shown below :

TABLE 3. <u>TABULAR FORMAT FOR DETERMINING AFTER-TAX CASH FLOW</u>

1	2	3	4	5	6	7
End of Year	Before-Tax Cash-Flow	Deprec.	Loan and Interest	Taxable Income (2-3-4)	Taxes (5*Tax Rate)	After-Tax Cash-flow (2-6)

Several types of depreciation methods can be used to determine the amount to be depreciated and such methods are described in Appendix 6 .

After-tax comparison of proposals can be made using any of the comparison methods : - Equivalent Annual Cost Method

- Present Worth Method
- Discounted Cash Flow Yield Method
- Net Present Value Method.

Once the tax effects on cash flows have been determined, the computational procedure and the interpretation of results are the same as in before-tax analysis.

In this study, before-tax economic analysis have been carried out for various shovel-truck combinations. This is because of that the income taxes often do not have any effect on the decision that would be made in selecting the proper shovel-truck combination since the combinations being compared would be affected identically by taxes if the after-tax analysis would have been carried out. The decision would be the same as in before — tax analysis since all shovel-truck combinations would be treated by the same depreciation method and tax rate.

4.0. PROBLEM APPROACH

Three possible sizes of electric power shovels ranging from 8 cu.yd to 12.5 cu.yd dipper capacities and three different sizes of mechanical drive rear dump trucks from 85 short ton to 120 short tons are considered throughout the study. To be more specific shovel sizes of 8, 10, 12.5 cu.yd and dump trucks sizes of 85, 100 and 120 short tons are considered. Three computer program models, relating to each method, are developed and these program models are discussed in detail in this chapter.

4.1. CONVENTIONAL DETERMINISTIC ESTIMATING PROCEDURE

A computer program called Program HAULCO has been developed and the assumptions used in formulating this program are given in appendix 4.

4.1.1. PROGRAM HAULCO

This program is the main program for the method and it's main objective is to determine the optimum combination of the shovel-truck fleet size. To be more specific, the program is designed to estimate the total cycle times of the trucks, shift performance, truck fleet size for each shovel, number of shovels required, productivity of the combined shovel-truck fleet and finally the cost per ton of the truck-fleet and shovel and total costs per ton of the combined shovel truck-fleet. A desired production rate of 6000 tons/shift is assumed in this program.

Travel time estimations (haul and return) are first computed by using the empirical formula (see chapter 2.2), which utilizes the relationship of engine horsepower of the vehicle and maximum speed to develop required rimpull.

Then shift performance per truck, number of trucks used, and fleet productivity are computed by use of the formulae given in Appendix 2.

Finally, the costs per ton for each shovel-truck combination are computed on the basis of capital recovery factor.The interest rate is assumed at 15 % , the useful life of a truck is 8 years, and a shovel, 20 years.

Operating and maintenance costs are estimated according to Appendix 3. Figures 4.1 and 4.2 show the program Flowchart and a sample output of the program.

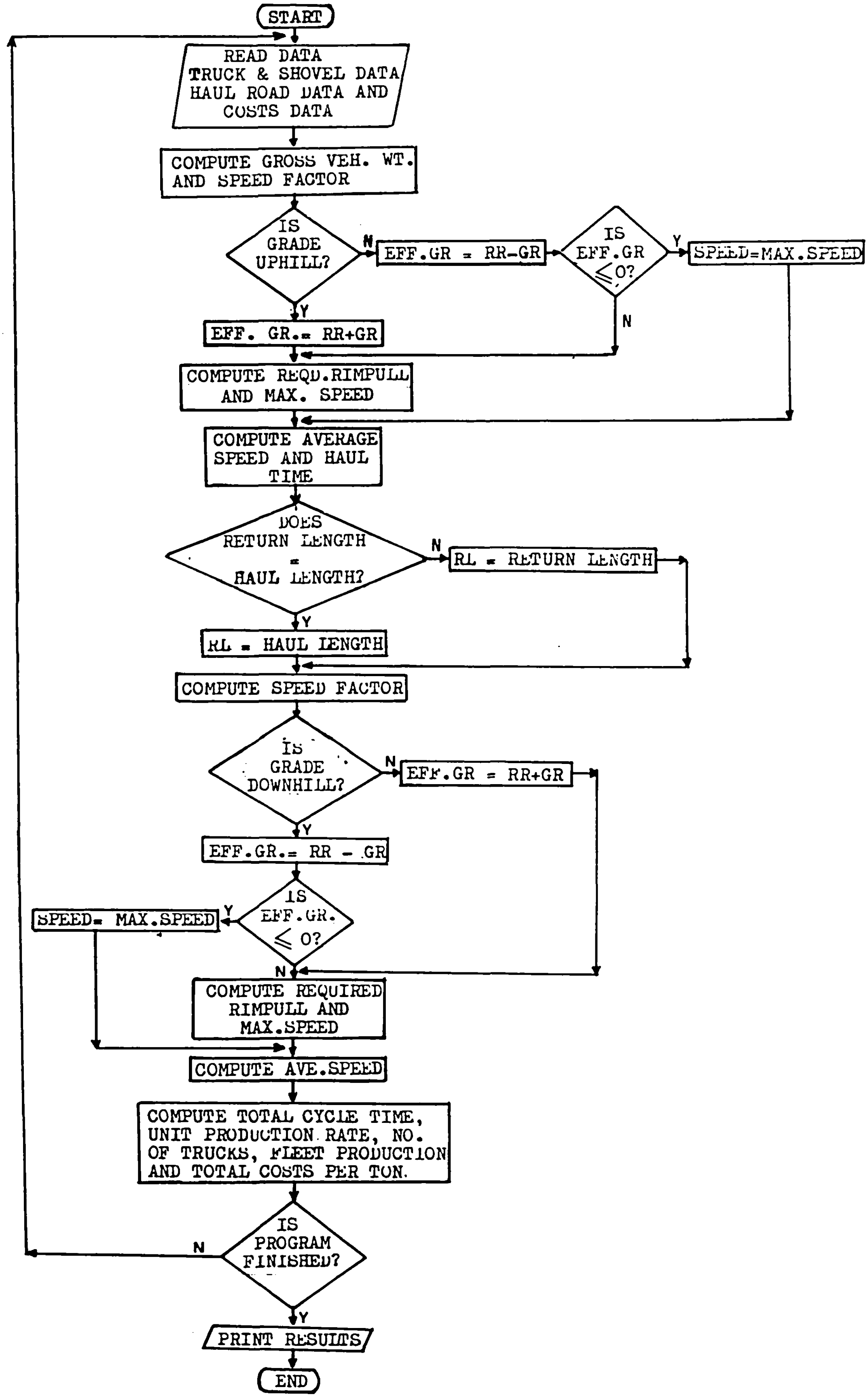

FIG. 4.1 FLOW CHART FOR PROGRAM HAULCO

INPUT DATA

TRUCK PAYLOAD	=	85.00 TONS
NET VEHICLE WEIGHT	=	60.40 TONS
GROSS VEHICLE WEIGHT	=	145.40 TONS
HORSE POWER	=	880.00
TRANSMISSION EFF.	=	.87
TRUCK VOLUME	=	64.80 CU.YD.
HAUL DISTANCE	=	14600.00 FT
AVERAGE GRADE	=	5.00 PERCENT
ROLLING RESISTANCE	=	3.00 PERCENT
MAX. SPEED (LOADED)	=	12.34 MPH
MAX. SPEED (EMPTY)	=	18.50 MPH
SPEED FACTOR (LOADED)	=	.92
SPEED FACTOR (RETURN)	=	.92
TRUCK EFFICIENCY	=	.87
SHOVEL EFFICIENCY	=	.87
MATERIAL DENSITY	=	2700.00 LBS/CU.YD
PRODUCTION DESIRED	=	6000.00 TONS/SHIFT
SHIFT DURATION	=	8.00 HOURS

RESULTS

NUMBER OF PASSES AND TOTAL CYCLE TIMES

DIPPER SIZE CU.YD.	NO.OF PASSES	LOADING TIME MIN	TOTAL HAUL TIME MIN	TOTAL RETURN TIME MIN	TOTAL FIXED TIME MIN	TOTAL CYCLE TIME MIN
8.00	8	4.90	19.20	13.47	1.50	39.08
10.00	6	3.74	19.20	13.47	1.50	37.91
12.50	5	3.26	19.20	13.47	1.50	37.43

SHIFT PERFORMANCE PER TRUCK AND NO. OF TRUCKS USED

DIPPER SIZE CU.YD.	TONS/SHIFT TONS	TRIPS/SHIFT TRIPS	TONS/TRIP TONS	NO.OF TRUCKS
8.00	1061.28	12.28	86.40	6
10.00	1025.62	12.66	81.00	6
12.50	1082.07	12.82	84.38	6

FIG. 4.2 SAMPLE OUTPUT OF PROGRAM HAULCO

AVERAGE ARRIVAL AND LOADING RATES AND NO.OF SHOVELS

DIPPER SIZE CU.YD.	AVE.ARRIVAL RATE TRUCKS/HR	AVE.LOADING RATE TRUCKS/HR	NO.OF SHOVELS
8.00	9.21	12.23	1
10.00	9.50	16.06	1
12.50	9.62	18.42	1

FLEET PRODUCTION

DIPPER SIZE CU.YD.	FLEET PRODUCTION T/S	SHOVEL CAPACITY T/S	TONS MOVED T
8.00	6367.65	8456.40	6367.65
10.00	6153.70	10407.88	6153.70
12.50	6492.12	12435.88	6492.12

COSTS PER TON

DIPPER SIZE CU.YD.	COST PER TON TRUCK $/TON	COST PER TON SHOVEL $/TON	TOTAL COST PER TON $/TON
8.00	.903	.213	1.116
10.00	.934	.276	1.211
12.50	.886	.322	1.208

CALCULATION COMPLETED

FIG. 4.2. CONTINUED

4.1.2 RESULTS FROM PROGRAM HAULCO

After completing the computations for all trucks over a given haul road, a summary of travel times (haul and return) is shown in table 4.1. These travel times are exclusive of other fixed time elements.

Table 4.1 Summary of Truck Travel Times by sizes.

Truck Size	Travel Time (Min.)		
(T)	HAUL	RETURN	TOTAL
85	19.20	13.47	32.67
100	20.14	13.47	33.61
120	22.32	13.47	35.79

The return times of all trucks appear to be the same amount, since the return speed for all trucks is, limited at 18.5 mph for down haul.

A typical relationship between the production capacities and the number of trucks is shown in figure 4.3. The figure shows that 5.6 number of trucks are required to fulfill the desired production of 750 t/hr. However, the number of trucks could not be the real number and therefore the next higher integer value is chosen and thus becomes 6.

Having obtained the number of trucks to be used, fleet production of the truck-fleet and shovel is then computed for each truck and various shovel size combinations. And then costs data for trucks and shovel are read in and costs per ton values for each truck and corresponding shovel combination are computed. Table 4.2 shows a summary of haul

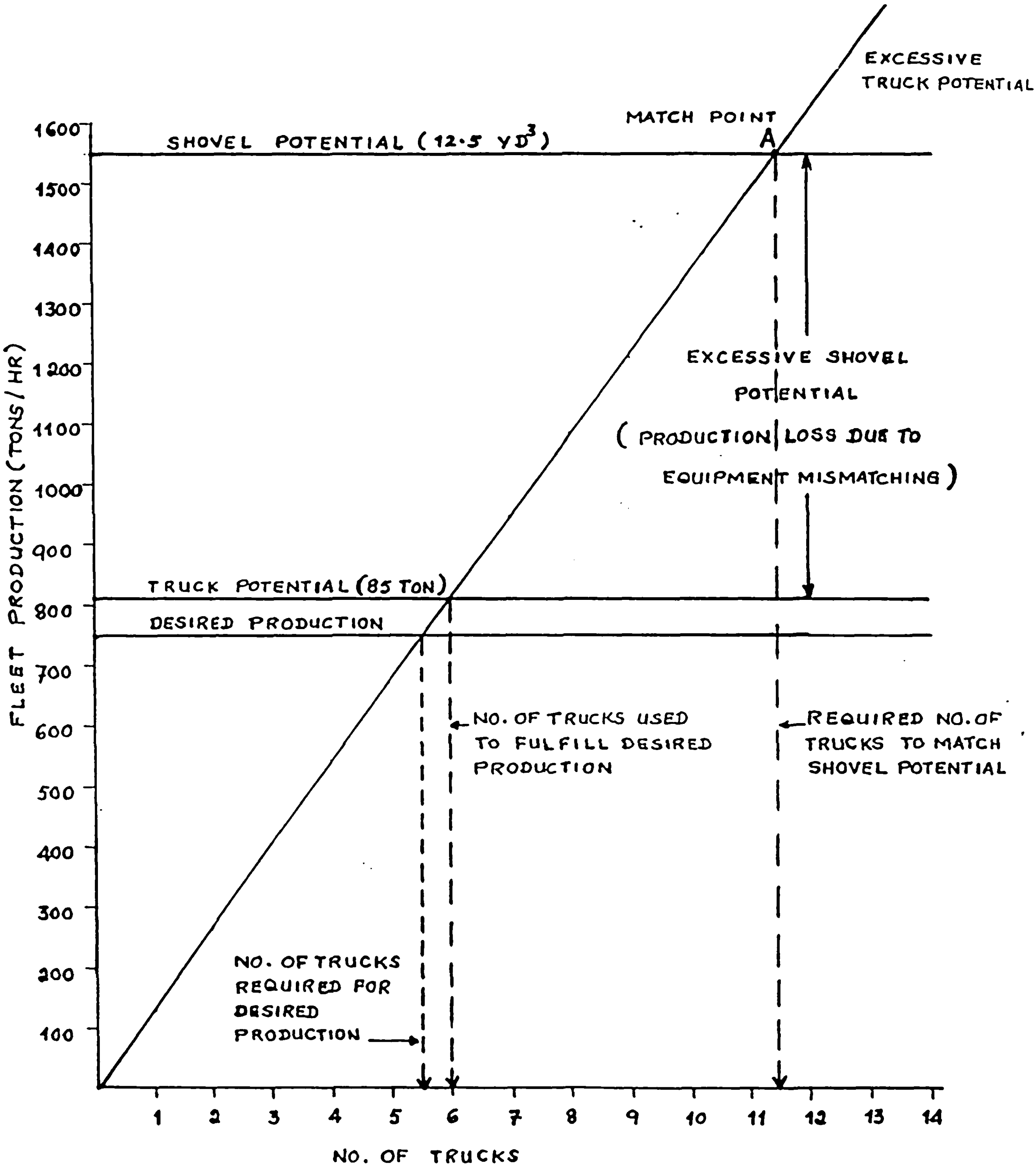

FIG. 4.3 PRODUCTION CAPACITIES VS. NO. OF TRUCKS

performance of 85 ton truck and its coresponding shovel size combination.

TABLE 4.2. HAUL PERFORMANCE SUMMARY FOR 85 TON TRUCK

TRUCK SIZE =85 TON HAUL LENGTH = 14600 FT

DIPPER SIZE (CU.YD.)	8.0	10.0	12.5
Number of Passes	8	6	5
Total cycle time (min.)	39.08	37.91	37.4
Productivity			
Tons/ Shift/ Truck	1061.28	1025.62	1082.02
Trips/ Shift/Truck	12.28	12.66	12.82
Tons/ Trip/Truck	86.40	81.00	84.38
Number of trucks used	6	6	6
Ave. arrival rate (truck/hr)	9.21	9.50	9.62
Ave. loading rate (truck/hr)	12.23	16.06	18.42
No. of shovel used	1	1	1
Truck fleet production (Tons/ shift)	6367.65	6153.70	6492.12
Shovel capability (Tons /shift)	8456.40	10407.88	12435.88
Tons moved (tons shift)	6367.65	6153.70	6492.12
Costs:			
Truck costs ($/ton)	0.90	0.93	0.89
Shovel costs($/ton)	0.21	0.28	0.32
Total costs ($/ton)	1.12	1.21	1.21

It is observed that the lowest truck cost per ton is obtained with 12.5 cu.yd. shovel at $ 0.89 per ton, while the lowest shovel cost is obtained with an 8 cu.yd. shovel at $ 0.21 per ton . Overall total costs per ton is obtained with 85 ton truck with 8 cu.yd. shovel at $ 1.12 per ton. Following the computation of each shovel-truck combination for a given set of operating conditions, the summaries are shown in tables 4.3 and 4.4.

TABLE 4.3. HAUL PERFORMANCE SUMMARY FOR 100 TON TRUCK

TRUCK SIZE = 100 TON HAUL LENGTH = 14600 FT.

Dipper size (cu. yd.)	8.0	10.0	12.5
Number of passes	9	7	6
Total cycle time (min.)	40.63	39.47	39.02
Productivity			
Tons/shift/truck	1148.39	1149.28	1245.58
Trips/shift/truck	11.81	12.16	12.30
Tons/trip/truck	97.20	94.50	101.25
Number of trucks used	6	6	5
Av. Arrival rate (trucks/hr.)	8.86	9.12	7.69
Av. loading rate (trucks/hr.)	10.88	13.77	15.35
Number of shovels used	1	1	1
Truck fleet production (Tons/shift)	6890.36	6895.68	6227.90
Shovel capability (Tons shift)	8456.40	10407.88	12435.88
Tons moved (Tons shift)	6890.36	6895.68	6227.90
Costs			
Truck costs ($/ton)	1.02	1.02	0.94
Shovel costs ($/ton)	0.20	0.25	0.34
Total costs ($/ton)	1.22	1.27	1.28

TABLE 4.4. HAUL PERFORMANCE SUMMARY FOR 120 TON TRUCK

TRUCK SIZE = 120 TON HAUL LENGTH = 14600 FT.

Dipper size (cu.yd.)	8.0	10.0	12.5
Number of passes	11	9	7
Total cycle time (min.)	44.04	44.90	41.86
Productivity			
Tons/ shift/ truck	1294.77	1359.38	1354.58
Trips/ shift/truck	10.90	11.19	11.47
Tons/trip/truck	118.80	121.50	118.13
Number of trucks used	5	5	5
Ave.arrival rate (trucks/hr)	6.81	6.99	7.17
Number of shovel used	1	1	1
Truck fleet production (Tons / shift)	6473.85	6796.89	6772.92
Shovel capability (Tons/shift)	8456.40	10407.88	12435.88
Tons moved (tons/ shift)	6473.85	6796.89	6772.92
Costs			
Truck costs ($/Ton)	1.03	0.98	0.98
Shovel costs ($/Ton)	0.21	0.25	0.31
Total costs ($/Ton)	1.24	1.23	1.29

A summary of total costs per ton for each shovel-truck combination is listed in the following table 4.5.

Table 4.5. SUMMARY OF TOTAL COSTS PER TON ($/TON) FOR 6000T/SHIFT

TRUCK SIZE (TON)	SHOVEL SIZE (CU.YD.) 8.00	10.00	12.50
85	1.12	1.21	1.21
100	1.22	1.27	1.28
120	1.24	1.23	1.29

It is observed that the lowest costs per ton for 85 ton truck with its corresponding shovel combination appears to be with 8 cu. yd shovel at $ 1.12 per ton, for 100 ton truck with 8 cu. yd. shovel at $ 1.22 per ton, for 120 ton truck with 10 cu. yd. shovel at $ 1.23 per ton respectively. The best performance shovel-truck combination is found to be with 85 ton truck with 8 cu.yd. shovel at the cost of $ 1.12 per ton. It is also observed that the total costs per ton for 85 ton truck with its corresponding shovel combination appear to be less than that of the other two trucks. This is because of the faster total cycle time (i.e. faster loading time due the lesser number of passes and faster travel times) and limited production rate desired which is one of the factors affecting the total cost per ton.

In the above mentioned case, the truck-fleets are not producing as much as the shovel is producing. That is the maximum production is not achieved by the truck-fleet. For example, in the above case the production potentials of shovels are 8456 tons/shift for 8 cu.yd. shovel, 10407 tons/shift for 10 cu. yd. shovel and 12435 tons/shift for 12.5 cu. yd. shovel respectively and the 85 ton truck is producing about 75 % of 8 cu.yd. shovel prodution, about 59 % of 10 cu. yd. shovel production and 52 % of 12.5 cu. yd. shovel production, since the desired production rate is limited at 6,000 tons per shift.

In cases where the maximum production with the lowest cost per ton is to be obtained it is necessary to determine the production rates and costs by varying the number of trucks. Production potential versus number of trucks for 85 ton truck and 12.5 cu. yd. shovel is already shown in fig 4.3. It is noted that more than 11 trucks would be used to meet the match point <A> , where the shovel potential and truck potential meet.Generally speaking , maximum unit costs for the shovel-truck may be obtained with slightly fewer trucks than indicated by the match point (Morgan and Peterson, 1968). The following tables 4.6, 4.7, 4.8 show the production rates and costs per ton of the various shovel-truck fleet with the increasing number of trucks.

TABLE 4.6. HAUL PERFORMANCE OF 85 T AND 8 CU.YD.TRUCK SHOVEL COMBINATION WITH VARYING NUMBER OF TRUCKS

No.of trucks	Shovel production (T/shift)	Truck-fleet Production (T/shift)	Cost per ton ($/t)
5	8456.40	5306.40	1.16
6	8456.40	6367.68	1.12
7	8456.40	7428.96	1.09
8	8456.40	8456.40	1.07
9	8456.40	8456.40	1.18
10	8456.40	8456.40	1.29

TABLE 4.7. HAUL PERFORMANCE OF 100 T AND 10 CU.YD.TRUCK-SHOVEL COMBINATION WITH VARYING NO. OF TRUCKS

No.of trucks	Shovel production (T/shift)	Truck-fleet production (T/shift)	Cost per ton ($/t)
6	10407.88	6895.68	1.27
7	10407.88	8044.96	1.23
8	10407.88	9194.24	1.21
9	10407.88	10343.52	1.19
10	10407.88	10407.88	1.29
11	10407.88	10407.88	1.40

TABLE 4.8. HAUL PERFORMANCE OF 120 T AND 12.5 CU. YD. TRUCK-SHOVEL COMBINATION WITH VARYING NO. OF TRUCKS.

No.of trucks	Shovel production (T/shift)	Truck-fleet production (T/shift)	Cost per ton ($/t)
5	12435.88	6772.90	1.29
6	12435.88	8127.48	1.24
7	12435.88	9482.06	1.20
8	12435.88	10836.64	1.18
9	12435.88	12191.22	1.15
10	12435.88	12435.88	1.24
11	12435.88	12435.88	1.34

Table 4.9 is the summary of the shovel-truck combinations which produce the maximum production output with the lowest cost per ton.

TABLE 4.9. SUMMARY OF THE MAXIMUM PRODUCTION WITH LOWEST COST PER TON SHOVEL-TRUCK COMBINATION

Dipper capacity (cu.yd.)	8.0	10.0	12.5
Shovel potential	8456.40	10407.88	12435.88
Truck size : 85 ton			
No. of trucks	8	10	11
Fleet prod. T/shift	8456.40	10256.20	11902.22
Cost per ton, $/t	1.07	1.10	1.06
Truck size : 100 ton			
No. of trucks	7	9	10
Fleet prod., T/shift	8038.73	10343.52	12435.88
Cost per ton, $/t	1.19	1.19	1.11
Truck size: 120 ton			
No. of trucks	6	7	9
Fleet prod., T/shift	7768.62	9515.66	12191.22
Cost per ton, $/t	1.20	1.16	1.15

According to table 4.9 , it is observed that the larger equipment would be used if the larger production rates are required. However, the cost per ton is the deciding factor for the selection of the optimum shovel-truck combination Although the cost per ton figures are the minimum cost per ton for each combination, it is still possible to choose the optimum combination for a certain range of production rates. For production rate of up to about 8400 tons per shift the best performance combination appears to be with 85 tons truck with 8 cu. yd. shovel at the cost of $ 1.07 per ton, while 85 ton truck with 10 cu. yd. shovel is the best performance combination for production of about 10,000 tons/shift at the cost of $1.10 per ton and the 85 ton truck with 12.5 cu. yd. shovel is the best performance combination for production rate of about 12,000 tons/shift at the cost of $ 1.06 per ton with the production potential of 11,902 tons/shift.

It is now obvious that in selecting the optimum shovel-truck combination for a given rate of production, it is important to choose the right shovel size whose output will fulfill the desired production rate and hence the maximum production can be obtained by the truck-fleet at the lowest cost per ton of material moved by varying the number of trucks. In the above case, the desired production rate should be about 8,000 tons/shift rather than 6,000 tons/shift for 85 ton truck and 8 cu.yd. shovel combination. The desired production rate , 6000 tons/shift is fulfilled by six 85 ton trucks at the cost of $ 1.12 per ton (which is the best performance combination for that production rate) with 8 cu.yd. shovel. However, the production output of 8 cu. yd. shovel estimated at 8456.40 tons/shift which is quite too far above the desired production rate. According to Morgan and Peterson (1968), as already mentioned, that the unit cost for shovel-truck may be obtained with slightly fewer trucks than indicated by the match point, and following this statement, the optimum costs per ton are determined as already shown in tables 4.6 through 4.8. The following table 4.10 shows the differences between the production capacities and cost per ton for 85 ton truck and 8 cu. yd. shovel combination.

TABLE 4.10. **PRODUCTION AND COST DIFFERENCES FOR LIMITED PRODUCTION RATE AND MAXIMUM PRODUCTION POTENTIAL WITH 85 TON TRUCK AND 8 CU.YD. SHOVEL COMBINATION.**

	Shovel potential (T/shift)	No.of trucks used	Fleet prod. (T/shift)	Cost per ton ($/ton)
Limited prod. (at 6,000 t/s)	8456.40	6	6367,68	1.12
Prod.maximization Cost minimization	8456.40	8	8456.40	1.07
DIFFERENCE		2	2088 .72	0.05

From the above table, it is observed that the truck-fleet production is increased by 32.80% and the cost reduced by 4.4 % in the latter case.

Therefore it is better to use the shovel-truck combination which gives the maximum output with minimum cost rather than limiting the fleet production as in limited production case. If the limited production rate is in question or of prime importance further analyses over a series of wider range of shovel-truck combinations should be carried out so that the combination with maximum production and minimum cost closest to the given production rate could be chosen.

4.1.3 GENERAL REVIEW ON CONVENTIONAL DETERMINISTIC PROCEDURE

Conventional estimate does not model the performance of a shovel-truck system properly, nor does it take into account the random variability of a real situation since the average figures are used in the procedure. (For example, payload weights of the truck, spotting and dumping times). Loading times and truck times are estimated in the program depending on the shovel-and-truck performance. However, once they are obtained, the adjustment is made to obtain the adjusted times and these adjusted times are used throughout the calculation for each shovel- and- truck combination. Conventional estimates do not take into consideration the interactive effect between the shovel and truck at the loading point. That is, it does not take into account the effect of equipment mismatching between shovel and trucks. Since the number of trucks, required to balance the shovel production or the desired production rate is not usually an integer it is necessary to round up to the next higher integer value and this in turn leads to the production loss due to equipment mismatching. A typical example of production loss due to equipment mismatching is already shown in fig. 4.3. In the figure, although the desired production of 6000 tons/shift is fulfilled there is loss in production between shovel and truck-fleet due to mismatching effect. In this case, the shovel potential is estimated at 12345 tons/shift and the fleet production is found to be only

6492 tons/shift and the production loss is 5943 tons/shift. On the other hand, more than 11 trucks are required to meet the shovel potential. There will be still loss in production due to mismatching even if 12 trucks were used. The production loss will be 548 tons/shift and this results in the excessive capacity of the truck potential which is estimated at 12984 tons/shift which is impossible in ideal case. As stated by Morgan and Peterson(1968) and Deakin(1978), there will be some additional loss and this caused the effect of bunching, because of variation in haul unit's cycle times. This effect of bunching is not taken into account in the conventional deterministic estimating procedure. The variation in haul unit's cycle times can be caused by what is commonly known as 'queueing' and 'waiting time' and the estimate of this time cannot be obtained by the conventional procedure by use of an efficiency factor.

At best only average productivity can be predicted and still, requires some further adjustment or some other methods would be applied to get close estimation to the real life. The conventional procedure gives no idea of the degree of variation that would occur in shift to shift operation.

In the above study it was observed that the costs per ton figures for the various combinations seem to be not much different to each other. However, the values are determined on the basis of cost per ton and even this little difference could obtain the significant amount of savings if the annual cost is considered. For example, the costs per ton for 120 ton with 8 cu.yd. and 10 cu.yd. shovel sizes are \$ 1.24 and \$ 1.23 respectively for the required production of 6000 tons/shift, working 2 shifts/day and 300 working days per year as already assumed. The difference in cost \$ 0.01 per ton could save \$ 36000 annually if 120 ton and 10 cu.yd. combination is chosen.

4.2 BY METHODS APPLYING QUEUEING THEORY

Apart from the assumptions mentioned in Appendix 4, the following additional assumptions are made to formulate the queueing model.

Type of Queue Model: Finite Calling Population or Limited Source Model.

Arrival Pattern: Trucks arriving randomly and the back cycle times follow the exponential distribution.

Service Pattern : At random and exponentially distributed.

Queue Discipline: First come first serve; no priorities.

Customer population: Finite calling population.

Service Facility : Single server (one shovel).

A computer program called Program QHAUL is developed to estimate the shovel-truck productivity of an open pit mining operation using one shovel and a number of trucks.

4.2.1 PROGRAM QHAUL

This program is chiefly designed to estimate state probabilities, measures of effectiveness, fleet performance and finally total cost per ton of the shovel-truck system for various truck and shovel combinations. The total cycle times without waiting times first calculated for the trucks by use of the formulae already mentioned in chapter 2. And then, having obtained the average arrival rates of the trucks and the average loading rate of the shovel, the probability of zero truck in the system (Pzero), and other state probabilities (PN) are computed by use of the following expressions:

$$Pzero = \frac{1}{\sum_{N=0}^{NT}\left[NT!/(NT-N)!*(\lambda/\mu)^{N}\right]}$$

$$PN = \frac{NT!}{(NT-N)!} * (\lambda/\mu)^{N} * (Pzero) \quad ; \quad N = 1,2, \text{---} NT$$

Where Pzero = Probability of zero truck at the loading shovel

NT = Total number of trucks (i.e. customer population).

N = No. of truck units in the system (queueing and in service).

λ = The mean arrival rate of the trucks, trucks per hour.

μ = The mean loading rate of the shovel,trucks per hour.

PN = The probability of N trucks waiting or being served.

The program continues to compute the measure of effectiveness of the shovel-truck system. That is expected number of units in the queue, expected number of units in the system (queueing plus servicing), expected waiting time of a unit in the queue and expected waiting time of a unit spends are computed by the following relationships:-

$$XLQ = NT - \frac{\lambda + \mu}{\lambda}(1 - Pzero)$$

$$XLS = XLQ + (1 - Pzero)$$

$$XWQ = \frac{XLQ}{\lambda(NT - XLS)}$$

$$XWS = \frac{XLS}{\lambda(NT - XLS)}$$

Where XLQ = Expected no. of units in the queue.

XLS = Expected no. of units in the system.

XWQ = Expected waiting time of a unit in the queue, hrs.

XWS = Expected waiting time of a unit in the system, hrs.

Having obtained these measures of effectiveness, the effective average total cycle time of a truck is computed by adding the expected waiting time in the queue to the cycle time without waiting time consideration.

$$ATCT = XWQ + TCT$$

Where ATCT = Effective average Total cycle time, min.

XWQ = Expected waiting time of a unit in the queue, min.

TCT = Cycle time (Loading Time + Travel Time + Fixed time) less waiting time, min.

To compute the performance of a shovel-truck fleet system one of the following two expressions can be applied :

$$\text{Fleet production} = (1 - Pzero) * \mu * TPT * SHDU.$$

(OR)

$$\text{Fleet production} = \frac{60 * TPT * NT * SHDU}{ATCT}$$

Where $(1 - Pzero)$ = Fraction of time shovel is busy (usually considered as "production index").

μ = shovel servicing rate, trucks per/hr

TPT = Tons carried per trip, tons.

SHDU = Duration of shift, hrs.

ATCT = Effective ave. total cycle time, min.

NT = No. of trucks in the fleet.

The program, then continues to estimate the total costs per ton of the shovel-truck fleet for the corresponding combination being considered. The total costs per ton is computed on the basis of equivalent annual cost method. The annual equivalent costs for both shovel and truck are computed and the total costs per ton of earth moved is then calculated by

$$TOTCOS = \frac{C_1 + C_2 * NT}{(1\text{-}Pzero) * \mu * TPT * SHDU}$$

(After Carmichael, 1987).

Where TOTCOS = Total cost per ton of the shovel-truck fleet, \$/ton.

C_1 = Cost per shift of the shovel, \$/shift

C_2 = Cost per shift of a truck, \$/shift

Figures 4.4 and 4.5 show the program flowchart and a sample output of the program QHAUL.

4.2.2 RESULTS FROM PROGRAM QHAUL

After completing the computation for all trucks and corresponding shovel combination over a given haul road with varying number of trucks, the following table 4.11 below can be summarized to show the required number of trucks and their related waiting time per cycle, effective total cycle time with waiting time) and their production potentials to fulfill the desired production rate of 6,000 tons/shift.

TABLE 4.11. NO. OF TRUCKS REQUIRED, WAITING TIME AND FLEET PRODUCTION

Truck size t	Shovel size cu.yd	Prod. desired t/sh	No.of trucks required	Waiting time per cycle min.	Eff.total cyc.time min	Fleet prod. t/sh
	8.0	6000	7	6.51	45.58	6368
85	10.0	6000	7	3.53	41.44	6568
	12.5	6000	6	1.97	39.39	6168
	8.0	6000	6	6.00	46.62	6004
100	10.0	6000	6	3.60	43.06	6319
	12.5	6000	6	2.84	41.95	6967
	8.0	6000	6	8.59	52.63	6501
120	10.0	6000	5	4.09	46.99	6205
	12.5	6000	5	2.66	44.51	6369

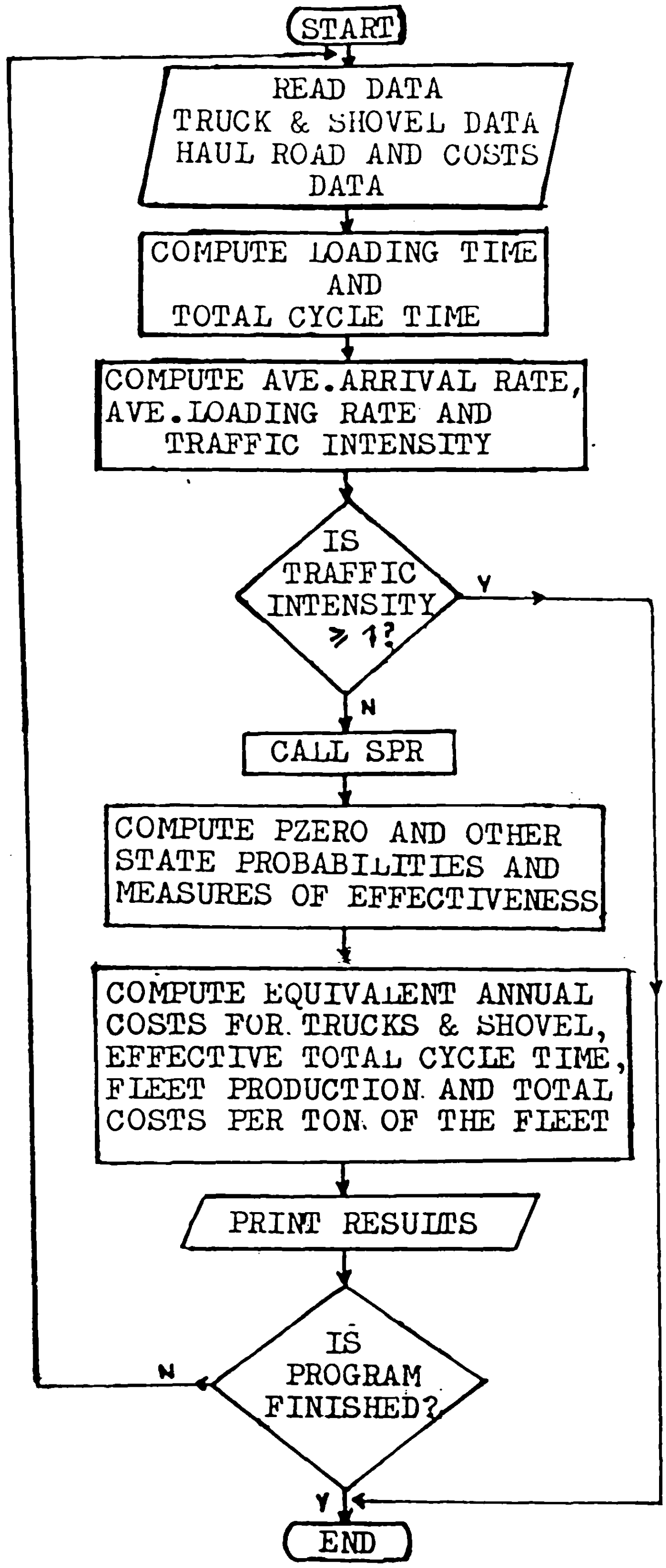

FIG. 4.4 FLOW CHART FOR PROGRAM QHAUL

JOB CONDITIONS

HAUL DISTANCE	=	14600.00 FT
AVERAGE GRADE	=	5.00 PERCENT
ROLLING RESISTANCE	=	3.00 PERCENT
DENSITY	=	2700.00 LBS./CU.YD.
DESIRED PRODUCTION	=	6000.00 TONS/SHIFT
SHIFT DURATION	=	8.00 HOURS

TRUCK AND SHOVEL DATA

TRUCK PAYLOAD	=	85.00 TONS
TRUCK VOLUME	=	64.80 CU.YD.
NO. OF DIPPER SIZES	=	3

NUMBER OF PASSES AND TOTAL CYCLE TIME

DIPPER SIZE CU.YD.	NO.OF PASSES	LOADING TIME MIN	TOTAL HAUL TIME MIN	TOTAL RETURN TIME MIN	TOTAL FIXED TIME MIN	TOTAL CYCLE TIME MIN
8.00	8	4.90	19.20	13.47	1.50	39.07
10.00	6	3.74	19.20	13.47	1.50	37.91
12.50	5	3.26	19.20	13.47	1.50	37.43

TOTAL BACK CYCLE TIME LESS LOAD TIME= 34.17 MIN

TOTAL BACK CYCLE TIME LESS LOAD TIME= 34.17 MIN

TOTAL BACK CYCLE TIME LESS LOAD TIME= 34.17 MIN

AVE.ARRIVAL AND LOADING RATES AND TRAFFIC INTENSITY

DIPPER SIZE CU.YD.	NO.OF TRUCKS	AVE.ARRIVAL RATE TRUCKS/HR	AVE.LOADING RATE TRUCKS/HR	TRAFFIC INTEN. RHO	NO.OF SHOVELS
8.00	7	1.76	12.23	.1435	1
10.00	7	1.76	16.06	.1093	1
12.50	7	1.76	18.42	.0953	1

FIG. 4.5 SAMPLE OUTPUT OF PROGRAM QHAUL

RELATED DIPPER SIZES, NO. OF TRUCKS AND PZEROS

DIPPER SIZE CU.YD.	NO.OF TRUCKS	PZERO
8.00	7	.246855
10.00	7	.368930
12.50	7	.430337

MEASURES OF EFFECTIVENESS

DIPPER SIZE CU.YD.	NO.OF TRUCKS	WAIT TIME IN QUEUE MIN.	WAIT TIME IN SYSTEM MIN.	NUMBER IN QUEUE	NUMBER IN SYSTEM
8.00	7	6.51	11.41	1.00	1.75
10.00	7	3.53	7.27	.60	1.23
12.50	7	2.59	5.85	.45	1.02

STATE PROBABILITIES

DIPPER	NO.OF	STATE PROBABILITIES						
SIZE	TRUCKS	P1	P2	P3	P4	P5	P6	P7
8.00	7	.24801	.21357	.15326	.08799	.03788	.01087	.00156
10.00	7	.28233	.18520	.10123	.04427	.01452	.00317	.00035
12.50	7	.28710	.16418	.07824	.02983	.00853	.00163	.00015

FIG. 4.5 CONTINUED

WAITING TIME PER TRIP,EFF.CYCLE TIME AND ARRIVAL RATE

DIPPER SIZE CU.YD.	NO.OF TRUCKS	WAITING TIME/TRIP MIN.	EFF.CYC. TIME MIN.	EFF.ARR. RATE/HR TRUCKS	TONS/ TRIP TONS	FLEET PROD/HR TONS
8.00	7	6.51	45.58	1.32	86.40	796.11
10.00	7	3.53	41.44	1.45	81.00	821.01
12.50	7	2.59	40.02	1.50	84.38	885.53

FLEET PRODUCTION

DIPPER SIZE CU.YD.	FLEET PRODUCTION T/S	SHOVEL CAPACITY T/S	TONS MOVED T/S
8.00	6368.90	8456.40	6368.90
10.00	6568.10	10407.88	6568.10
12.50	7084.26	12435.88	7084.26

COSTS PER TON

DIPPER SIZE CU.YD.	COST PER TON TRUCK $/TON	COST PER TON SHOVEL $/TON	TOTAL COST PER TON $/TON
8.00	1.053	.213	1.267
10.00	1.021	.259	1.280
12.50	.947	.295	1.242

CALCULATION COMPLETED

FIG. 4.5 CONTINUED

The above table 4.11 indicates that for a particular size of truck, the waiting time per cycle of a truck reduces as the size of shovel increases. It is also noticed that the waiting time per cycle of a truck increases, the shovel size and number of truck being the same, since the number of passes required to load a truck increases as the size of truck increases.

The relationship between the waiting time and number of trucks is shown in figure 4.6 below, and the detailed performance summaries of each truck and its corresponding shovel size combination for a given set of operating conditions and production rate desired are shown in tables 4.12 through 4.14. The utilization of a truck can be determined by the following expression :

$$Em = \frac{\text{Travel Time (Out-of-system time)}}{\text{Total cycle time}}$$

OR

$$Em = \frac{A}{NT}$$

Where Em = Truck Utilization, %

A = Average no. of units out of system

= NT - XLS

NT = Customer population (total no. of trucks).

XLS = EXpected no. of units in the system.

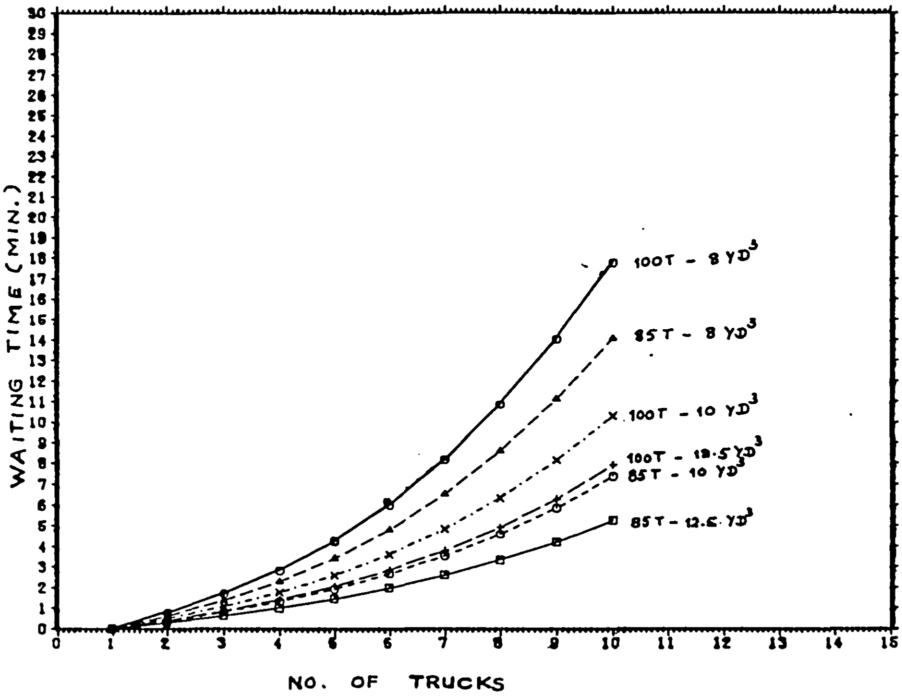

FIG. 4.6 TRUCKS WAITING TIMES VERSUS NUMBER OF TRUCKS

TABLE 4.12 SUMMARY OF HAUL PERFORMANCE FOR 85 TON TRUCK

DIPPER SIZE (CU.YD.)	8.0	10.0	12.5
No. of passes used	8	6	5
Waiting time per trip (mn).	6.51	3.53	1.97
Effec. tot. cyc. time (mn).	45.58	41.44	39.39
Tons/Shift/Truck (tons)	796.11	821.01	771.04
Tons/Trip/Truck (tons)	86.40	81.00	84.34
Trips/Shift/Truck	10.53	11.58	12.19
No. of trucks used	7	7	6
Ave. arrival rate (trks/hr)	1.76	1.76	1.76
Ave. service rate (trks/hr)	12.23	16.06	18.42
No. of shovels used	1	1	1
Truck Flt prod.(t/sh)	6368.90	6568.10	6168.32
Shovel potential(t/sh)	8456.40	10407.88	12435.88
Tons moved/sh (t/sh)	6368.90	6568.10	6168.32
COSTS			
Truck cost ($/ton)	1.05	1.02	0.93
Shovel cost ($/ton)	0.21	0.26	0.34
Total cost $/ton)	1.26	1.28	1.27
Truck utilization (%)	75 .00	82.43	86.57

4.13 **SUMMARY OF HAUL PERFORMANCE FOR 100 TON TRUCK**

DIPPER SIZE (CU.YD.)	8.0	10.0	12.5
No. of passes used	9	7	6
Waiting time per trip (min)	6.00	3.60	2.84
Effec. tot. cyc.time (min)	46.62	43.06	41.85
Tons/Shift/Truck (tons)	750.54	789.97	870.88
Tons/Trip/Truck (tons)	97.20	94.50	101.25
Trips/Shift/Truck	10.30	11.15	11.47
No. of trucks used	6	6	6
Ave. arrival rate (trks/hr)	1.17	1.17	1.17
Ave. service rate (trks/hr)	10.88	13.77	15.35
No. of shovels used	1	1	1
Truck Flt. prod.(t/sh)	6004.31	6319.76	6967.06
Shovel potential (t/sh)	8456.40	10407.88	12435.88
Tons moved/sh (t/sh)	6004.31	6319.76	6967.06
COSTS			
Truck cost ($/ton)	1.17	1.11	1.01
Shovel cost ($/ton)	0.23	0.27	0.30
Total cost ($/ton)	1.40	1.38	1.31
Truck utilization (%)	75.33	81.54	83.83

TABLE 4.14 SUMMARY OF HAUL PERFORMANCE FOR 120 TON TRUCK

DIPPER SIZE (CU.YD.)	8.0	10.0	12.5
No. of passes used	11	9	7
Waiting time per trip (min)	8.59	4.09	2.66
Effec. tot. cyc. time (min)	52.63	46.99	44.51
Tons/Shift/Truck (tons)	812.65	775.74	796.24
Tons/Trip/Truck (tons)	118.80	121.50	118.13
Trips/Shift/Truck	9.12	10.24	10.80
No. of trucks used	6	5	5
Ave. arrival rate (trks/hr)	1.61	1.61	1.61
Ave. service rate (trks/hr)	8.90	10.71	13.16
No. of shovels used	1	1	1
Truck Flt.prod.(t/sh)	6501.17	6205.89	6369.90
Shovel potential (t/sh)	8456.40	10407.88	12435.88
Tons moved/sh (T/sh)	6501.17	6205.89	6369.90
COSTS			
Truck cost ($/ton)	1.23	1.07	1.04
Shovel cost ($/ton)	0.21	0.27	0.33
total cost ($/ton)	1.44	1.44	1.37
Truck utilization (%)	70.85	79.40	83.80

According to the tables 4.12, 4.13, and 4.14 it is observed that the best performance shovel-truck combination appears to be with 85 ton truck with 8 cubic yard shovel at the cost of $ 1.26 per ton. The best performance seems to be the same as in conventional procedure. However, the number of trucks used with the queueing model is 7 instead of 6 as in conventinal procedure. This means that the more number of trucks would be used to meet the same production rate with the queueing model than with the conventional model, since the waiting times are taken into account in the queueing calculations.

Like the conventional procedure, because of the limited rate of production desired, the trucks are not producing the maximum output. For example in 85 ton truck, 7 trucks are producing about 75 % of 8 cubic yard shovel production, about 63% of 10 cubic yard shovel production and 6 trucks are producing about only 50 % of 12.5 cubic yard shovel production.

In cases where costs are of prime importance so as to produce maximum production with minimum total cost per ton, it is necessary to carry out the optimization studies of the operation being considered. To do this, the production rates and related costs are estimated by varying the number of trucks. The following tables 4.15 through 4.19 are some of the results from the optimization study with increasing number of trucks.

TABLE 4.15. **HAUL PERFORMANCE OF 85 TON - 8 CU.YD.TRUCK-SHOVEL COMBINATION WITH VARYING NUMBER OF TRUCKS**

NO. OF TRUCKS	FLEET PRODUCTION (TONS/SHIFT)	COST/TON ($/TON)
1	1061.36	2.18
2	2089.81	1.57
3	3076.69	1.38
4	4011.34	1.29
5	4881.08	1.26
6	5671.75	1.25
7	6368.90	1.27
8	6980.03	1.30
9	7450.54	1.34
10	7801.49	1.40

TABLE 4.16 **HAUL PERFORMANCE OF 85 TON - 12.5 CU.YD. TRUCK-SHOVEL WITH VARYING NUMBER OF TRUCKS**

NO. OF TRUCKS	FLEET PRODUCTION (TONS/SHIFT)	COST/TON ($/TON)
1	1082.12	2.82
2	2147.97	1.87
3	3194.16	1.55
4	4216.53	1.41
5	5209.97	1.32
6	6168.32	1.27
7	7084.26	1.24
8	7949.31	1.23
9	8754.01	1.22
10	9488.40	1.23

TABLE 4.17. HAUL PERFORMANCE OF 100 TON -10 CU. YD. TRUCK-SHOVEL COMBINATION WITH VARYING NUMBER OF TRUCKS

NO. OF TRUCKS	FLEET PRODUCTION (TONS/SHIFT)	COST/TON ($/TON)
1	1149.28	2.50
2	2270.87	1.78
3	3358.00	1.55
4	4402.27	1.45
5	5393.51	1.40
6	6319.76	1.38
7	7167.73	1.38
8	7923.80	1.40
9	8575.76	1.43
10	9115.23	1.47

TABLE 4.18 HAUL PERFORMANCE OF 120 TON - 12.5 CU. YD. TRUCK-SHOVEL COMBINATION WITH VARYING NUMBER OF TRUCKS

NO. OF TRUCKS	FLEET PRODUCTION (TONS/SHIFT)	COST/TON ($/TON)
1	1354.86	2.52
2	2677.93	1.77
3	3961.51	1.54
4	5196.04	1.43
5	6369.90	1.37
6	7469.43	1.35
7	8479.31	1.34
8	9383.69	1.36
9	10168.03	1.38
10	10821.81	1.42

TABLE 4.19 HAUL PERFORMANCE OF 100 TON - 12.5 CU.YD. TRUCK-SHOVEL COMBINATION WITH VARYING NUMBER OF TRUCKS

NO. OF TRUCKS	FLEET PRODUCTION (TONS/SHIFT)	COST/TON ($/TON)
1	1245.58	2.62
2	2466.41	1.80
3	3656.81	1.53
4	4809.74	1.41
5	5916.59	1.35
6	6967.06	1.31
7	7949.28	1.29
8	8850.18	1.30
9	9656.51	1.31
10	10356.42	1.34

The following table 4.20 shows a summary of the best performance shovel-truck combinations with their related fleet production and cost per ton.

TABLE 4.20. SUMMARY OF THE BEST PERFORMANCE SHOVEL-TRUCK COMBINATIONS

COMBINATION	NO. OF TRUCKS USED	FLEET PRODUCTION (TONS/SHIFT)	COST PER TON($/T)
85T-8Cu.Yd.	6	5671.75	1.25 *
100T-8Cu.Yd.	5	5200.27	1.39
120T-8Cu.Yd.	4	4752.87	1.41
85T-10Cu.Yd.	8	7320.07	1.28 *
100T-10Cu.Yd.	6	6319.76	1.38
100T-10Cu.Yd.	7	7167.73	1.38
120T-10Cu.Yd.	5	6205.89	1.35
85T-12.5Cu.Yd.	9	8754.01	1.22 *
100T-12.5Cu.Yd.	7	7949.28	1.29
120T-12.5Cu.Yd.	7	8479.31	1.34

According to table 4.20, it is observed that for each shovel category size it is possible to select the size of truck which will give the maximum production with the minimum costs per ton. For example for 8 cubic yard shovel category size, the maximum production with the minimum cost per ton is obtained with 85 ton truck, while for 10 cubic yard and 12.5 cubic yard shovels also give the maximum production with lowest costs per ton with 85 ton truck as shown by (*) in the table. The overall best performance combination appears to be with 85 ton and 12.5 cubic yard truck-shovel combination with the fleet production output of 8754 tons/shift at the cost of $ 1.22 per ton. However, this is only true in cases where number of trucks exceeds 6. The relationship between number of trucks versus total cost, per ton is shown in figure 4.7.

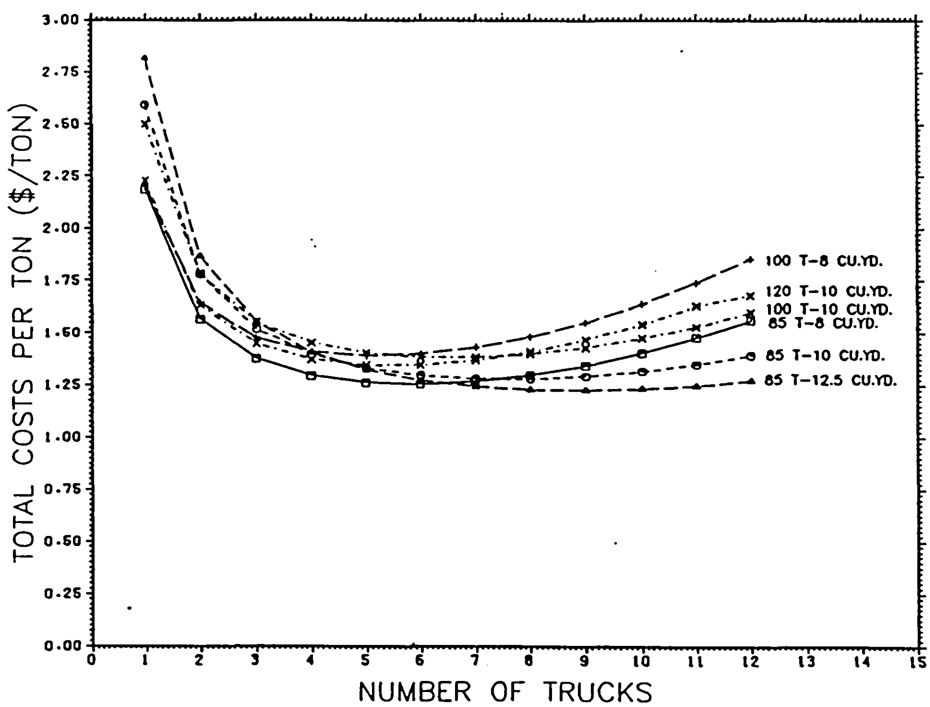

FIG. 4.7 TOTAL COSTS PER TON VERSUS NUMBER OF TRUCKS

According to figure 4.7, it is observed that the total costs per ton curves for 85 ton truck with 12.5 cubic yard combination and 85 ton truck with 8 cubic yard shovel combination appear to the least cost per ton curves compared with other cost per ton curves.

The 85 ton truck with 12.5 cubic yard shovel combination is found to be best suited where working with 7 to 10 number of trucks; i.e. fleet production of about 7000 tons to about 9500 tons per shift (see table 4.16 as well). On the other hand the 85 ton truck and 8 cubic yard combination seems to be best suited for number of trucks less than or equal to 6 ; i.e. production between 1000 tons to about 5600 tons per shift (see table 4.15 as well). According to tables 4.15 and 4.16 it is observed that if the desired production is limited at 6000 tons per shift, it is better to use 85 ton truck and 12.5 cubic yard shovel with 7 trucks which will produce 7084 tons per shift at the cost of $ 1.24 per ton rather than using 7 or 8 trucks with 85 ton and 8 cubic yard shovel combination which produce the fleet production of 6368 tons or 6960 tons at the cost of $ 1.27 and $ 1.30 per ton.

It is now clear that 85 ton truck and 12.5 cubic yard shovel combination should be used where production desired ranges from 6000 to 9500 tons per shift. Figure 4.8 shows the fleet production output versus number of trucks for the various shovel-truck combinations being considered.

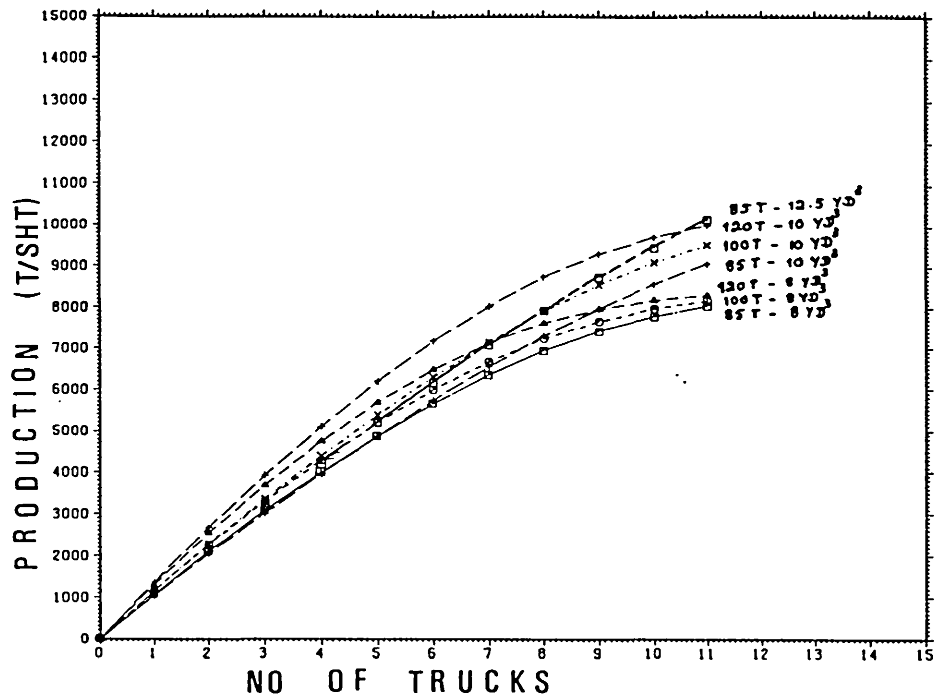

FIG. 4.8 PRODUCTION OUTPUT VERSUS NUMBER OF TRUCKS

It is observed that the combinations with a smaller shovel dipper size seem to be more influenced by the queues than the combinations with bigger shovel dipper size. This is because more waiting times are spent by larger trucks with smaller shovel dipper size. The relationship between waiting time versus number of trucks is already shown in figure 4.6.

According to table 4.20 , it is also observed that the optimum number of truck fleet for 85 ton, 100 ton and 120 ton with 8 cubic yard shovel combinations are found to be 6, 5 and 4 and producing 5671 tons, 5200 tons and 4752 tons per shift respectively. In case of production rate limited at 6000 tons per shift, it is obvious that these 3 sets of combinations are out of question to be selected, although 7 , 6 and 6 number of trucks, in which 85 ton - 8 cubic yard combination being the best, have been allocated to each related combination previously before introducing the optimization study (see table 4.12 as well).

According to figure 4.8, it is also observed that although the high capacity trucks require more time to load and generally have less maneuverability they can transport larger volume or tonnage of material and thus obtain the greater production than that for the fleet of smaller trucks. This relationship is also shown in figure 4.9.

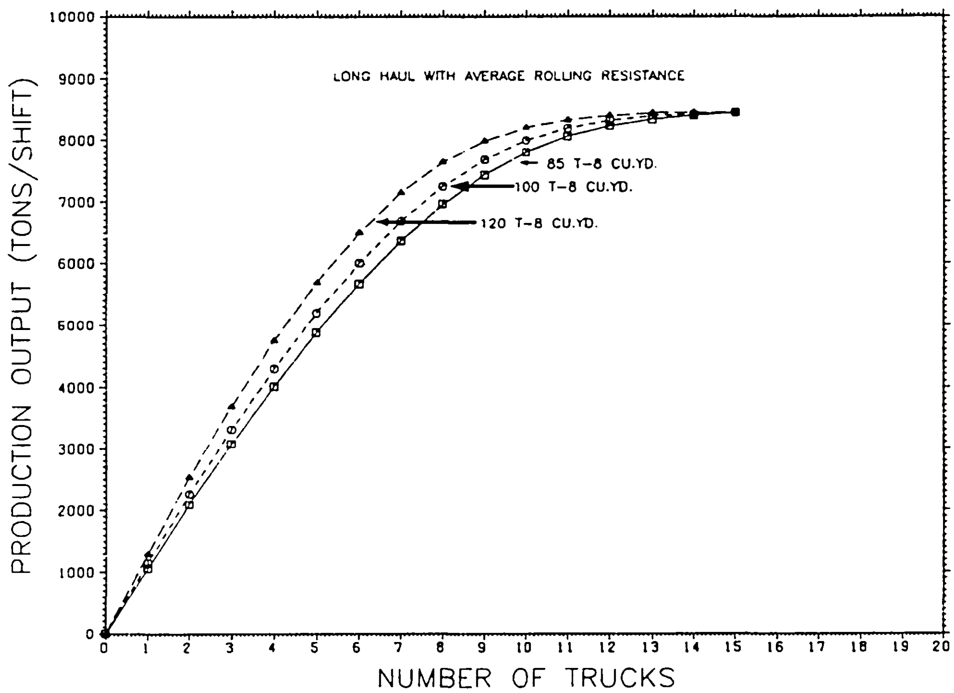

FIG. 4.9 SENSITIVITY ON PRODUCTION OUTPUT DUE TO CHANGE IN TRUCK SIZE

On the other hand,with the increasing capacities of loader size the fleet production seems little changes with smaller trucks (i.e. with smaller number of trucks).However, the fleet production increases outstandingly as the number of trucks increases since the larger capacity shovels can handle more number of trucks as shown in figure 4.10.

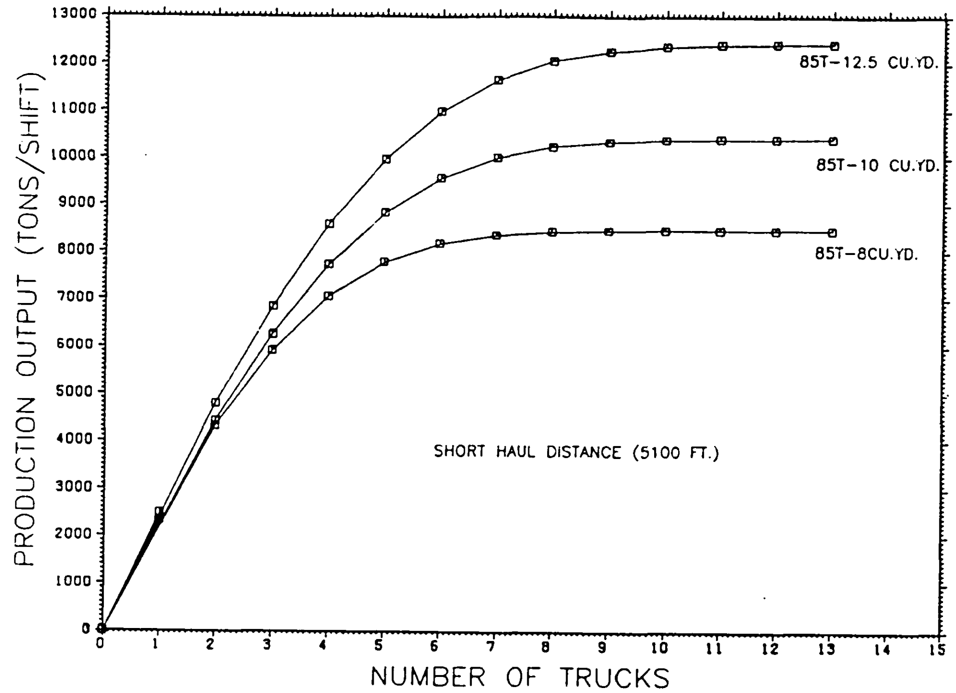

FIG. 4.10 PRODUCTION OUTPUT VERSUS NUMBER OF TRUCKS OVER SHORT HAUL WITH INCREASING SHOVEL SIZE

Further analysis over short haul distance is also carried out with 8, 10 and 12 cubic yard shovel sizes. The relationship between the production output capacities versus the number of trucks can be seen in figure 4.10. The figure also states the sensitivity on production output of the truck-fleets due to changes in shovel dipper size.

Figure 4.11 shows the sensitivity on production output of 85 ton truck fleet with 8 cubic yard shovel due to changes in haul distance.

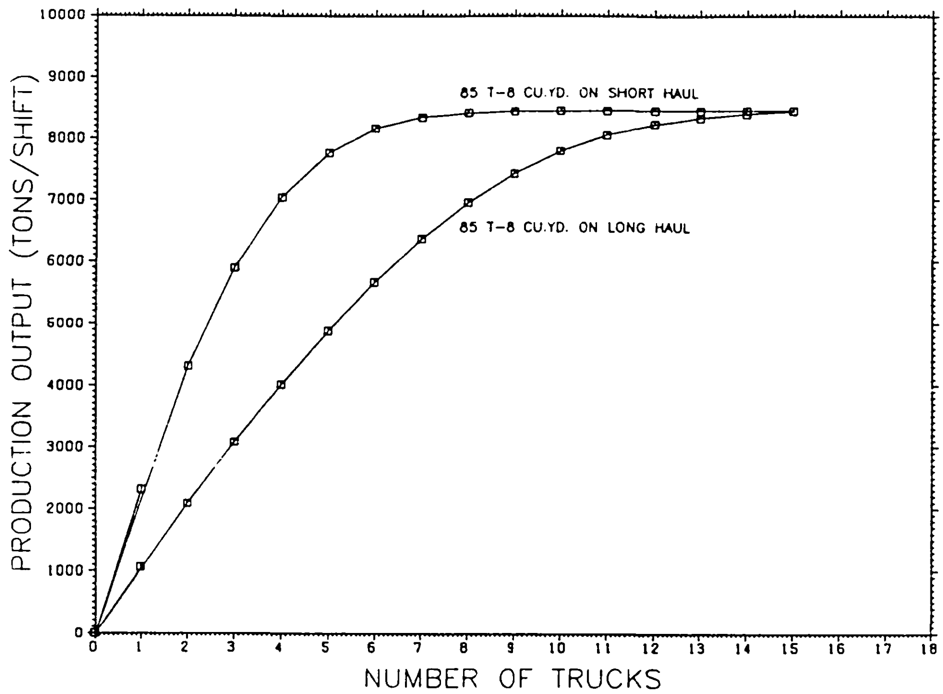

FIG. 4.11 SENSITIVITY ON PRODUCTION OUTPUT DUE TO CHANGE IN LENGTH OF HAUL ROAD

It is observed that the total operation is more affected by the queues and waiting times in short haul than the long haul even in the smaller number of trucks.

4.3.0 COMPUTERIZED SIMULATION METHOD.

In large open-pit mines the truck haulage system becomes so complex that quantitative results are difficult to obtain analytically by use of the conventional empirical analysis. Computer simulation methods can be used to overcome such difficulties especially the effects of bunching of haul units . The computer simulation program MONTE is developed for this purpose and the program mainly consists of two parts: the first part as a deterministic part, in which the truck's load capacity and performance characteristics are dealt with while the second part as a stochastic part, in which the loading cycle, spotting and dumping times, and the breakdown times are treated.

The following additional assumptions are made to formulate the program.

(1) <u>Shift Time</u>

One nominal shift was taken to be of eight hours duration which is equivalent to 28800 seconds.

(2) <u>Shift Start</u>

It was assumed that at the start of shift, shift elapsed time was set at zero and the allowance for preparatory time of 10 min (600 sec.) was given and hence the elapsed shift time was advanced to 600 sec. immediately. All trucks were available and waiting at the loading point to be loaded. The shovel was considered to start loading when the first truck had been spotted into a suitable loading position.

(3) <u>Shift End</u>

The shift was effectively terminated after 7 hours 50 minutes had elapsed from shift start.

Before shift ends 10 minutes were considered necessary for all men and mobile equipment to return to the loading point.

(4) Work Breaks

Three work breaks were designed into the program:

(a) A 10 minute break after two hours of shift commencement.

(b) A 20 minute break after four hours of shift commencement.

(c) A 10 minute break after six hours of shift commencement.

It was assumed, that on each of these occasions, work would stop at the signal of a hooter. All equipment would then stop effective operation and the men would take their rest/lunch period on the job and would continue at the end of each work break from where they had left off.

(5) Truck field Breakdowns

It was assumed that up to 25 % of the trucks would have field repair breakdowns per shift. That is, if total number of trucks allocated to the shovel = N, then, the assumed number of trucks having field breakdowns per shift is

$$N / 4$$

The number of breakdowns is then rounded to the nearest integer number in the program.

Having obtained the number of breakdowns, it was assumed the mean breakdown time of each breakdown to be 50 minutes (3000 sec.) with 10 minutes (600 sec.) standard deviation. The field breakdown repair time was considered the total elapsed

time from when the truck was stopped with an unspecified fault to the time the truck resumed natural operations.
The maximum shift time available for production would be as follows:-

Total shift time	=	480 min.
Total productive time loss due to work breaks	=	40 min.
End of shift time loss	=	10 min.
maximum shift time available for production		430 min.

The maximum available time ratio for the broken down truck and the maximum available time ratio for the remaining trucks can be expressed as :

$$\text{maximum available time ratio for broken down truck} = \frac{(430 - 50) * N/4}{(430 * N)}$$

$$\text{maximum available time ratio for the remaining trucks} = \frac{430 * 3/4 * N}{(430 * N)}$$

Therefore the maximum overall availability for the truck fleet would be maintained at about 97 % during the shift throughout the calculation.

(6) Shovel breakdown

The shovel breakdown time was considered as the total elapsed from the time the shovel was stopped with an unspecified fault to the time it resumed normal operations. Mean breakdown time per shift was assumed 50 mins. (3000 sec.) so as to maintain the shovel availability of 90 % and the estimated standard deviation would be 10 mins. (600 secs). It was assumed that one breakdown time per shovel per shift would be taken place.

4.3.1 PROGRAM MONTE

This program is the main program to simulate the operations of one shovel and its associated trucks. The program consists of two main parts :

(1) Deterministic Part - in which the truck's load capacity and performance characteristics are dealt.

(2) Stochastic Part - in which the loading cycle, spotting and dumping times and the breakdown times are treated.

(1) Deterministic Part

The Deterministic Simulation is used to simulate the movement of a truck along a haul profile with a given set of road grades and rolling resistances. Computation of acceleration is the key to writing a truck haul simulation program to determine travel time. The performance of an automotive vehicle can be calculated by the equation of Newton's Second Law of Motion :

$$\Sigma F_{\theta} = M a_{\theta}$$

If rate of acceleration is constant, rectilinear motion formulae regarding velocity and speed relationships can be reduced to familiar algebraic equations. Typical truck performance curves as supplied by manufacturers of all off-high way haulage trucks relate vehicle speed and rimpull or propelling force available at the drive wheels. It is evident that rimpull, and consequently, acceleration vary greatly with vehicle speed so that the constant acceleration equations are seemingly inapplicable. However, if the small increments of velocity, time and distance are considered, acceleration during this increments is approximately constant, and this allows the use of constant acceleration formulae as shown below :

$$V_i = V_{i-1} + a_i t_i$$
$$d_i = V_{i-1} t_i + 1/2 a_i t_i$$
$$V_i^2 = V_{i-1}^2 + 2 a_i d_i$$

Where

V_i = velocity at end of increment i

a_i = acceleration during increment i

t_i = time during increment i

d_i = distance travelled during t_i

O'Neil and Manula. (1967) in their computer program considered increments of time, while Gibbs et al. considered increments of velocity. In this program the increments in velocity of 1 mph is used.

Starting from a known velocity, the avilable rimpull is computed by using a mathematical representation of the truck performance chart or using the approximate expression shown below if truck performance charts are not available.

$$\text{Available Rimpull} = \frac{375 * \text{Horsepower Mech. Efficiency}}{\text{Velocity}} \quad ; \text{ lbs.}$$

If the rimpull capabilities of a truck are in excess of the requirements necessary to overcome the combined rolling resistance and grade resistance (i.e. total effective resistance), acceleration will occur and of course, the converse also is true. The amount of acceleration can be derived from the basic equation :

$$A = F / M$$

Since, the acceleration in mph/sec is desired, the following conversion is used to obtain the desired acceleration :

$$A = \frac{21.95 * \text{Net Rimpull in pounds}}{\text{operating weight of the truck in pounds}} \quad ; \text{ mph/sec.}$$

Where Net Rimpull = (Available Rimpull - Rimpull required to overcome Total Resistance).

This acceleration is then assumed constant for an incremental increase in the velocity. The new velocity fixes a new rimpull and acceleration. Time and distance are computed for each increment in velocity and are cumulated and recorded.

If the haulage road profile remains unchanged, a maximum velocity is soon achieved where the truck has no surplus power for acceleration. Acceleration therefore becomes zero and velocity constant. When the profile changes, a new set of retarding forces is encountered, and a new acceleration rate, either positive or negative is computed.

Subroutine VELPOT is called on whenever necessary to compute the potential velocity of the truck at a given road grade, rolling resistance, total weight of the truck, horsepower and mechanical transmission efficiency of the truck.

Subroutine AVRIM is called on whenever necessary to compute the available rimpull produced by the truck's power train at a given velocity, horsepower and mechanical efficiency of the truck.

The haulage profile is broken up into a number of sections each with its own road grade and surface condition. This defines the two resisting forces rolling and grade resistances. These two resistances are added algebraically to truck available rimpull to obtain the net rimpull. A positive value accelerates the truck, and the negative one causes deceleration. Rimpull may vary continuously over a haul road section, but profile resistances are constant for any given road section.

Travel Constraints.

Every mine has peculiar operating conditions that make constraints necessary. Listed below are the constraints that are considered in the program.

(1) Maximum Speed - Normally, for safety reasons,trucks are not allowed to exceed a certain maximum speed even though they are capable of doing so. 30 mph is set as a maximum

speed in the program.

(2) Deceleration - A deceleration rate, considered constant, is needed to stop the truck. 3 mphps is used in the model.

(3) Maximum Acceleration - Too large an acceleration will produce spinning wheels and worn out tyres. A maximum acceleration rate is therefore needed to be specified. 3 mphps is established for the program.

(4) Maximum Downhill Speeds - Safe truck speeds decrease as downhill grades become steeper. For each downhill grade, a maximum allowable speed must be specified, and these speeds are listed in table 8 in appendix 1 .

Output

The cumulative time and distance, available and net rimpull for each section of given distance are printed out.

(2) Stochastic Part

This is the part in which the input variables such as the loading time, spotting and dumping times and breakdown times for shovel and trucks are concerned. Stochastic or probabilistic simulation, which is also known as Monte Carlo Simulation is a random analysis and is defined as a procedure by which one can obtain approximate evaluations of mathematical expressions which are built up of one or more probability function. In this type of simulation, a series of random numbers is generated with each number having an equal likelihood of occuring. Each of these numbers has associated with it the resulting condition that corresponds to that number.

There are two possible methods of assigning values to the variables in a random analysis. One uses a random number to fix the value of a random variable through the use of a probability distribution function and the other uses a cumulative relative frequency polygon plot of the paticular element. The method of assigning values to the variables in the program was through the use of a probability distribution function provided that each variable would have a separate distribution function defining it.

The program is based on some restricting assumptions. Firstly, it only deals with a single shovel system. Secondly it does take into account equipment interactions only at the shovel not on the haul roads or at the dump.

The general operation of the program is listed as follows :

Shift Start

At the beginning of the shift, a shovel was assigned a certain number of trucks. The shovel is considered to start loading when the first truck has been spotted in suitable loading position. All trucks are loaded by the shovel followed by each other and the loaded trucks travel from the loading point and then return back to the shovel. A record is kept of the loads carried, shovel idle time and truck waiting time at the shovel.

Truck Field Breakdown Positions.

It was assumed that approximatelt 25 % of the truck fleet would have field repairs carried out per shift. Therefore the number of trucks NT allocated to the shovel was divided by four to obtain the number of field breakdown and the value is rounded to the nearest integer value. Having obtained the number of truck field breakdowns, their respective positions, LCBDWN, in terms of the shovel load number, were generated by subroutine BRDGEN.

Elasped Time

This is the elasped time of shift for each truck and is considered as the array ELT (I). This ELT (I) for each truck is advanced by the spotting time at the loading point, loading time, spotting time at dumping point, dumping time and the travelling time. These time elements, except the travelling time, are considered to vary randomly. Each of these time variables are controlled by separate probability distributions. Waiting time, if it is associated by a truck at the shovel, is also added to the corresponding ELT of that truck.

Mode of operation of the truck.

This is considered as Array ITYPE (I) in the program. It determines which three operations (loading, travelling and dumping) are to be performed by each truck.The program determines the lowest ELT (I) of the trucks and carries out the operation determined by ITYPE (I). If this is a loading operation, for example, the program will check for truck or shovel waiting time and the values are cumulated and recorded. It will also generate the payload of the truck which is also considered to vary randomly and the cumulated payload is recorded as well.

Truck Breakdown Times.

The truck breakdown times are also considered to vary randomly. The breakdown of a truck will take place when the progressive truck load counter equals to the previously generated load counter positions which were already stored in array LCBDWN(K). Then the breakdown time is computed by subroutine TIME. Once the breakdown time is computed, it is added to the corresponding ETL(I) of that truck which is broken down.

Truck Spotting Times at Loading and dumping points and loading Time.

These times are also considered as random variables and are computed by Subroutine ANORM. These time elements are then added to the corresponding ELT(I) of the trucks.

Truck Payloads.

Truck payload is calculated by Subroutine ANORM.

Shovel Time.

The shovel time was advanced immediately at shift start to the time taken for the preparatory operations. Shovel breakdown times are added to the shovel time when the shovel breaks down. Times taken for three workbreaks are also added to shovel time when they are due.

Shovel Breakdowns.

The load counter for shovel breakdown, LCSBD, in terms of the shovel load number, was generated by Subroutine BRDGEN. Then the shovel breakdown will take place when the progressive truck load counter equals to the generated load counter and the shovel breakdown time is calculated by Subroutine TIME. The shovel time is then advanced by adding the shovel breakdown time and the elasped shift times of trucks (ELT(I)) is also advanced by the shovel breakdown time and hence the truck waiting times are computed due to the breakdown of the shovel.

Work Break Periods.

Three work break periods were incorporated into the program. In each case it was assumed that all equipment immediately stopped effective operations for the duration of the work break period. Therefore the elapsed shift time, shovel time were advanced by the duration of the work break. The program determines the elapsed shift time at which each of the previously designed work break time starts.

Shift End.

When the lowest ELT(I) becomes more than shift time (i.e. shift duration less the finishing time at end of shift) the shift is considered finished. However, in practice, especially if the long haul distances are incorporated, the allowance of finishing time (600 sec in this program) at the end of shift would not be quite sufficient to call the trucks back to the shovel. In such a case, the shift will be finished when the lowest ELT(I) becomes more than shift time less the total travelling time back to the shovel.

Cost Estimation

Having obtained the truck travel time and travelling distance by deterministic simulation procedure and the flleet production, waiting times and and breakdown times by stochastic simulation procedure, the program continues to compute the total cost per ton of the shovel-truck fleet by use of an equivalent annual cost method. Capital Recovery factor and annual cost for each truck and shovel are computed previously in the program.

Output

Finally, both the results from the deterministic and stochastic parts are printed out. Travel and return performances of a truck over a given length of separate sections of the haul road and and total travelling time and distance are printed out first and then the results from the stochastic part. The stochastic results are printed out on the basis of series of shift with a certain number of trucks allocated to the shovel. The average total tonnage and average total cost per ton for each number of each truck are also printed out so as to carry out the optimization procedures.

Program flowchart, and sample output are shown in figures

4.12 and 4.13.

Input Parameters For Stochastic part.

Means and Standard deviations are required for the variable input parameters such as loading time, payloads, dumping time, truck and shovel breakdown times, etc., to define their probability functions.

If the equipment is already in operation and working under similar conditions to that being modelled, the work study will give all the values needed. It will also give an indication of the best distribution to use or allow the use of empirical distributions. In the case of a feasible studies, the manufacturers' figures should be consulted.

For estimation purposes, the following assumptions were made for the various parameters used in the program.

(1) Loading Time.

Loading time is considered to be normally distributed. O'Neil and Manilla (1967) derive the loading time for the various trucks and shovel from the main distributions for the 32 ton trucks by increasing the parameters proportionately. They used the mean loading time of 2.11 min with a variance of 0.43 min for the 32 ton truck under hard digging conditions. The data were found to be discribed very well by the normal distribution. Due to the lack of data, the above information is also used to find the loading times of various trucks and shovel in the program.

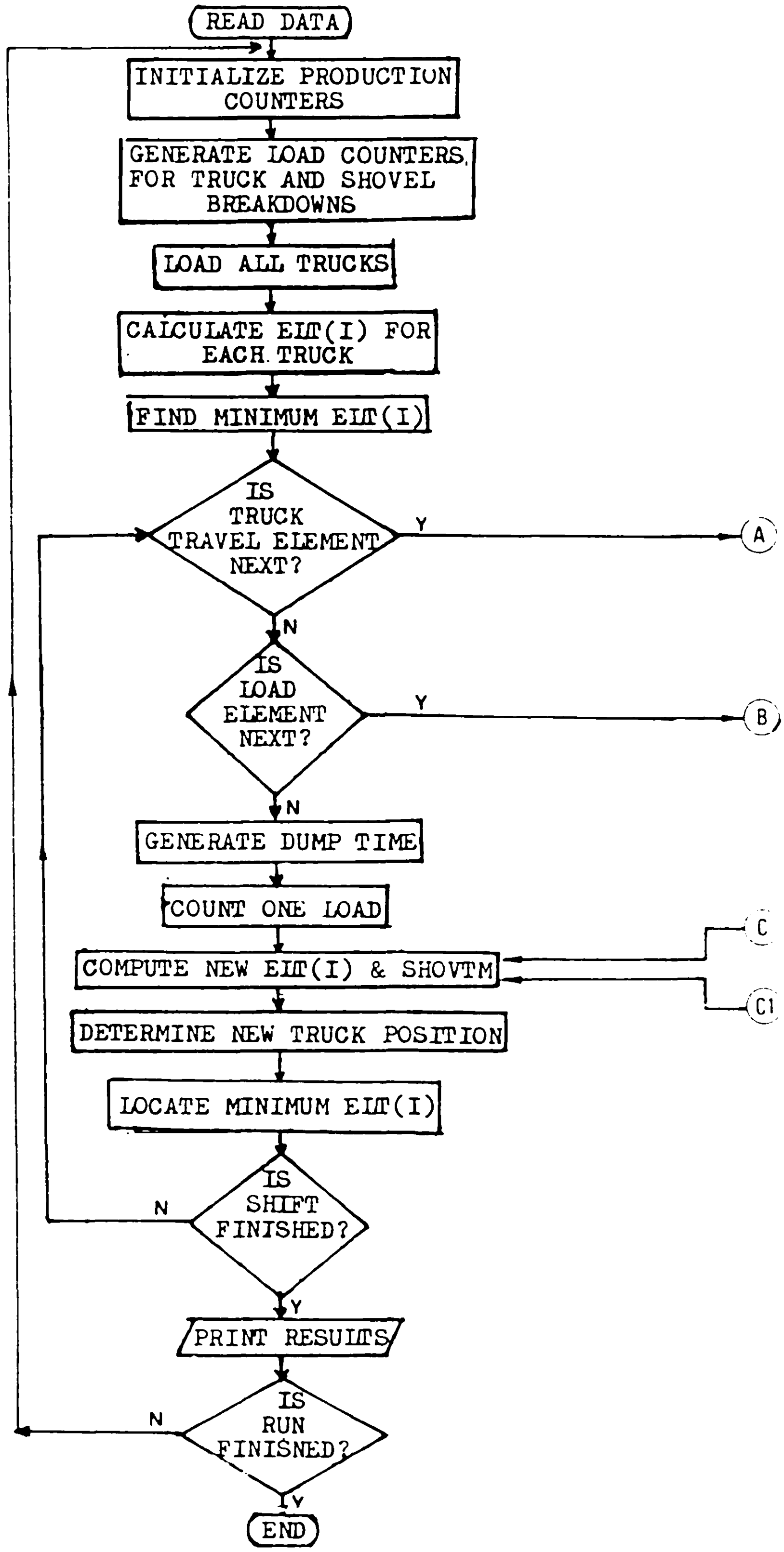

FIG. 4.12 FLOW CHART FOR PROGRAM MONTE

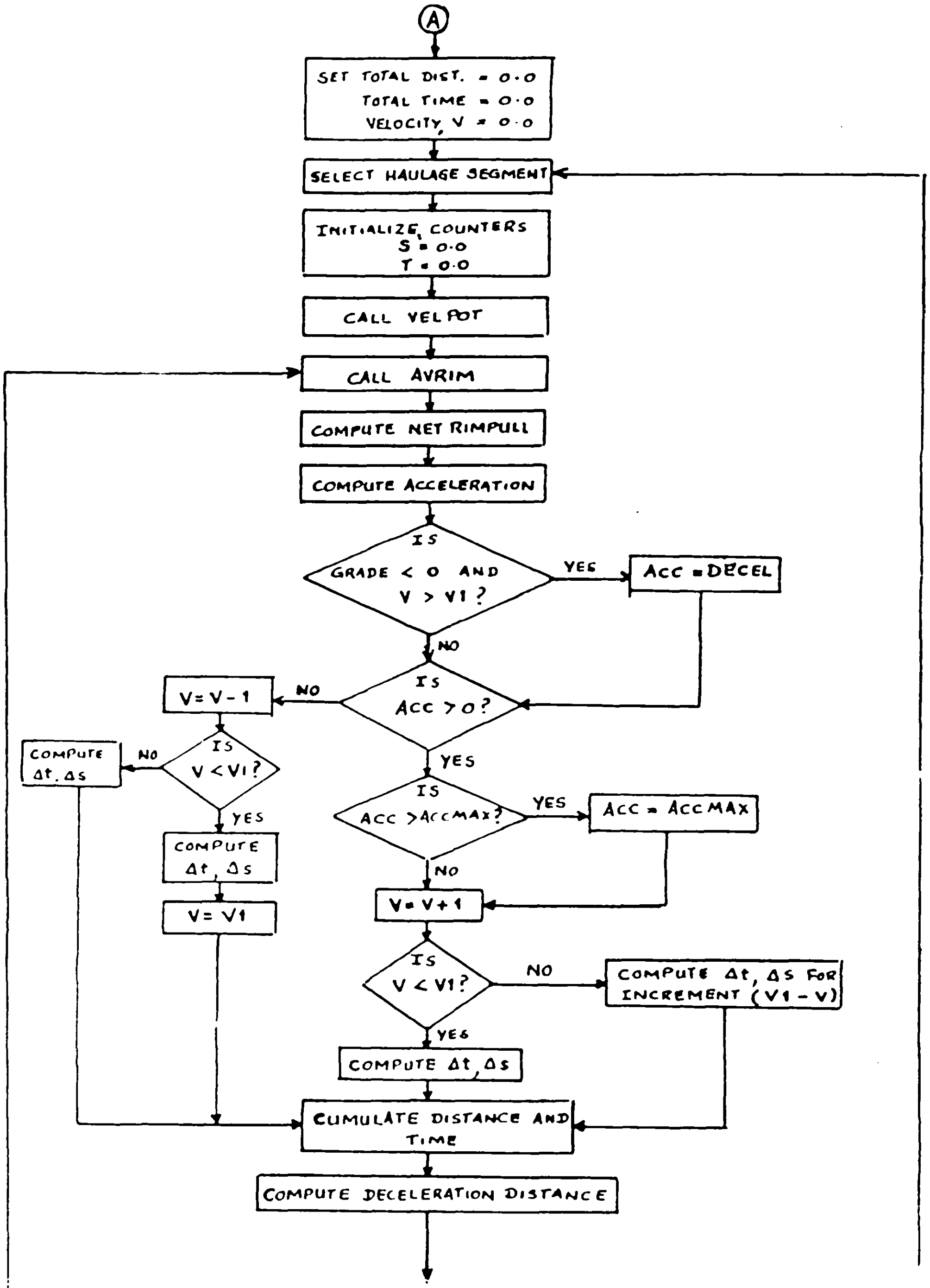

FIG. 4.12 CONTINUED

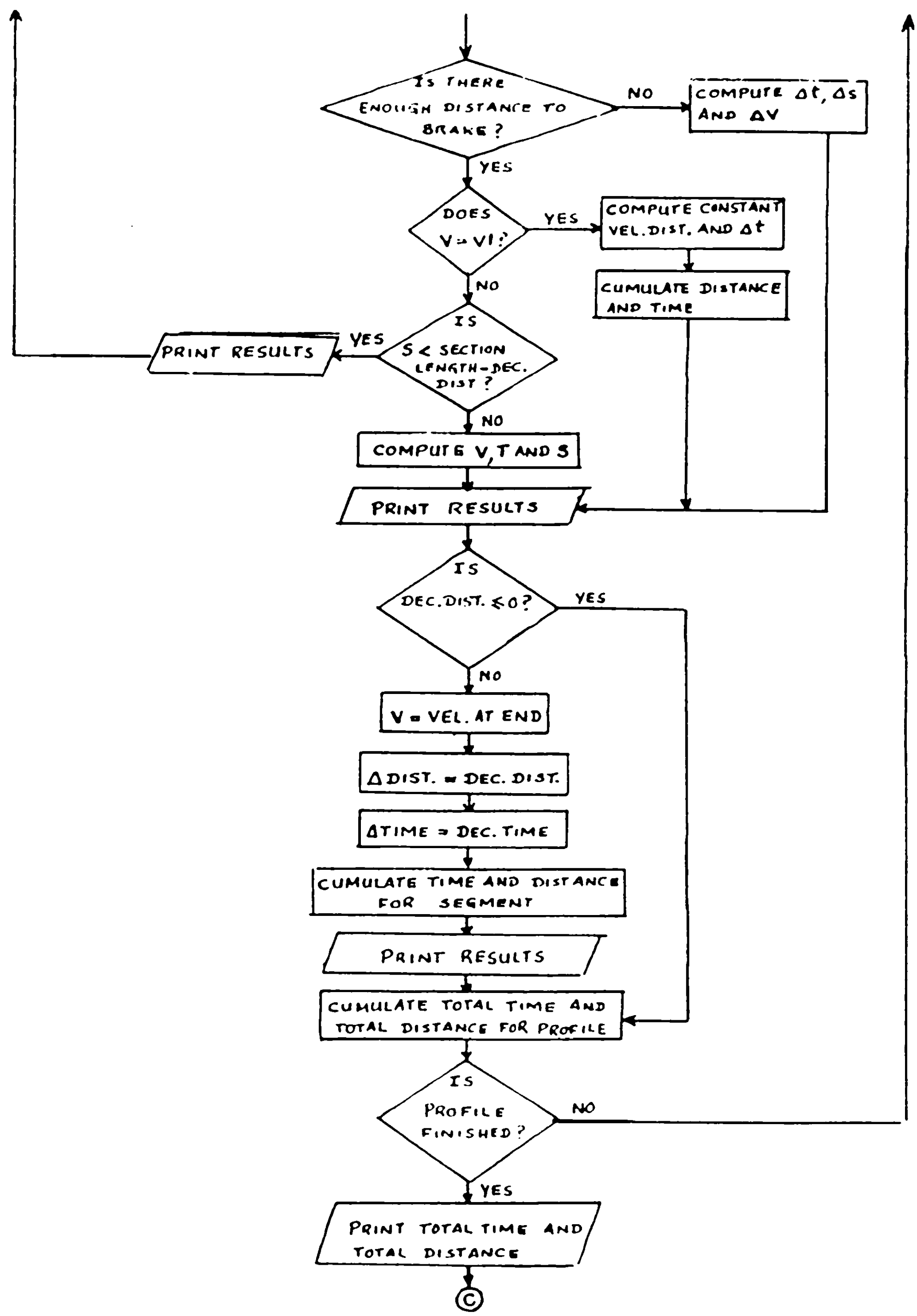

FIG. 4.12 CONTINUED

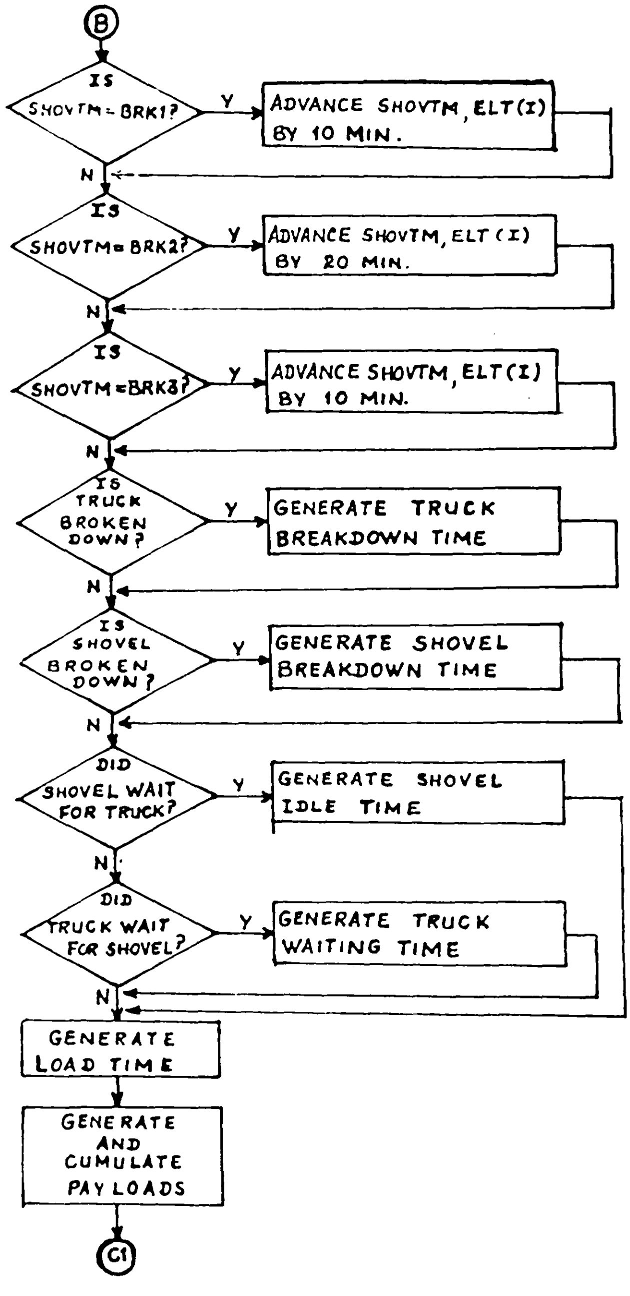

FIG 4.12. CONTINUED

PREDICTED HAUL PERFORMANCE FOR 85.00 TON TRUCK
VELOCITY LIMITED TO = 30.00 MPH

HAUL ROAD SPECIFICATION PRINTOUTS

NO. OF SECTIONS= 10   INPUT DATA

SECTION	LENGTH (FT)	GRADE (%)	ROLL.RES. (%)	MAX.VEL.FOR SECTION (MPH)	VEL.REQD. AT END (MPH)
1	800.0	.0	4.0	8.00	8.00
2	4000.0	8.5	3.0	30.00	30.00
3	5000.0	9.5	3.0	30.00	30.00
4	5600.0	7.0	3.0	30.00	30.00
5	1000.0	.0	6.0	30.00	.00
6	1000.0	.0	6.0	10.00	10.00
7	5600.0	-7.0	3.0	20.00	20.00
8	5000.0	-9.5	3.0	17.50	17.50
9	4000.0	-8.5	3.0	18.50	18.50
10	800.0	.0	4.0	18.50	.00

SECTION = 1   DISTANCE = 800.0FT

VELOCITY (MPH)	AVAIL. RIMPULL (LBS)	NET RIMPULL (LBS)	ACCEL. RATE (MPHPS)	TIME (SEC)	CUM. TIME (SEC)	DIST (FT)	CUM. DIST (FT)
1.00	108500.00	96868.00	3.00	.33	.33	.24	.24
2.00	96000.00	84368.00	3.00	.33	.67	.73	.98
3.00	82000.00	70368.00	3.00	.33	1.00	1.22	2.20
4.00	72000.00	60368.00	3.00	.33	1.33	1.71	3.91
5.00	66000.00	54368.00	3.00	.33	1.67	2.20	6.11
6.00	52800.00	41168.00	3.00	.33	2.00	2.69	8.80
7.00	44000.00	32368.00	2.44	.41	2.41	3.90	12.70

FIG. 4.13 SAMPLE OUTPUT FROM PROGRAM MONTE

8.00	37714.29	26082.29	1.97	.51	2.92	5.58	18.28
8.00	37714.29	26082.29	1.97	66.65	69.57	781.72	800.00
8.00	37714.29	26082.29	1.97	.00	69.57	.00	800.00

SECTION = 2 DISTANCE = 4000.0FT

VELOCITY (MPH)	AVAIL. RIMPULL (LBS)	NET RIMPULL (LBS)	ACCEL. RATE (MPHPS)	TIME (SEC)	CUM. TIME (SEC)	DIST (FT)	CUM. DIST (FT)
7.89	33000.00	-442.00	-.03	3.17	3.17	36.92	36.92
7.89	33000.00	-442.00	-.03	342.44	345.61	3963.08	4000.00
7.89	33000.00	-442.00	-.03	.00	345.61	.00	4000.00

SECTION = 3 DISTANCE = 5000.0FT

VELOCITY (MPH)	AVAIL. RIMPULL (LBS)	NET RIMPULL (LBS)	ACCEL. RATE (MPHPS)	TIME (SEC)	CUM. TIME (SEC)	DIST (FT)	CUM. DIST (FT)
7.26	33442.00	-2908.00	-.22	2.88	2.88	31.97	31.97
7.26	33442.00	-2908.00	-.22	466.61	469.48	4968.03	5000.00
7.26	33442.00	-2908.00	-.22	.00	469.48	.00	5000.00

SECTION = 4 DISTANCE = 5600.0FT

VELOCITY (MPH)	AVAIL. RIMPULL (LBS)	NET RIMPULL (LBS)	ACCEL. RATE (MPHPS)	TIME (SEC)	CUM. TIME (SEC)	DIST (FT)	CUM. DIST (FT)
8.26	36350.00	7270.00	.55	1.82	1.82	20.74	20.74
9.08	31950.72	2870.72	.22	3.76	5.59	47.85	68.59
9.08	31950.72	2870.72	.22	415.62	421.20	5531.41	5600.00
9.08	31950.72	2870.72	.22	.00	421.20	.00	5600.00

SECTION = 5 DISTANCE = 1000.0FT

VELOCITY (MPH)	AVAIL. RIMPULL (LBS)	NET RIMPULL (LBS)	ACCEL. RATE (MPHPS)	TIME (SEC)	CUM. TIME (SEC)	DIST (FT)	CUM. DIST (FT)
10.08	29080.00	11632.00	.88	1.14	1.14	15.99	15.99
11.08	26194.62	8746.62	.66	1.51	2.65	23.49	39.48
12.08	23830.15	6382.15	.48	2.08	4.73	35.24	74.72
13.08	21857.19	4409.19	.33	3.00	7.73	55.41	130.12
14.08	20185.95	2737.95	.21	4.84	12.57	96.32	226.44
15.08	18752.13	1304.13	.10	10.16	22.73	217.11	443.56
15.13	17508.48	60.48	.00	11.45	34.18	253.52	697.08
15.13	17508.48	60.48	.00	11.13	45.32	246.99	944.06
.00	17508.48	60.48	.00	5.04	50.36	55.94	1000.00

SECTION = 6 DISTANCE = 1000.0FT

VELOCITY (MPH)	AVAIL. RIMPULL (LBS)	NET RIMPULL (LBS)	ACCEL. RATE (MPHPS)	TIME (SEC)	CUM. TIME (SEC)	DIST (FT)	CUM. DIST (FT)
1.00	108500.00	101252.00	3.00	.33	.33	.24	.24
2.00	96000.00	88752.00	3.00	.33	.67	.73	.98
3.00	82000.00	74752.00	3.00	.33	1.00	1.22	2.20
4.00	72000.00	64752.00	3.00	.33	1.33	1.71	3.91
5.00	66000.00	58752.00	3.00	.33	1.67	2.20	6.11
6.00	52800.00	45552.00	3.00	.33	2.00	2.69	8.80
7.00	44000.00	36752.00	3.00	.33	2.33	3.18	11.97
8.00	37714.29	30466.29	3.00	.33	2.67	3.67	15.64
9.00	33000.00	25752.00	3.00	.33	3.00	4.15	19.79
10.00	29333.33	22085.33	3.00	.33	3.33	4.64	24.43

10.00	29333.33	22085.33	3.00	66.55	69.88	975.57	1000.00
10.00	29333.33	22085.33	3.00	.00	69.88	.00	1000.00

SECTION = 7 DISTANCE = 5600.0FT

VELOCITY (MPH)	AVAIL. RIMPULL (LBS)	NET RIMPULL (LBS)	ACCEL. RATE (MPHPS)	TIME (SEC)	CUM. TIME (SEC)	DIST (FT)	CUM. DIST (FT)
11.00	26400.00	31232.00	3.00	.33	.33	5.13	5.13
12.00	24000.00	28832.00	3.00	.33	.67	5.62	10.75
13.00	22000.00	26832.00	3.00	.33	1.00	6.11	16.86
14.00	20307.69	25139.69	3.00	.33	1.33	6.60	23.46
15.00	18857.14	23689.14	3.00	.33	1.67	7.09	30.54
16.00	17600.00	22432.00	3.00	.33	2.00	7.57	38.12
17.00	16500.00	21332.00	3.00	.33	2.33	8.06	46.18
18.00	15529.41	20361.41	3.00	.33	2.67	8.55	54.73
19.00	14666.67	19498.67	3.00	.33	3.00	9.04	63.77
20.00	13894.74	18726.74	3.00	.33	3.33	9.53	73.30
20.00	13894.74	18726.74	3.00	188.50	191.83	5526.70	5600.00
20.00	13894.74	18726.74	3.00	.00	191.83	.00	5600.00

SECTION = 8 DISTANCE = 5000.0FT

VELOCITY (MPH)	AVAIL. RIMPULL (LBS)	NET RIMPULL (LBS)	ACCEL. RATE (MPHPS)	TIME (SEC)	CUM. TIME (SEC)	DIST (FT)	CUM. DIST (FT)
19.00	13200.00	21052.00	-3.00	.33	.33	9.53	9.53
18.00	13894.74	21746.74	-3.00	.33	.67	9.04	18.57
17.50	14666.67	22518.67	-3.00	.17	.83	4.34	22.91
17.50	14666.67	22518.67	-3.00	194.00	194.83	4977.09	5000.00
17.50	14666.67	22518.67	-3.00	.00	194.83	.00	5000.00

SECTION = 9 DISTANCE = 4000.0FT

VELOCITY (MPH)	AVAIL. RIMPULL (LBS)	NET RIMPULL (LBS)	ACCEL. RATE (MPHPS)	TIME (SEC)	CUM. TIME (SEC)	DIST (FT)	CUM. DIST (FT)
18.50	15085.71	21729.71	3.00	.33	.33	8.80	8.80
18.50	15085.71	21729.71	3.00	147.16	147.50	3991.20	4000.00
18.50	15085.71	21729.71	3.00	.00	147.50	.00	4000.00

SECTION = 10 DISTANCE = 800.0FT

VELOCITY (MPH)	AVAIL. RIMPULL (LBS)	NET RIMPULL (LBS)	ACCEL. RATE (MPHPS)	TIME (SEC)	CUM. TIME (SEC)	DIST (FT)	CUM. DIST (FT)
18.50	14270.27	9438.27	1.71	.00	.00	.00	.00
18.50	14270.27	9438.27	1.71	26.41	26.41	716.38	716.38
.00	14270.27	9438.27	1.71	6.17	32.58	83.62	800.00

TOTAL TRAVEL TIME FOR PROFILE= 1992.85 SEC
TOTAL DISTANCE TRAVELLED= 32800.00 FT

OVERALL SHIFT STATISTICS

NO. OF TRUCKS USED = 1
RATED PAYLOAD OF TRUCK,TONS = 85.00
NO. OF SHOVEL USED = 1
DIPPER CAPACITY,CU.YD = 10.00
TOTAL DISTANCE TRAVELLED(2 WAYS),FT= 32800.00
TOTAL TIME TRAVELLED,SEC = 1992.85

SHIFT NO.	TOT.TONS MOVED (TONS)	TOT.WAITING TIME TRUCK (SEC)	TOT.WAITING TIME SHOVEL (SEC)	TOT.BREAKDOWN TRUCK (SEC)	TOT.BREAKDOWN SHOVEL (SEC)	TOT.COST PER TON ($/TON)
1	990.14	.00	22933.87	.00	.00	2.685
2	993.58	.00	22918.98	.00	.00	2.676
3	980.73	.00	22916.89	.00	.00	2.711
4	992.55	.00	22929.45	.00	.00	2.679
5	981.66	.00	22914.15	.00	.00	2.708
6	985.14	.00	22922.34	.00	.00	2.699
7	905.07	.00	20888.52	.00	3217.40	2.937
8	989.62	.00	22924.85	.00	.00	2.686
9	901.44	.00	20879.54	.00	.00	2.949
10	979.29	.00	22937.48	.00	.00	2.715
11	892.64	.00	20878.40	.00	2811.85	2.978
12	988.33	.00	22932.45	.00	.00	2.690
13	985.52	.00	22926.06	.00	.00	2.698
14	990.85	.00	22935.51	.00	.00	2.683
15	993.16	.00	22915.13	.00	.00	2.677

AVERAGE TOTAL TONNAGE,TONS = 969.98
AVERAGE TOTAL COSTS PER TON,$/TON = 2.745

OVERALL SHIFT STATISTICS

NO. OF TRUCKS USED = 2
RATED PAYLOAD OF TRUCK,TONS = 85.00
NO. OF SHOVEL USED = 1
DIPPER CAPACITY,CU.YD = 10.00
TOTAL DISTANCE TRAVELLED(2 WAYS),FT= 32800.00
TOTAL TIME TRAVELLED,SEC = 1992.85

SHIFT NO.	TOT.TONS MOVED (TONS)	TOT.WAITING TIME TRUCK (SEC)	SHOVEL (SEC)	TOT.BREAKDOWN TRUCK (SEC)	SHOVEL (SEC)	TOT.COST PER TON ($/TON)
1	1889.15	1653.88	17210.21	.00	3396.04	1.914
2	1976.85	474.93	19316.72	.00	.00	1.830
3	1805.09	2572.48	18217.79	.00	3080.18	2.004
4	1899.64	214.81	21145.98	3730.27	.00	1.904
5	1983.61	1099.97	20154.72	.00	.00	1.823
6	1626.35	1865.41	16436.82	3210.76	3436.23	2.224
7	1976.98	1106.52	20074.05	.00	.00	1.829
8	1889.34	1462.87	19967.37	.00	3807.11	1.914
9	1725.55	3610.93	16661.41	.00	3211.38	2.096
10	1699.82	4140.86	17099.14	.00	3155.13	2.128
11	1799.01	3632.15	17419.63	.00	2029.43	2.010
12	1980.55	146.68	19456.90	.00	.00	1.826
13	1798.29	2021.10	18675.72	2474.09	.00	2.011
14	1974.91	354.91	19685.64	.00	.00	1.831
15	1906.20	818.09	19856.82	3517.76	.00	1.897

AVERAGE TOTAL TONNAGE,TONS = 1862.09
AVERAGE TOTAL COSTS PER TON,$/TON = 1.949

OVERALL SHIFT STATISTICS

NO. OF TRUCKS USED	=	3
RATED PAYLOAD OF TRUCK,TONS	=	85.00
NO. OF SHOVEL USED	=	1
DIPPER CAPACITY,CU.YD	=	10.00
TOTAL DISTANCE TRAVELLED(2 WAYS),FT	=	32800.00
TOTAL TIME TRAVELLED,SEC	=	1992.85

SHIFT NO.	TOT.TONS MOVED (TONS)	TOT.WAITING TIME TRUCK (SEC)	TOT.WAITING TIME SHOVEL (SEC)	TOT.BREAKDOWN TRUCK (SEC)	TOT.BREAKDOWN SHOVEL (SEC)	TOT.COST PER TON ($/TON)
1	2960.42	1323.33	18846.51	2822.99	.00	1.545
2	2964.42	993.41	16694.09	.00	.00	1.543
3	3059.89	1296.32	18531.93	.00	.00	1.495
4	2684.74	1128.65	15604.88	.00	3208.11	1.704
5	2874.16	582.67	17716.66	3819.88	.00	1.592
6	2973.95	3158.24	17567.20	.00	.00	1.538
7	2800.39	1124.23	17058.41	3593.04	.00	1.634
8	2956.10	1510.60	17351.40	.00	.00	1.548
9	2701.94	6795.37	14861.71	.00	4030.85	1.693
10	2974.11	1518.92	17151.06	.00	.00	1.538
11	2969.74	1420.69	17331.72	.00	.00	1.540
12	2701.75	3883.26	15934.72	2575.64	2390.99	1.693
13	2889.00	4115.06	15337.84	.00	3132.80	1.584
14	2972.41	1498.12	17237.07	.00	.00	1.539
15	2780.40	3762.86	16536.24	3186.87	2419.90	1.645

AVERAGE TOTAL TONNAGE,TONS = 2884.23

AVERAGE TOTAL COSTS PER TON,$/TON = 1.589

OVERALL SHIFT STATISTICS

NO. OF TRUCKS USED = 4
RATED PAYLOAD OF TRUCK,TONS = 85.00
NO. OF SHOVEL USED = 1
DIPPER CAPACITY,CU.YD = 10.00
TOTAL DISTANCE TRAVELLED(2 WAYS),FT= 32800.00
TOTAL TIME TRAVELLED,SEC = 1992.85

SHIFT NO.	TOT.TONS MOVED (TONS)	TOT.WAITING TIME TRUCK (SEC)	SHOVEL (SEC)	TOT.BREAKDOWN TRUCK (SEC)	SHOVEL (SEC)	TOT.COST PER TON ($/TON)
1	3855.68	2615.71	15066.63	2341.20	.00	1.435
2	3969.00	3206.06	14633.68	.00	.00	1.394
3	3772.09	2720.38	14731.42	.00	2428.51	1.467
4	3884.58	3073.94	15196.74	2477.42	.00	1.424
5	3874.33	2082.61	16336.06	3136.16	3778.05	1.428
6	3862.08	7288.76	13416.57	.00	3075.52	1.433
7	4051.69	2057.93	15175.02	.00	.00	1.366
8	3519.87	9033.14	12881.03	3238.13	3564.67	1.572
9	3634.42	1596.05	14062.38	3388.63	3762.51	1.522
10	4046.41	2237.08	15816.79	.00	.00	1.367
11	3781.87	7443.96	12674.04	.00	3159.12	1.463
12	3866.24	2913.43	14834.15	2061.98	.00	1.431
13	3875.73	5616.68	12737.42	.00	2676.97	1.428
14	3955.47	2497.44	15614.69	3344.67	.00	1.399
15	3965.25	1881.51	15539.94	2332.87	.00	1.395

AVERAGE TOTAL TONNAGE,TONS = 3860.98
AVERAGE TOTAL COSTS PER TON,$/TON = 1.435

OVERALL SHIFT STATISTICS

NO. OF TRUCKS USED	=	5
RATED PAYLOAD OF TRUCK, TONS	=	85.00
NO. OF SHOVEL USED	=	1
DIPPER CAPACITY, CU.YD	=	10.00
TOTAL DISTANCE TRAVELLED(2 WAYS), FT	=	32800.00
TOTAL TIME TRAVELLED, SEC	=	1992.85

SHIFT NO.	TOT.TONS MOVED (TONS)	TOT.WAITING TIME TRUCK (SEC)	SHOVEL (SEC)	TOT.BREAKDOWN TRUCK (SEC)	SHOVEL (SEC)	TOT.COST PER TON ($/TON)
1	4572.77	14876.48	10906.51	.00	3541.57	1.420
2	4863.01	3699.90	12394.74	3005.45	.00	1.335
3	4227.79	15401.03	11282.26	3661.28	2560.56	1.535
4	4780.77	9750.78	11512.65	.00	2639.78	1.358
5	4510.73	13167.76	10954.54	.00	3274.94	1.439
6	4744.73	11179.12	11613.81	.00	2697.80	1.368
7	4338.83	18165.25	10009.07	.00	4071.74	1.496
8	4711.89	10757.18	10569.64	.00	2830.11	1.378
9	4791.06	9108.87	11709.25	3398.62	3028.18	1.355
10	4949.85	3186.84	13263.88	2070.26	.00	1.311
11	4581.43	10597.69	11517.91	2353.28	2852.91	1.417
12	4850.68	4346.52	12935.89	2845.47	.00	1.338
13	4513.27	11496.74	11038.60	3611.30	3064.87	1.438
14	4570.34	12379.87	11180.27	.00	3117.06	1.420
15	4586.10	16029.60	12016.94	.00	2519.41	1.415

AVERAGE TOTAL TONNAGE, TONS = 4639.55

AVERAGE TOTAL COSTS PER TON, $/TON = 1.402

OVERALL SHIFT STATISTICS

NO. OF TRUCKS USED = 6
RATED PAYLOAD OF TRUCK, TONS = 85.00
NO. OF SHOVEL USED = 1
DIPPER CAPACITY, CU.YD = 10.00
TOTAL DISTANCE TRAVELLED (2 WAYS), FT = 32800.00
TOTAL TIME TRAVELLED, SEC = 1992.85

SHIFT NO.	TOT.TONS MOVED (TONS)	TOT.WAITING TIME TRUCK (SEC)	SHOVEL (SEC)	TOT.BREAKDOWN TRUCK (SEC)	SHOVEL (SEC)	TOT.COST PER TON ($/TON)
1	5032.97	19440.51	9143.81	6003.95	3226.20	1.480
2	4746.66	16123.64	7462.19	3423.77	2614.09	1.569
3	5125.67	22345.47	9356.71	2420.47	3464.33	1.453
4	5030.37	21703.41	8933.96	4950.70	3764.14	1.481
5	5853.70	4263.54	10671.54	3878.90	2337.05	1.273
6	5753.48	6253.51	11068.88	6723.58	.00	1.295
7	4959.43	15817.63	8108.04	3601.78	2930.09	1.502
8	5388.86	12288.74	9559.35	6551.29	2873.13	1.382
9	5862.06	6422.55	10385.00	4126.99	.00	1.271
10	5130.49	19893.45	9903.96	6162.32	3258.58	1.452
11	5134.91	20899.08	8931.24	5341.11	3541.62	1.451
12	5592.21	7582.42	10874.76	4982.67	2174.53	1.332
13	5049.76	20684.33	9834.04	5807.71	3564.78	1.475
14	5195.81	6853.75	10532.67	5309.76	3605.01	1.434
15	5234.85	14866.07	8633.91	6566.99	3255.92	1.423

AVERAGE TOTAL TONNAGE, TONS = 5272.75
AVERAGE TOTAL COSTS PER TON, $/TON = 1.418

OVERALL SHIFT STATISTICS

NO. OF TRUCKS USED	=	7
RATED PAYLOAD OF TRUCK, TONS	=	85.00
NO. OF SHOVEL USED	=	1
DIPPER CAPACITY, CU.YD	=	10.00
TOTAL DISTANCE TRAVELLED (2 WAYS), FT	=	32800.00
TOTAL TIME TRAVELLED, SEC	=	1992.85

SHIFT NO.	TOT.TONS MOVED (TONS)	TOT.WAITING TIME TRUCK (SEC)	TOT.WAITING TIME SHOVEL (SEC)	TOT.BREAKDOWN TRUCK (SEC)	TOT.BREAKDOWN SHOVEL (SEC)	TOT.COST PER TON ($/TON)
1	6124.85	22075.34	7266.81	5667.29	2820.56	1.373
2	6031.54	23625.52	7450.94	5144.17	2954.11	1.394
3	6308.91	22523.44	7922.04	6347.42	2969.20	1.333
4	6222.82	24762.14	6951.09	2700.40	3422.45	1.351
5	5949.31	24980.97	6847.98	6175.45	3264.77	1.413
6	6492.70	19205.93	7925.23	2162.25	2498.38	1.295
7	5867.00	24634.82	7397.75	5562.47	3283.49	1.433
8	6219.96	19260.72	7916.76	5268.05	2561.18	1.352
9	5862.41	26663.64	7438.83	6190.37	3829.89	1.434
10	6183.95	18678.78	7030.06	7532.43	2669.34	1.360
11	5947.95	21678.04	6906.91	6301.73	3102.41	1.414
12	6208.34	18337.16	7789.47	6116.32	2270.09	1.354
13	5775.93	28913.77	7186.92	6120.90	3371.69	1.456
14	6771.45	7383.10	8094.71	6367.60	.00	1.242
15	6414.69	17641.89	6920.51	5300.39	2202.45	1.311

AVERAGE TOTAL TONNAGE, TONS = 6158.79

AVERAGE TOTAL COSTS PER TON, $/TON = 1.368

(2) Dumping time

Dumping time is considered to be exponentially distributed with the average value of 0.42 min. Although the simulator can accomodate dumping parameters for each truck type, it was assumed in this study that dumping time is independent of truck size.

(3) Truck Payloads

The payload would seem logically to follow the normal distribution. As mentioned in loading time estimation, the payloads for various truck sizes were also derived from the known distributions for 32 ton truck by increasing the parameters proportionately. Average load weight for the 32 ton truck is considered as 34 ton subject to variations of + or - 2 tons.

(4) Truck and Shovel Breakdown Times

These variables were all considered to be normally distributed about the mean, ranging from the mean minus two standard deviation to the mean plus three standard deviation in order to eliminate 2.14 % of the shortest times which could be generated from the continuous normal probability distribution.

Mean breakdown repair time for each truck and shovel is assumed at 50 min (3000 sec) throughout the shift and thus maintaining the machine availability at about 97% for all the trucks and 90% for the shovel.

4.3.2. RESULTS FROM PROGRAM MONTE

To test the program, the entire haul road is broken up into a number of sections each having its own grade and rolling resistance.The haul road condition is considered as long haul distance with Average rolling resistance (see table 5 in Appendix 1). Detail information is as follows:

TABLE 4.21 DETAIL INFORMATION OF HAUL ROAD

ROAD SECTION	LENGTH (FT.)	GRADE (%)	ROLLING RESISTANCE	REMARKS
1	800	0.0	4.0	Start of loaded haul in the pit
2	4000	+8.5	3.0)	
3	5000	+9.5	3.0)	Main haul
4	5600	+7.0	3.0)	= 14,600 FT
5	1000	0.0	6.0	End of loaded haul.
6	1000	0.0	6.0	Start of empty return
7	5600	-7.0	3.0	
8	5000	-9.5	3.0	
9	4000	-8.5	3.0	
10	800	0.0	4.0	End of empty return

After completing computation for all types of trucks with corresponding shovel sizes over a given haul road with a known values of truck performance characteristics, the deterministic part would give the travelling performance of a truck. The following table 4.22 shows the travelling times of the varrious types of trucks being considered in the study.

TABLE 4.22 TRAVELLING TIMES OF VARIOUS TYPES OF TRUCKS

TRUCK SIZE (ton)	HAUL TIME (min)	RETURN TIME (min)	TOTAL TRAVEL TIME (min)
85	22.60	10.61	33.21
100	23.79	10.61	34.40
120	26.56	10.61	37.17

These travel times are used with their corresponding truck size in the stochastic part when the travel times come in question. The program is run for each truck and shovel combination with a varying number of trucks over a series of shift. Performance summaries for some of the shovel-truck combinations are shown in tables 4.23 through 4.26.

TABLE 4.23 PERFORMANCE SUMMARY FOR 85 TON - 8 CU.YD.TRUCK-SHOVEL COMBINATION

No. of Trucks	Fleet Production (Tons / Shift)	Cost per Ton ($ / Ton)
1	961.31	2.42
2	1913.89	1.71
3	2919.54	1.45
4	3803.62	1.37
5	4674.59	1.33
6	5370.35	1.33
7	6315.02	1.28
8	6850.98	1.29
9	7404.25	1.31
10	7729.25	1.38
11	8208.53	1.43
12	8453.86	1.51

TABLE 4.24. PERFORMANCE SUMMARY FOR 85 TON - 12.5 CU.YD.COMBINATION

No. of Trucks	Fleet Production (Tons / Shift)	Cost / Ton ($ / Ton)
1	1078.17	3.11
2	2063.95	2.04
3	3059.99	1.68
4	4121.39	1.50
5	4807.24	1.45
6	5678.52	1.38
7	6655.73	1.33
8	7337.53	1.32
9	8110.82	1.32
10	8614.38	1.35
11	9738.82	1.35
12	10130.49	1.36
13	10703.71	1.36
14	10779.45	1.44
15	10861.66	1.52

TABLE 4.25 PERFORMANCE SUMMARY OF 100 TON - 10 CU.YD. COMBINATION

No. of Trucks	Fleet Production (Tons / Shift)	Cost / Ton ($ / Ton)
1	1084.55	2.70
2	2178.98	1.86
3	3328.95	1.57
4	4399.94	1.46
5	5372.85	1.41
6	6333.78	1.39
7	7141.32	1.40
8	8026.58	1.38
9	8674.64	1.42
10	9395.83	1.43
11	9595.85	1.53
12	9759.19	1.62
13	10154.31	1.67
14	10069.09	1.80
15	10287.72	1.88

TABLE 4.26 PERFORMANCE SUMMARY FOR 120 TON - 12.5 CU.YD. COMBINATION

No. of trucks	Fleet Production (Tons / Shift)	Cost / Ton ($ / Ton)
1	1191.85	2.88
2	2401.67	1.98
3	3642.48	1.67
4	4816.31	1.54
5	5952.05	1.47
6	7009.17	1.44
7	7930.21	1.44
8	9036.41	1.41
9	9862.44	1.43
10	10306.27	1.50
11	10966.67	1.53
12	11069.26	1.64
13	11147.48	1.75
14	11379.18	1.82
15	11631.72	1.90
16	11697.07	2.00

The following table 4.27 shows the summary of the best performance combinations from each shovel-truck combination.

TABLE 4.27 SUMMARY OF THE BEST PERFORMANCE COMBINATIONS

COMBINATION	NO. OF TRUCKS USED	FLEET PROD. (TONS/ SHF)	COST PER TON ($ / TON)
85T-8cu.yd.	7	6315.02	1.28
100T-8cu.yd.	6	6179.90	1.36
120T-8cu.yd.	6	6590.69	1.42
85T-10cu.yd.	8	7015.41	1.31
100T-10cu.yd.	8	8026.58	1.38
120T-10cu.yd.	7	7819.21	1.41
85T-12.5cu.yd.	9	8110.82	1.32
100T-12.5cu.yd.	9	9153.35	1.38
120T-12.5cu.yd.	8	9036.41	1.41

Since the maximum production output of the shovel-truck fleet is limited by the shovel potential, it is necessary to choose the larger shovel size if the larger production rate is desired. Theoretically, the shovel potential can be matched with truck fleet production by increasing the number of trucks in the fleet. However, in practice, more number of trucks have to be used than that of in theoretical case because of the production losses of the truck fleet due to the bunching effects of the trucks. Here comes the cost per ton of the material moved is of major importance since it is necessary to choose the optimum number of truck fleet with maximum production output and minimum cost.

According to the above table 4.27 it is noticed that in each shovel size category, depending on the size of truck, there is an optimum number of trucks at which the maximum output and minimum cost are achieved. It is observed that 8 cubic yard shovel is best suited for production rate of round about 6000 tons/Shift, whereas 10 cubic yard for about 7000 to 8000 tons/shift and 12.5 cubic yard for over 8000 to 9000 tons/shift. Fleet production per shift versus number of trucks for various shovel-truck combinations is shown in figure 4.14.

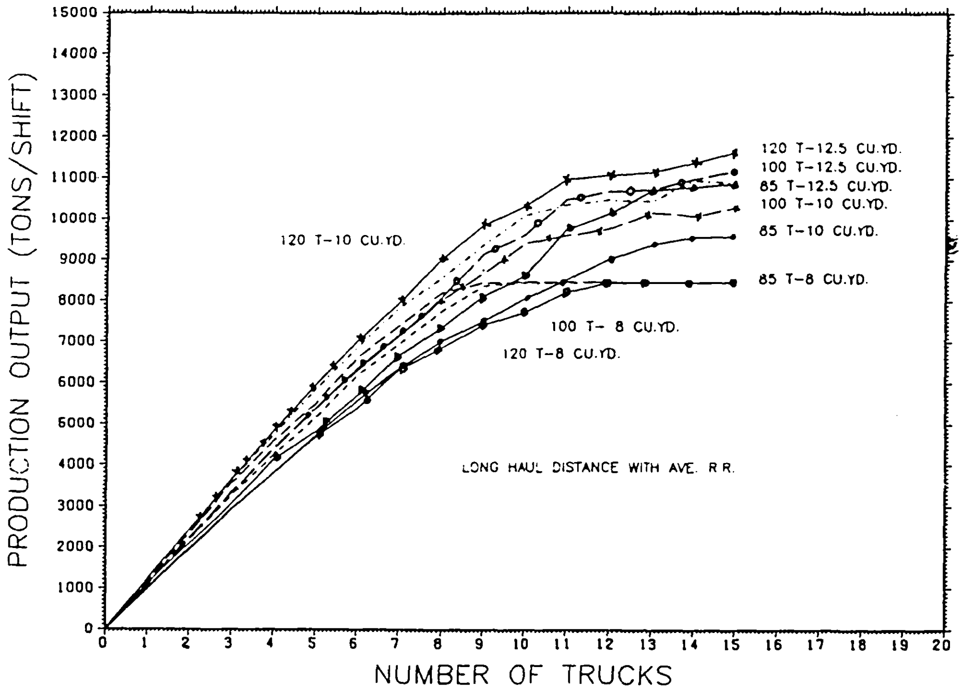

FIG. 4.14 PRODUCTION OUTPUT VERSUS NUMBER OF TRUCKS WITH VARIOUS SHOVEL-TRUCK COMBINATIONS

Although, the values shown in table 4.27 are the optimal values for each set of shovel-truck combination, it is still possible to find out the range of production rate at which the lower cost per ton are obtained compared to each other. This can be achieved by finding out the break-even points of the various combinations. Figure 4.15 shows the relationship of cost per ton versus number of trucks for each set of combination. The cost per ton values were carefully studied before drawing these curves and it is found that the 85 ton - 8 cubic yard , 85 ton - 12.5 cubic yard and 100 ton - 8 cubic yard are the only deciding combinations and therefore the cost per ton curves for 120 ton and its relating shovel sizes are not included in the figure.

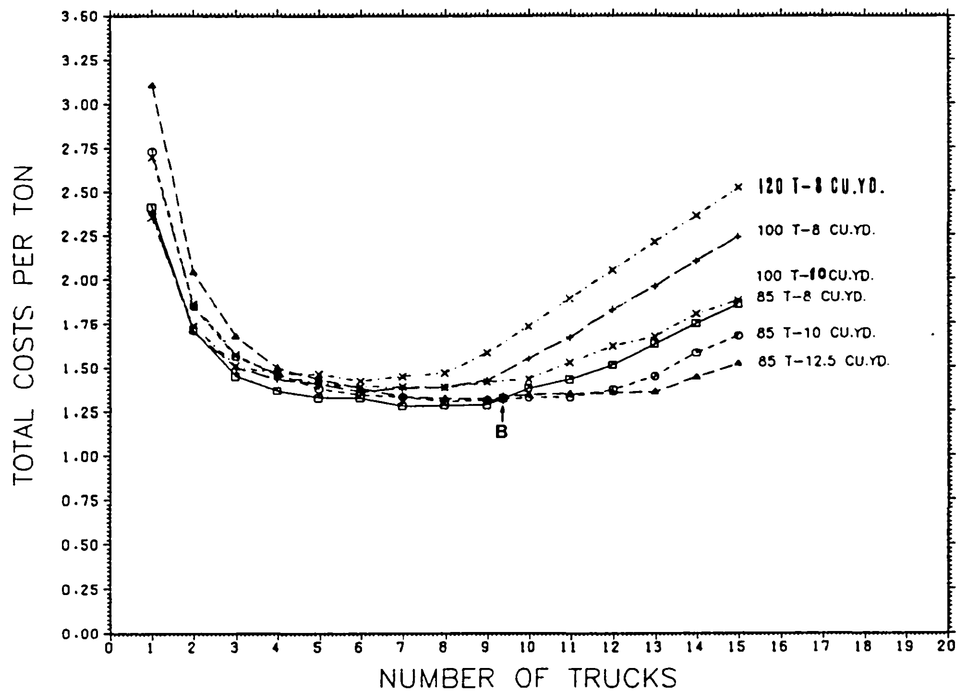

FIG. 4.15 TOTAL COSTS PER TON VERSUS NUMBER OF TRUCKS

According to the figure there are some break-even points, at which the costs per ton curves of various combinations intersect each other. However, the most outstanding breakeven point is obtained at the point where 85 t - 8 cu.yd. and 85 ton 12.5 cu.yd. combinations meet each other. This point is shown by symbol 'B ' in figure 4.15.

The point shows that the 85 ton - 8 cu.yd. shovel-truck combination would be best suited for conditions where 9 or less trucks (i.e. production rate of up to about 7400 tons per shift) are operating, while 85 ton - 12.5 cu.yd.

shovel-truck combination would be best suited for conditions where more than 9 trucks (i.e. production rate from over 7400 to about 10,800 tons/Shift). The combination 120 ton - 12.5 cu.yd. shovel-truck combination seems to be best suited for production of more than 11,000 tons/shift (Please see tables 4.23, 4.24 and 4.26 as well). However, further analysis should be carried out for larger production of more than 11000 tons/shift with larger size shovel-truck fleet combinations.

Further test on the program over the short haul distance, the other assumptions being unchanged, is also carried out and the results are shown in the tables 4.28 and 4.29 below. The distance for main haul length is 4500 ft plus 300 ft each for in-pit distance and the distance from dump point to main haul road.

TABLE 4.28 TRAVELLING TIMES FOR DIFFERENT TYPES OF TRUCKS

TRUCK SIZE (Ton)	HAUL TIME (min)	RETURN TIME (min)	TOTAL TRAVELLING TIME (min)
85	7.50	3.74	11.24
100	7.90	3.74	11.54
120	8.73	3.74	12.47

TABLE 4.29 SUMMARY OF THE BEST PERFORMANCE COMBINATIONS

COMBINATION	NO. OF TRUCKS USED	FLEET PROD. (TONS/SHIFT)	COST PER TON ($/TON)
85T-8cu.yd.	3	6528	0.65
100T-8cu.yd.	3	7323	0.67
120T-8cu.yd.	3	7789	0.69
85T-10cu.yd.	4	8536	0.65
100T-10cu.yd.	4	9487	0.68
120T-10cu.yd.	3	8384	0.68
85T-12.5cu.yd.	5	10626	0.65
100T-12.5cu.yd.	4	9526	0.71
120T-12.5cu.yd.	4	10646	0.70

Table 4.29 shows that for short haul distance, the 85 ton - 12.5 cubic yard seems to be the overall best performance - combination producing the maximum production of 10,626 tons/Shift with the lowest cost per ton of $ 0.65 / ton using 5 trucks.

Furthermore,the analysis of sensitivity on production and costs due to the following changes are carried out:

(1) changes in truck size

(2) changes in shovel dipper size

(3) changes in length of Haul

(4) changes in Rolling Resistances and

(5) changes in grade of haul road.

Sensitivities on production due to the above changes are shown in figures 4.16 through 4.20.

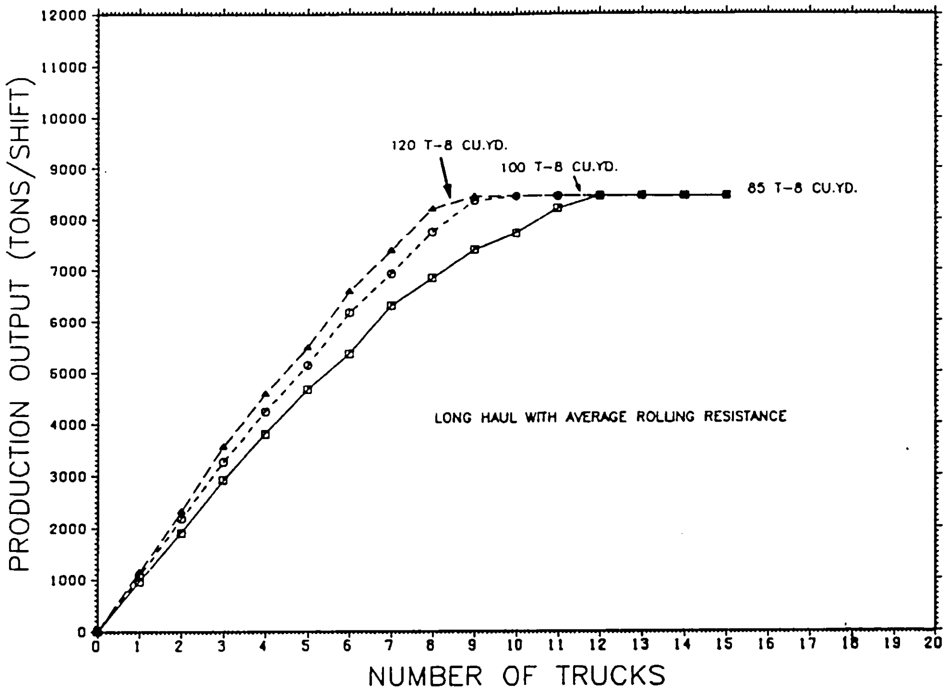

FIG. 4.16 SENSITIVITY ON PRODUCTION OUTPUT DUE TO CHANGE IN TRUCK SIZE

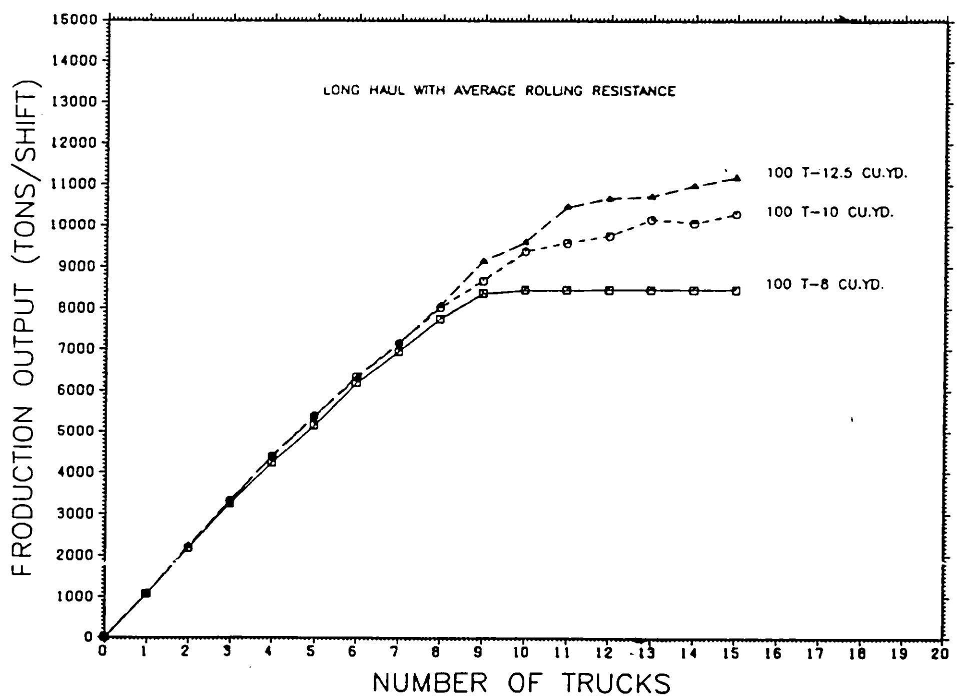

FIG. 4.17 SENSITIVITY ON PRODUCTION OUTPUT DUE TO CHANGE IN SHOVEL SIZE

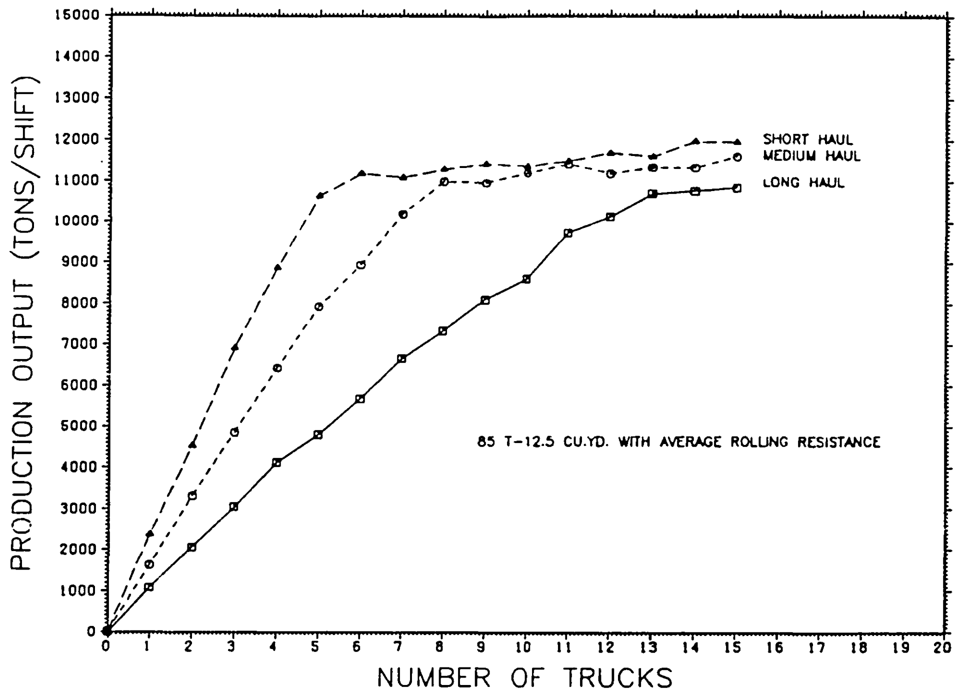

FIG. 4.18 SENSITIVITY ON PRODUCTION OUTPUT DUE TO CHANGE IN LENGTH OF HAUL ROAD

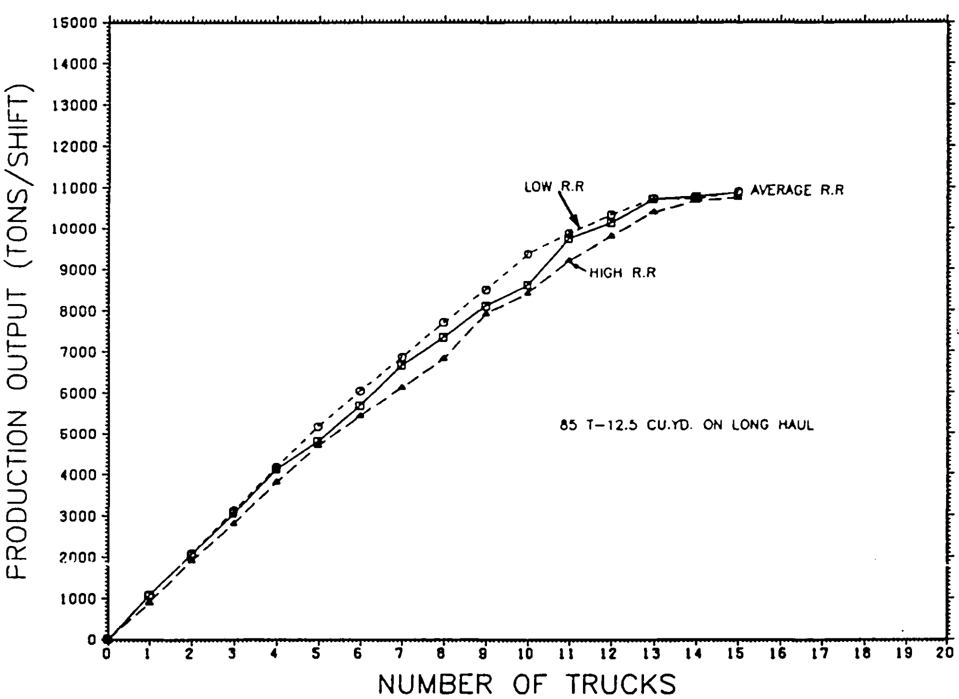

FIG. 4.19 SENSITIVITY ON PRODUCTION OUTPUT DUE TO CHANGE IN ROLLING RESISTANCE

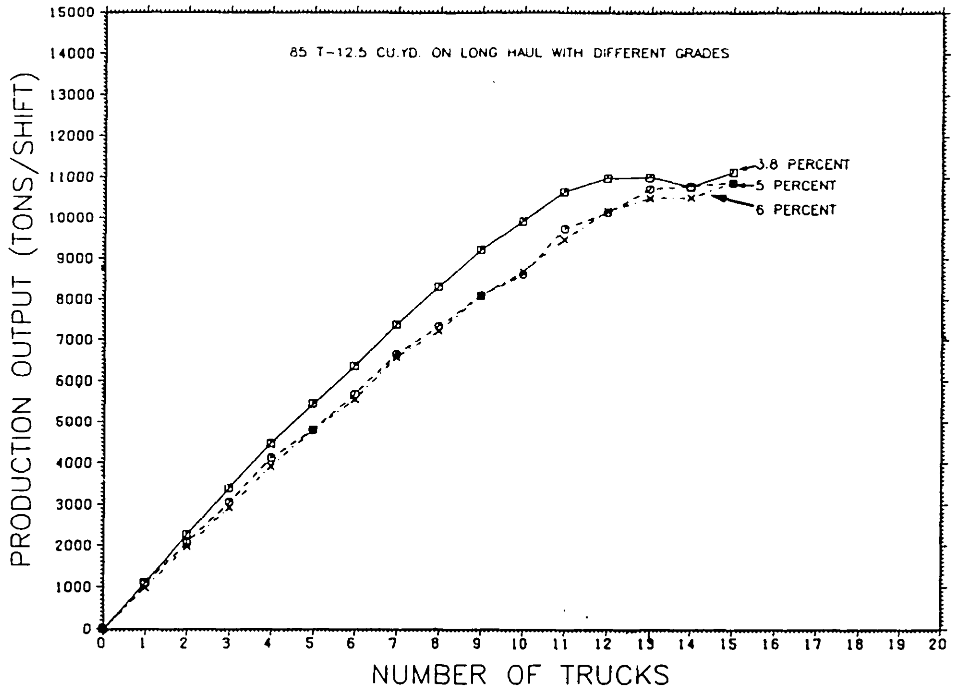

FIG. 4.20 SENSITIVITY ON PRODUCTION OUTPUT DUE TO CHANGE IN GRADES

Since the total production is limited by the shovel potential regardless of cost, the fleet production produced by changing the truck sizes cannot exceed the shovel potential as shown in figure 4.16. On the other hand, the fleet production (i.e. total production) increases, regardless of cost, as the shovel size increases . This is shown in figure 4.17. It is obvious that when larger production rates are required, it is better to use larger shovel rather than using the larger size trucks. On the other hand, one question is what size truck should be used if the required production is limited at a certain rate ? The answer to this question depends on several factors such as :

- Performance characteristics of the truck to be used (i.e. Rimpull - Grade- Speed Performance which in turn related to the Gross and Net vehicle weight (GVW and NVW) of the truck, engine efficiency, engine horsepower).
- Haul road conditions such as rolling resistance, grade resistances, length, bands, etc.
- Type of loading machine to be worked with
- Horsepower to weight ratio of the truck
- And management and supervision conditions.

In figure 4.21 the change in relative productivity, in per cent, due to the change in rolling resistance for the trucks over long and short haul is illustrated. It is observed that the effect of high rolling resistance is more pronounced for the larger trucks than for the smaller trucks and it is also noticed that the effect of change in rolling resistance is more pronounced for longer haul distance than for the short haul distance.

Values for rolling resistance for individual section are shown in table 5 in appendix 1 and the distances for main haul length are 14,600 ft for long haul and 4500 ft for short haul respectively.

Sensitivity on costs due to the changes in truck size and shovel size is shown in figures 4.22 and 4.23. It is observed that the costs increase as the truck size increases and the reverse is true as the shovel size increases. This is because the larger the shovel size the greater the output is produced and the number of trucks can be handled.

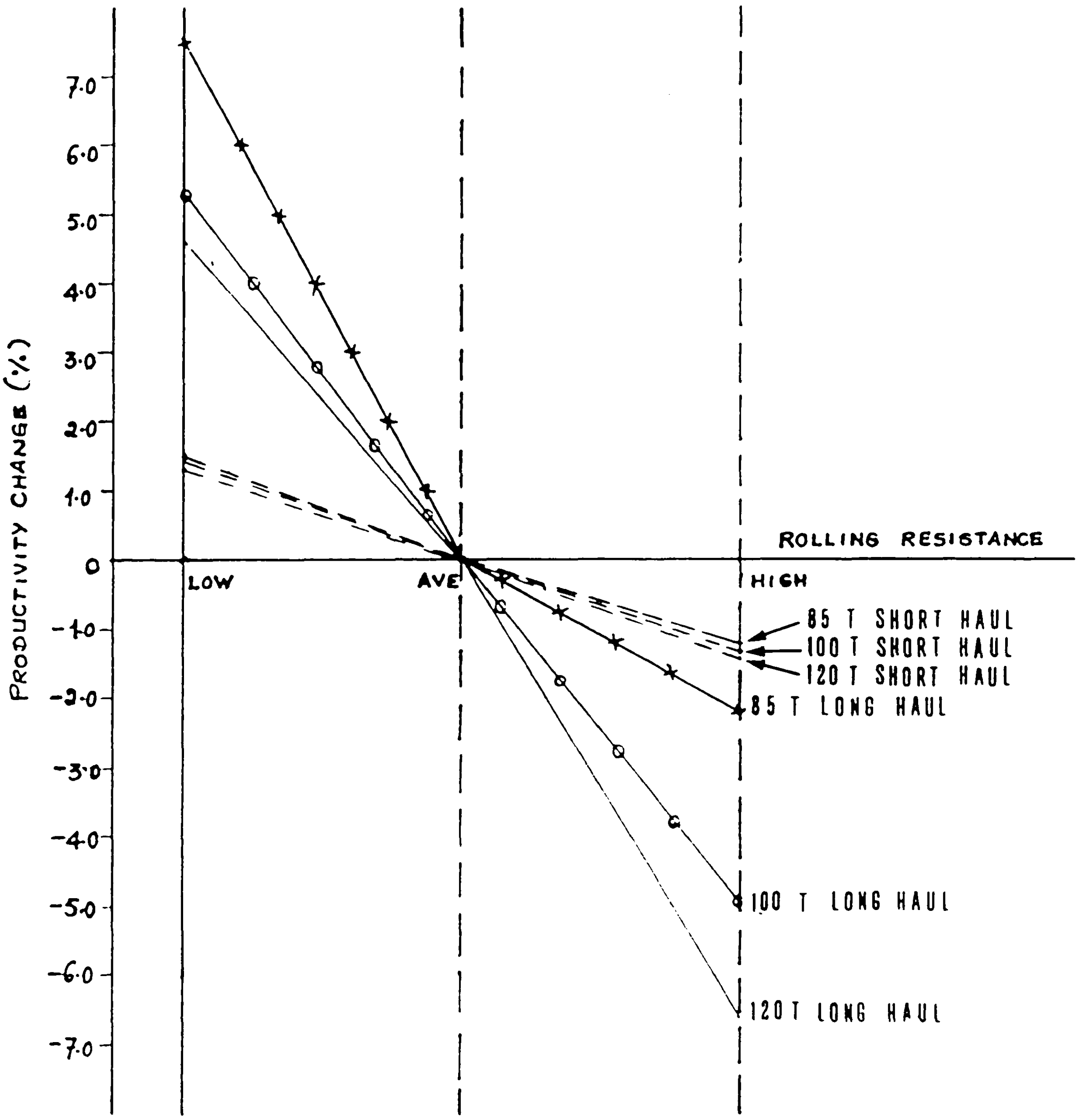

FIG. 4.21 RELATIVE PRODUCTIVITY CHANGE DUE TO CHANGE IN ROLLING RESISTANCE

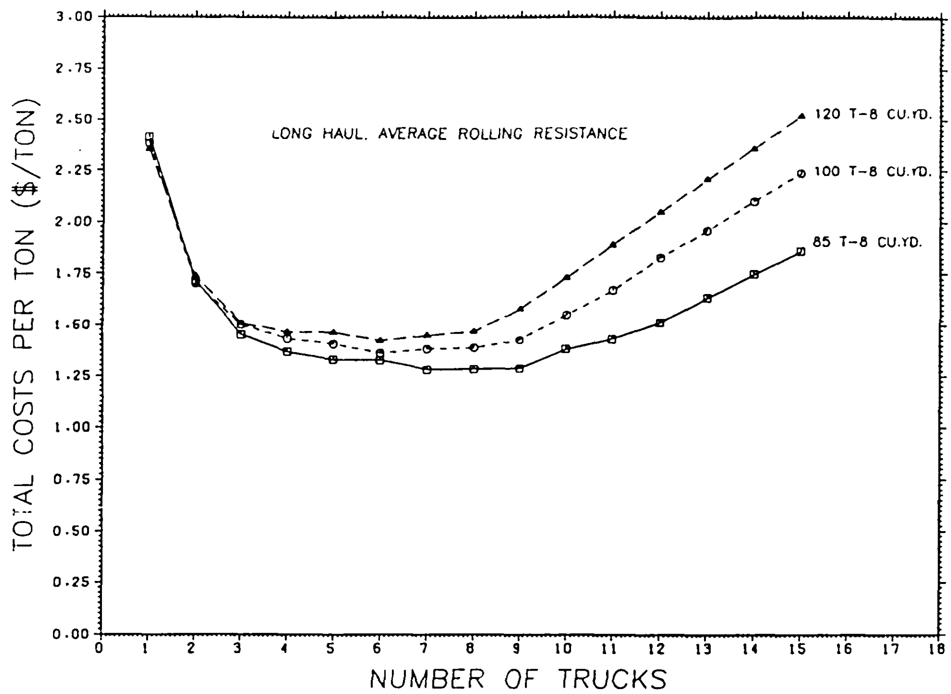

FIG. 4.22 SENSITIVITY ON COSTS DUE TO CHANGE IN TRUCK SIZE

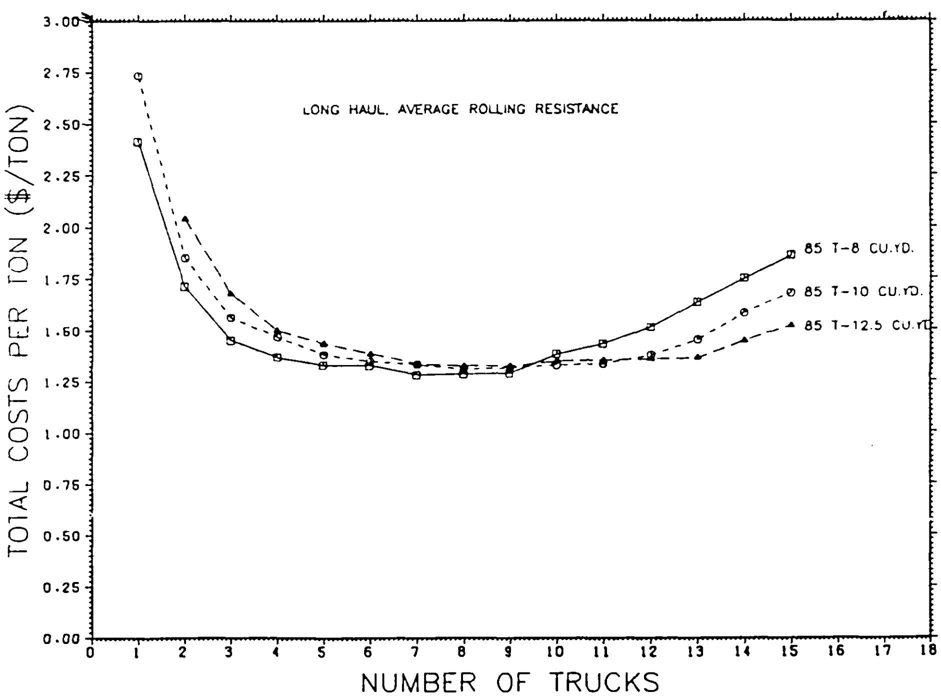

FIG. 4.23 SENSITIVITY ON COSTS DUE TO CHANGE IN SHOVEL SIZE

According to figure 4.23 it is observed that 85 ton truck with smaller 8 cubic yard shovel combination is suitable for only up to 9 trucks (i.e. smaller production) whereas 85 ton truck with larger 12.5 yard shovel combination is suitable for more than 9 trucks (i.e. larger production). However, both combination have their own optimum situation which is already mentioned previously.

Sensitivity on costs due to the changes in length of haul road, rolling resistance and grade is shown in figures 4.24, 4.25 and 4.26.

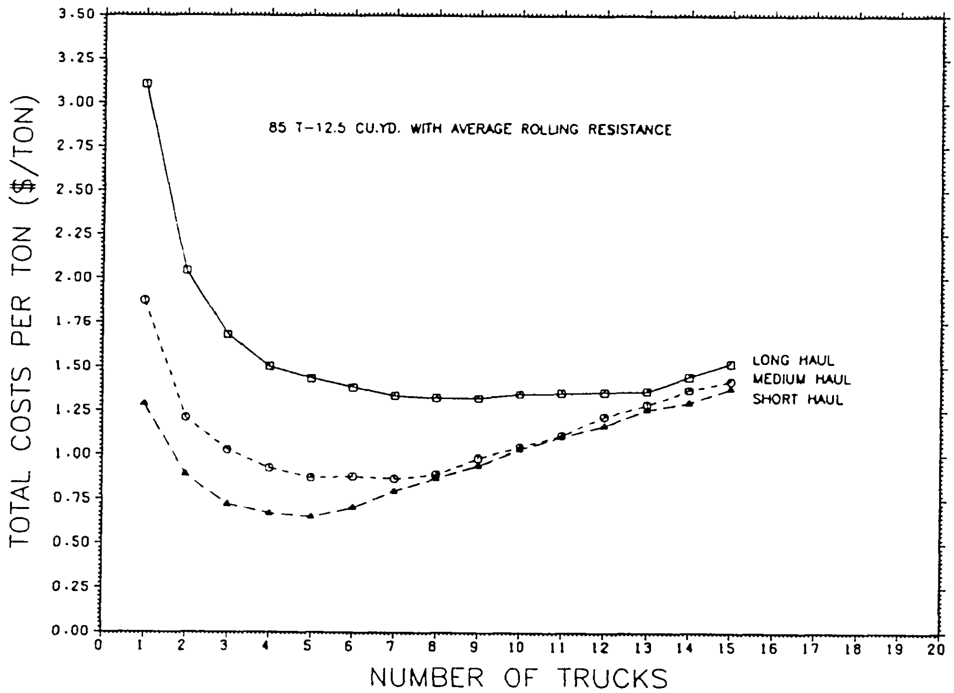

FIG. 4.24 SENSITIVITY ON COSTS DUE TO CHANGE IN HAUL LENGTH

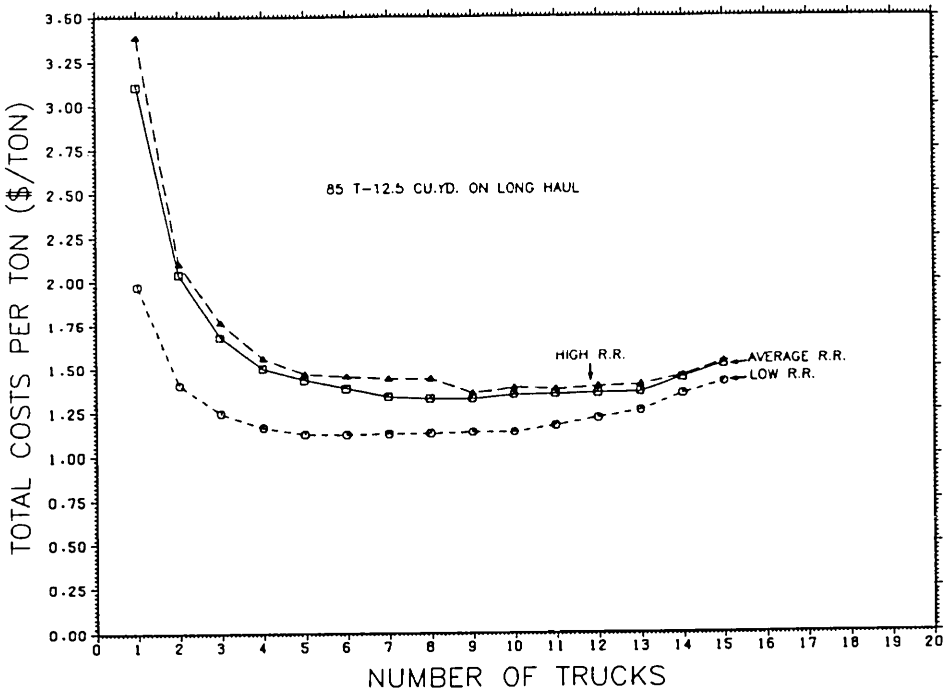

FIG. 4.25 SENSITIVITY ON COSTS DUE TO CHANGE IN ROLLING RESISTANCE

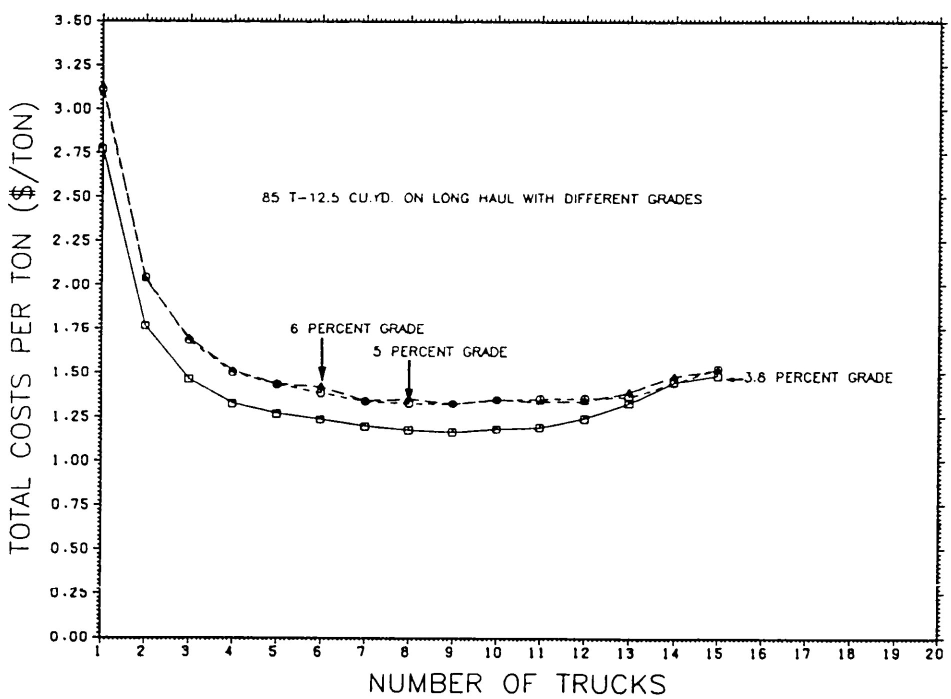

FIG. 4.26 SENSITIVITY ON COSTS DUE TO CHANGE IN GRADE OF HAUL ROAD

According to the figures, it is observed that the change in length of haul road has got the greatest sensitivity on the cost especially at the lesser number of trucks compared with other two parameters.

5.0 MODEL COMPARISON

In this chapter, an overall view of the different models would be discussed and their results be compared.

In figure 5.1, the relationship between number of trucks and production output with different models is shown for comparing purposes. The combination is 85 ton truck and 12.5 cubic yards for long haul distance.

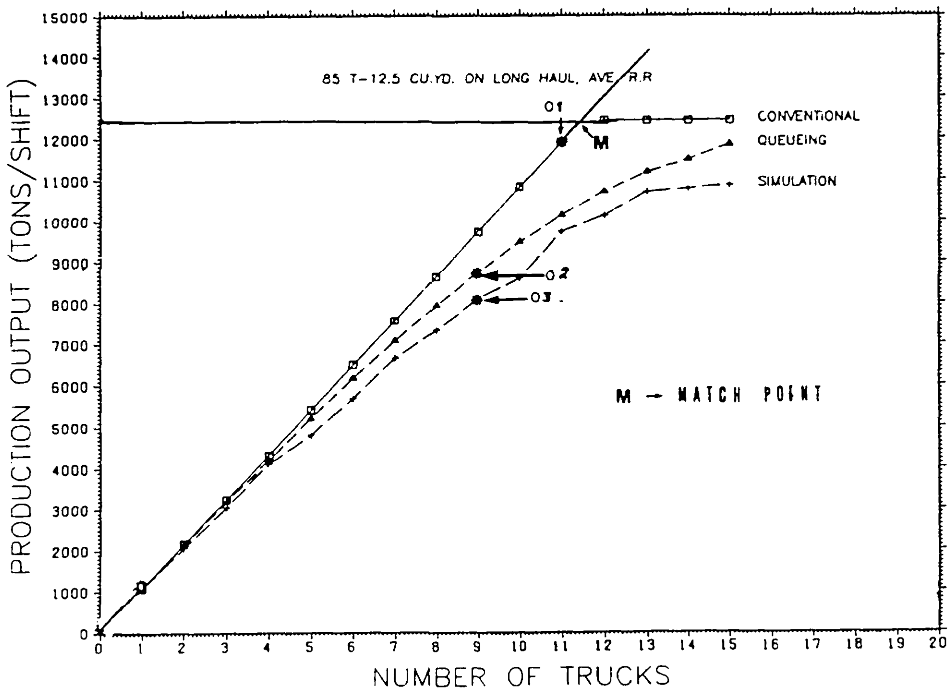

FIG. 5.1 COMPARISON OF METHODS ON PRODUCTION OUTPUT

The conventional model gives a straight line production which continues until the shovel is reached. It represents the maximum possible system production, attainable only in an ideal case. There is no loss of production due to the equipment bunching caused by the variation of haul unit's cycle times and therefore the overestimates are likely to be obtained by conventional model, and thus underestimates of costs per ton are occured. This is shown in figure 5.2 below.

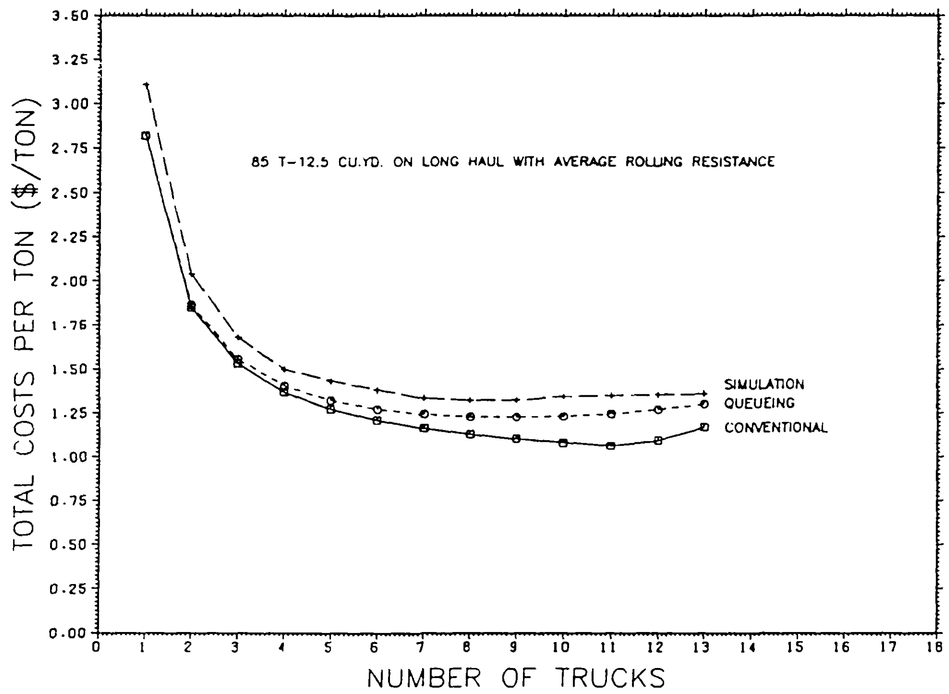

FIG. 5.2 COMPARISON OF METHODS ON COSTS PER UNIT OUTPUT

The use of fixed deterministic value of efficiency factor, in which external delays such as repair and breakdown times, is another reason of causing such overestimates of production.

The queueing model, on the other hand, when the fleet size is small, provides a production output that is very similar

to that estimated by the conventional procedure. However, as the number of trucks increases, so does the waiting time. This waiting time means a loss in production, and so the queueing model production curve starts to flatten off.
Stochastic simulation model furnishes a production output that is less than that of the other two models. The production curve is flatter than the queueing model production curves since the effect of bunching especially at the shovel are taken into account and repair and breakdown times of shovel and trucks are also treated stochastically so as to obtain more realistic results. However,the simulator developed in this study does not take into account the possible delays at the dumping point and/or on the road and therefore, the production figure is probably a little high compared to "real life situation".
Production estimates, obtained from queueing and simulation models, will not exceed the shovel potential once they have reached and therefore there is no equipment mismatching in queueing and simulation models. Method comparisons on production output for 85 ton and 10 cu.yd. and 85 ton and 8 cu.yd. are shown in figures 1 and 2 in Appendix 7 .

<u>Optimum Truck Fleet Size</u>

The determination of optimum truck fleet size will be carried out on the basis of the following two cases :-

case 1. Where the production rate is limited or given

case 2. Where the production rate is not given so that the maximum production with maximum cost could be obtained.

Case 1. Limited Production Rate

In this study, the production rate is limited at 6000 tons/shift as mentioned in the assumptions, and the long haul distance be considered. The following table 5.1 is the illustration of the summary of the optimum fleet size obtained by the different models.

TABLE 5.1 OPTIMUM COMBINATIONS WITH THEIR RELATED MODELS

MODEL	COMBINATION	NO.OF TRUCKS USED	FLEET PROD. (Tons/Shift)	COST PER TON ($/Ton)
Conv.	85T-8cu.yd.	6	6367	1.12
Queu.	85T-8cu.yd.	7	6368	1.26
Simu.	85T-8cu.yd.	7	6315	1.28

All three models show that the 85 ton - 8 cubic yards is the optimum combination for the given production rate of 6000 tons / shift. In conventional model the optimum truck fleet size is 6 while the other two models use 7 trucks. This means that the conventional model gives the overestimates of production output compared to the other two models.

In other words, the production output from queueing model and simulation model are underestimated compared to conventional model. For example, the production output for the conventional model at the point where number of trucks equals 7, is 7428 tons/shift and this is 14.3 % and 14.98

% overestimated over queueing and simulation results. It is also observed that the queueing production output is 0.85% overestimated over simulation results. It is therefore doubtless to say the cost per ton of conventional model appears to be underestimated than that of the other two models.

In determining the optimum fleet size in cases where the production rate is limited, the most important factor to consider is the dipper size of the shovel whose estimated production output is just large enough to meet the planned production rate.

Case 2. Maximum production output with minimum cost

The optimum truck fleet size for each set of shovel-truck combination has been mentioned in detail in chapter 4 and need not be detailed here. However, the following table 5.2 will give some summary of the optimum situations by different models.

TABLE 5.2 SUMMARY OF THE OPTIMUM SITUATIONS BY DIFFERENT MODELS

MODEL USED	COMBINATION ($T-YD^3$)	NO.OF TRUCKS USED	FLEET PROD. (TONS/SHIFT)	COST/TON ($/TON)
CON.	85-8	8	8456	1.07
	100-8	7	8038	1.19
	120-8	6	7768	1.20
	85-10	10	10256	1.10
	100-10	9	10343	1.19
	120-10	7	9315	1.16
	85-12.5	11	11902	1.06
	100-12.5	10	12435	1.11
	120-12.5	9	12191	1.15

TABLE 5.2 CONTINUED.

MODEL USED	COMBINATION ($T-YD^3$)	NO.OF TRUCKS USED	FLEET PROD. (TONS/SHIFT)	COST/TON ($/TON)
QUE.	85-8	6	5671	1.25
	100-8	5	5200	1.39
	120-8	4	4752	1.41
	85-10	8	7320	1.28
	100-10	7	7167	1.38
	120-10	5	6205	1.35
	85-12.5	9	8754	1.22
	100-12.5	7	7949	1.30
	120-12.5	7	8479	1.34
SIM.	85-8	7	6315	1.28
	100-8	6	6179	1.36
	120-8	6	6590	1.42
	85-10	8	7170	1.31
	100-10	8	8026	1.38
	120-10	7	7819	1.41
	85-12.5	9	8110	1.32
	100-12.5	9	9153	1.38
	120-12.5	8	9036	1.41

According to the above table, it is observed that the larger the size of the shovel the larger the production is obtained. The maximum production range that can be produced by the shovel-truck system at the optimum conditions can be summarized on the basis of shovel size as shown in the following table. The table can be used as a general guideline for the similar conditions to that being modelled in this study.

TABLE 5.3 MAXIMUM PRODUCTION OUTPUT RANGE FOR CASE 2

MODEL Used	SHOVEL (Cu.Yd.)	DUMP TRUCK (Ton)	PROD.RANGE (Tons/shift)	NO.OF TRUCKS (Opt.Flt.Size)
	8	85-120	7700-8400	6-8
CONV.	10	85-120	9300-10300	7-10
	12.5	85-120	11900-12400	9-11
	8	85-120	4700-5600	4-6
QUEU.	10	85-120	6200-7300	5-8
	12.5	85-120	7900-8700	7-9
	8	85-120	6100-6590	6-7
SIMU.	10	85-120	7100-8000	7-8
	12.5	85-120	8100-9100	8-9

It is obvious that the conventional results seem to be overestimated over queueing and simulation results. However, it is difficult to say, according to the above table, that the queueing results overestimate the simulation results. It will be helpful to refer back to figure 5.1 that this is true. In the figure 5.1, the optimum fleet sizes for 85 ton and 12.5 cubic yards with various models are shown by symbols "O1", "O2" and "O3". It is found that at the point O1 (at which the number of trucks equals 11), the conventional model produces 14.78 % and 23.22 % overestimated results over queueing and simulation results. And at the points O2 and O3 at which the number of trucks equals 9, the queueing model produces 7.35 % overestimates over simulation results. It is observed that the maximum bunching occurs at the point " M " which is termed matchpoint or balance point and so does the maximum losses in production estimates.

Further analysis on production output on the basis of break-even-analysis for queueing and simulation models have been made and the results are discussed separately in chapter 4. The analysis was made by determining the break-even points of the various combinations. This type of analysis can give further savings in costs since the break-even-points, at which the various cost curves of the combinations intersect, determine the optimum fleet size for the combinations so that the lower and upper limits of production for each combination can be established. It is also observed that at certain points, some of the combinations, although they are not optimum at these points, can give the costs lower than those of combinations whose costs are optimum at these points. In such conditions those conditions with lower costs should be selected, although they are not optimum, rather than selecting the combinations with optimum costs. The following table 5.4 shows the summary of the shovel-truck combinations that should be selected and their ranges of production which are established on the basis of break-even analysis.

TABLE 5.4 PRODUCTION RANGE AND SHOVEL-TRUCK COMBNATIONS

MODEL USED	COMBINATION ($T-YD^3$)	PRODUCTION RANGE (Tons/Shift)
QUEUEING	85 - 8	1000 - 5600
	85 - 12.5	6000 - 9500
SIMULATION	85 - 8	1000 - 7400
	85 - 12.5	8000 - 10800
	120 - 12.5	over 11000

As a whole, in determining the optimum truck fleet the conventional method always furnishes the overestimated output compared to the other two methods and the method should be used only in preliminary stage of estimating production output.

The queueing model seems to tend towards the simulation estimates; however, the results appear to be far from real life situation unless the simulation of the queues is carried out to get closer to real life situation.

The simulation model gives the flatter curve than the other two models and this makes the model tend towards the real life situation. However, the stochastic estimate appears to be probably higher than the real life estimate since the possible delays at the dumping point or on the road are not considered here in this study.

The costs per ton figures obtained from the various combinations being considered in this study seems to be not quite different to each other. However, the figures are determined on costs per ton basis and it is already mentioned that the certain amount of savings can be obtained in annual cost estimations.

6.0. CONCLUSION

The conventional approach is the simplest method to use. No computer facilities are required and only deterministic data are required to apply the method.The method is not suitable for use in studying equipment assignment problems, where the interaction between the equipment is of importance. The method always furnishes production losses due to equipment mismatching. The production output can be limited by either truck potential or shovel potential. Another disadvantage of the method is that the effect of equipment bunching cannot be handled by the method and this leads the overestimates of the production to be produced and hence the underestimates of the costs are obtained. Therefore the conventional method does not give the accurate representation in predicting shovel-truck productivity.

The method can be applied only in smaller fleets to compute the expected output per time unit, however the figures still do not reveal the whole picture of the operation.

The queueing approach, like the conventional one, is simple and can be carried out manually with some reasonable input data. However, more complex queueing problems such as simulation of queues can also be used in modelling the shovel-truck operation by the help of computers. In this study, the simple queueing model is used to perdict the shovel-truck productivity. Unlike the conventional approach the queueing method is not affected by the equipment mismatching. Truck potential cannot be exceeded the shovel potential once it reaches the shovel potential and thus the fleet production is limited by loading function only. The production figures seem to appear underestimate compared to that of conventional approach. This is because the waiting times are taken into consideration in the queueing model. The expected waiting times can be computed deterministically and this provides a useful means of calculating waiting times which can be used as guideline in predicting shovel-truck productivity. The production curve appears to be straight line when smaller number of trucks are used and the curve becomes flatten off as the number of trucks increases. This is because of the waiting times associated with the trucks and shovel.

Like the conventional model, the queueing model also cannot handle the problem of equipment bunching unless the stochastic simulation of a queue is introduced. The production curve of the queueing model lies between the conventional and stochastic curves.

The stochastic simulation approach gives the closest representation of the real life situation of a shovel-truck system. The stochastic approach is not affected by the equipment mismatching as in the conventional approach. The method allows the study of problems such as interactions of the equipment at the loading and dumping points and on the road. The stochastic model is therefore the most suitable method for applications such as :-

(1) The determination of the best haulage route profile for a given mine and equipment

(2) The evaluation of quantitatively the effects on production by altering the haul road parameters, thereby producing useful information to justify expenditures for better road maintenance

(3) The evaluation of changes in equipment sizes

(4) The determination of the best shovel-truck combination that gives the maximum production with minimum costs for a given set of operating conditions of a mine.

(5) The determination of possible range of production output that one set of shovel-truck combination can produce with the lower cost compared to the other shovel-truck combinations.

The following table gives the summary of the various influencing factors and behaviour of the methods on such factors in predicting the shovel-truck productivity.

TABLE 6.1. BEHAVIOUR OF THE METHODS

INFLUENCING	METHODS		
FACTORS	CONVENTIONAL	QUEUEING	STOCH. SIM.
-Mismatching	Affected	Not affected	Not affected
-Bunching	Can't handle	Can't handle	Can handle
-Payload	Deterministic	Deterministic	Stochastic
-Loading Time	Deterministic	Deterministic	Stochastic
-Dumping Time	Deterministic	Deterministic	Stochastic
-Travel Time	Deterministic	Deterministic	Determinisic OR Stochastic
-Waiting Time	Not considered	Deterministic	Stochastic
-Repairs and breakdowns	Fixed value	Fixed value	Stochastic

It is obvious that the above-mentioned abilities of the simulation model make the method an extremely powerful method among the three methods. However, the main problem for stochastic simulation model is that it is not always easy to obtain reliable input data on which to carry out a simulation. Unless the equipment is already being operated, it is very hard to obtain these data. In the case of a feasibility study or where the study of shovel-truck performance for a new mine is in question, the manufacturers' figures or some past figures of a mine, which has a similar working conditions to that being considered could be used.

Eventually, stochastic simulation is the most effective method in which the effect of bunching, the most influencing factors which cannot be determined deterministically, can be handled.

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APPENDIX 1

Table 1. Dumping time (after Morgan, 1968, Quarry Managers Journal, Dec.)

Material Conditions	Dumping time(sec)
Free-flowing, low angle of repose	15
Free-flowing, high angle of repose	25
Flow-resistant, high angle of repose	40
Sticky, flow-resistant	60

Table 2. Spot time at Loading Point(after Morgan, 1968, The Quarry Managers' Journal, Dec.)

Movement	Spot time (sec)
Direct entry to a shovel or loader	8
90 reversed entry to a shovel	15
180 reversed entry to a shovel	30

Table 3. Spot time at Dump point (after Morgan, 1968, The Quarry Mangers' Journal, (Dec.)

Movement	Spot time (sec)	
	90 Reversed Entry	180 Reversed Entry
Open area	15	25
Open edge	20	30
Hopper 100% wider than truck body	25	35
Hopper 50% wider than truck body	40	50
Hopper 10% wider than truck body	50	60
Open area (no reversing)	10	

APPENDIX 1 (CONTD.)

Table 4. Typical Rolling Resistance for various Road Conditions. (After Morgan, 1968, The Quarry Managers' Journal, Dec.)

Type of road surface and conditions	Rolling Resistance in % of Working Vehicle weight.
Hard, metalled road surface	1.5 - 2.0
Smooth,hard,compacted and ballasted surface	2.0 - 2.5
Dry dirt,sand and gravel partly compacted with loose surface material	2.5 - 3.5
Soft hoggin or dirt road	3.5 - 4.5
Soft,loose surface on firm base	4.0 - 5.0
Soft,loose or uncompacted fill	7.0 - 9.0
Loose,sand or gravel with little or no clay	10.0 - 12.0
Deeply rutted dirt on soft base	14.0 - 17.0
Wet,badly churned surface on firm base	15.0 - 20.0

Table 5. Range of Rolling Resistance for a given Road segment (After Runge, 1983, The Aus. I.M.M. Southern Qeensland Branch, Computers in Mining Symposium, May.)

Segment	Rolling Resistance (%)			
	Low	Average	High	Very High
Around Loader	4	6	8	10
In-pit haul road	3	4	6	8
Main out-of pit ramp	2	3	4	5
Surface haul road	2	3	4	5
Around dump area	4	6	7	8

APPENDIX 1 (CONTD.)

Table 6. Operating Efficiency (Terex Corporation, 1981, Production and Costs Estimating of Material Movement with Earthmoving Equipment (Metric version).

Working Condition	Job Efficiency		
	Favourable	Average	Unfavourable
Working min. Per Hour	55	50	40
Percent Efficiency	92	83	67

N.B.

Favourable Job Conditions

1. Material : Topsoil, Loam / clay mixture (low moisture content), compacted coal, Tight earth (no rock).
2. Loading Area : Unrestricted in length and width, Dry and smooth (or maitained by dozer or grader).
3. Total Rolling Resistance : Under 4%, Constant supervision at both loading and dumping areas.

Average Job Conditions

1. Material : Clay with some moiature, soft or well-ripped shale, loose sand with some binder, mixture of different earths, sand/fine gravel mixture.
2. Loading Area : Some restrictions in length or width, Dry with some loose material.

3.Total Rolling Resistance : 4% to 7%

4.Intermittent Supervision at both loading and dumping areas.

Unfavourable Job Conditions

1. Material : Heavy dense or wet clay, loose dry-blown sand with no binder, coarse gravel (no fines), caliche or unripped shale, frequent boulders or rock outcropping.

2. Loading Area : Restricted in length or width, wet, slippery and/or soft (not maintained).

3. Total Rolling Resistance : over 7%

4. No Supervision in loading and dumping areas.

Table 7. Nominal Shovel Cycle Times in seconds for 90% swing. (Ater 1971, Transactions of the Institution of Mining and Metallurgy, vol. 80).

Dipper size		Digging conditions			
cu.yd.	m^3	Easy	Medium	Med-Hard	Hard
4	3	18	23	28	32
5	4	20	25	29	33
6	5	21	26	30	34
7	5.5	21	26	30	34
8	6	22	27	32	35
10	8	23	28	32	36
12	9	24	29	32	37
15	11.5	26	30	33	38
20	15	27	32	35	40
25	19	29	34	37	42

APPENDIX 1 (CONTD.)

Table 8. Maximum Allowable Downhill Speeds

Grade (%)	Speed (mph)
-1	35.0
-2	32.0
-3	29.0
-4	26.0
-5	24.0
-6	22.0
-7	20.0
-8	19.0
-9	18.0
-10	17.0
-11	16.0
-12	15.0
-13	14.0
-14	13.0
-15	12.0

APPENDIX 2

Derivation of Formula for Shovel Production from known values of loading rate and Tons per trip per truck.

Let SHOCAP = Shovel prod. capacity (Tons/Shift)

XMU = Loading rate of the shovel, Trucks/hr

TPT = Tons per trip per truck, Tons

TL = Loading time per truck, min

OPEFS = Operating efficiency of the shovel

QTHS = Shift production per truck, Tons/shift

TRPSH = Trips per shift per truck

SHDU = Duration of shift, Hours

PNUM = No. of passes to fill a truck

tc = Nominal shovel cycle time, sec.

QTH = Hourly production per truck, Tons/hr

TRPH = Trips per hour per truck

BC = Shovel dipper capacity, cu.yds.

VHP = Heaped capacity per truck, cu,yds.

TCT = One round trip total cycle time, min

OPEFT = Operating efficiency of a truck.

$$SHOCAP = XMU * TPT * SHDU \qquad (1)$$

$$XMU = \frac{60 * OPEFS}{TL} \qquad (2)$$

$$TPT = QTHS / TRPSH \qquad (3)$$

$$TL = \frac{PNUM * TC}{60} \qquad (4)$$

$$QTHS = QTH * SHDU \qquad (5)$$

$$TRPSH = TRPH * SHDU \qquad (6)$$

$$PNUM = VHP / BC \qquad (7)$$

$$QTH = \frac{60 * PNUM * BC * OPEFT * DM}{TCT * 2000} \qquad (8)$$

$$TRPH = \frac{60 * OPEFT}{TCT} \qquad (9)$$

Combining Equations (1) through (9), shovel capacity is obtained as follows :-

$$SHOCAP = \frac{3600 * BC * OPEFS * DM}{tc * 2000} \qquad (10)$$

APPENDIX 3

General Concepts used for Estimating Operating Costs of Mining Equipment.

The operating cost items of a mining and construction equipment include the following :

- Tyre costs
- Fuel Costs
- Power Costs
- Costs for lubricating oils, hydraulic oils, etc.
- Repair and maintenance Costs
- Operator Costs.

(1) Tyre Costs

Tyre Costs include : - Tyre Replacement Cost and
- Tyre repair Cost.

Tyre replacement cost can be obtained from

Tyre Replacement Cost = Cost of Tyres / Life of tyres

Tyre Repair Cost can be considered as a percentage of Tyre Replacement Cost. Some manufacturers take 35 % of the list price plus tax.

(2) Fuel Costs

Fuel Costs are dependent on the fuel consumption of the vehicle which in turn depends on the operating conditions. For initial estimates the fuel consumption can be related to engine brake horsepower and the comparative amounts of low-gear and high-gear work. The table below gives typical fuel consumption rates for rear dump mining trucks. The fuel cost can can then be obtained from

Fuel Cost = Fuel Consumption (gal/hr) * Fuel Price ($/gal)

Table 1. Typical Rates of fuel Consuption. (After Gessel, 1978). (Gal. / hr.)

	Duty cycle		
Horsepower	Light	Medium	Heavy
4	5	7.1	8.5
475	6	8.5	10.2
635	8.9	12.7	15.2
700	10.2	14.6	17.5
800	11.8	16.8	20.2
850	12.0	17.0	20.4
1000	17.0	24.3	29.2
1200	17.6	25.6	30.7
1325	21.5	30.7	36.8
1600	23.3	33.3	40.0
2000	27.2	38.8	46.6
2475	32.5	46.4	55.7

(3) Power Costs

Power costs are considered when the equipment in use are electrically operated (e.g. electric power shovel). The following table gives some typical hourly power consumption for electric shovels.

Table 2. Hourly Power Consumption (KWH) (After Caterpillar Hand Book).

SHOVEL SIZE ($YD.^3$)	POWER CONSUMPTION (KWH)
6	225
8	300
10	375
12	475
17	740
20	1250

Power cost is then calculated as follows :

Power Cost = Power consumption (KWH) * Price of power in c/KWH.

(4) Costs for lubricating oil, grease, hydraulic etc.

This cost item is usually considered as 1/3 price of the power or fuel cost.

(6) Repair and Maintenance Costs

Repair and maintenance costs can be related to the duty of the operation and the purchase price of the truck for an approximate estimate.

Average hourly repair and maintenance cost can be expressed as follows :

$$\text{Av. hourly repair \& maintenance cost} = \frac{P}{F}$$

Where P = purchase price of new truck (less tyre)

F = a duty factor (given below). (After Morgan, 1969).

Percentage of operating time in high gears	75 %	50 %	25 %	0 %
Duty factor F	24000	20000	15000	8000

(6) Operator Costs

The cost of the operator should include the basic hourly wage, the average hourly costs of any bonus or incentive scheme, plus all other direct costs such as pension contributions, National Insurance contributions, sickness fund contributions etc.

APPENDIX 4

ASSUMPTIONS

The following assumptions were made for the estimation throughout the study. Short tons are used for the calculation purposes.

Type of material

Nature - Rock, hard, well ballasted.

Density - Bank state : 4000 lbs / cubic yard

Loose state : 2700 lbs / cubic yard

Haul Road Conditions

(1) Long Haul

Main Haul Length - 14600 ft, return on the main haul road

Main Haul Width - Ample room for two-way traffic.

In-pit length - 800 Ft

Around crusher length - 1000 Ft

Average Grade - 5 percent (see Table 4.21, Chapter 4.3.2 for detail).

Average Rolling Resistance - 3 percent (see Tables 4 and 5 for detail in Appendix 1.)

(2) Medium Haul

Main Haul length - 7500 Ft, two-way traffic

In-pit length - 400 Ft

Around crusher length - 700 Ft

Average Grade - same as in long haul

Average Rolling Resistance - same as in long haul.

(3) Short Haul

Main haul length - 4500 Ft, two-way traffic

In-pit length - 300 Ft

Around crusher length - 700 Ft

Average Grade - same as in long haul

Average Rolling Resistance - same as in long haul

Transport Facility

Payload and Type - 85,100, and 120 short tons rear dump trucks, with mechanical drive power train.

Horsepower - 880, 1000, and 1050 HP.

Engine efficiency - 87% for all trucks.

Truck operating efficieny - 87%.

Performance characteristics - see Appendices 5A,5B and 5C for detail characteristics for each type of truck.

Loading Facility.

Type and dipper size - Three electric mining shovels with 8,10 and 12.5 cu.yd. capacities.

Shovel operating efficiency - 87%

Nominal shovel cycle Time - Varying from 32 sec. to 34 sec. depending on the size of dipper (at 90% swing angle).

(See Table 7 in Appendix 1 for detail shovel cycle time in accordance with digging conditions).

Dumping Facility.

Material discharge	-	Eventually to Primary Crusher.
Crusher capacity	-	750 short tons per hour.
Type of crusher	-	Gyratory crusher.

Travel Constraints

Maximum allowable velocity - 30 mph. Trucks were forced to reduce speeds for safe travel around sharp corners and through dangerous intersections.

Miscellaneous

Working days per year	-	300 days.
No. of shifts per day	-	Two 8 hour shifts.

APPENDIX 5A

TEREX 33-11D HAULER

SPECIFICATIONS SUBJECT TO CHANGE WITHOUT NOTICE

CAPACITY

Struck (SAE)	51.4 yds.³ (39.3 m³)
Heaped 1:1	78.2 yds.³ (59.8 m³)
Heaped 2:1 (SAE)	64.8 yds.³ (49.5 m³)
Heaped 3:1	60.3 yds.³ (46.1 m³)

ENGINE

Detroit Diesel 16V-92TA Turbocharged, Aftercooled 2 Cycle Diesel

Gross Vehicle Power @ 2100 RPM	880HP (656 kW)
Flywheel Power @ 2100 RPM	840HP (626 kW)

NOTE: The above ratings are S.A.E. at 500 ft. (152 m) altitude and 85°F (29°C). Gross power rating includes standard engine equipment, such as water pump, fuel pump, and lubricating oil pump. Flywheel power is the net power after deductions from gross power for fan, alternator, and air compressor requirements. Turbocharged engine requires no deration to 10,000 ft. (3 000 m) altitude.

Number of Cylinders	16
Bore and Stroke	4.84 in. x 5.0 in. (123 mm x 127 mm)
Piston Displacement	1472 in.³ (24.1 litres)
Maximum Torque @ 1400 RPM	2372 ft. lbs. (3 216 N·m)
Fan Diameter	60 in. (1.52 m)
Radiator Frontal Area	26.7 ft.² (2.48 m²)
Air Starting System	125 PSI (862 kPa) with Neutral Start Feature

TRANSMISSION—Allison DP-8961

Allison transmission with integral torque converter and manual electric shifting mechanism mounted amidship in frame. Six speeds forward, one reverse. Automatic converter lock up in all forward speed ranges. Standard downshift inhibitor prevents downshifting into lower speed ranges at high engine speeds.

Stall Speed	1800-1900 RPM
Maximum speed @ 2100 RPM	36.6 MPH (58.9 km/hr)

DRIVE AXLE

Heavy duty, full floating axle shafts with single reduction spiral bevel gear differential and planetary final reduction in each wheel

Ratios: Differential	2.50:1
Planetary	9.10:1
Total Reduction	22.75:1

SUSPENSION

Front—Independent king pin strut type with self-contained variable rate nitrogen-oil cylinders.

Maximum Strut Stroke 9.25 in. (235 mm)

Rear—Self-contained variable rate nitrogen-oil cylinders with A-frame linkage and lateral stabilizer bar

Maximum Strut Stroke	6.80 in. (173 mm)
Maximum Axle Oscillation	7.0 Degrees

CAB

Heavy duty steel construction, 66 inches (1.68 m) wide, mounted to the left with entry from either side. Insulated for sound and temperature control, and has tinted safety glass in the front and side windows, 6-way adjustable operator's seat and 4-position tilt steering column

BODY

Longitudinal "V" type floor with eight integral transverse C section stiffeners. The body is exhaust heated and rests on resilient impact absorption pads.

Body Plate	Nominal Thickness Inches	(mm)	Minimum Yield Strength PSI	(mPa)
Floor	.75	(19.1)	110,000	(758)
Side	.38	(9.7)	90,000	(621)
Front-Upper	.31	(7.9)	90,000	(621)
Front-Lower	.38	(9.7)	90,000	(621)
Cab Guard	.19	(4.8)	90,000	(621)

Floor plate is high hardness (321 BNH minimum) abrasion resistant steel. All other body material is moderate strength, low alloy steel, with a minimum yield strength of 45,000 PSI (310 mPa).

HOIST

The body hydraulic system is independent of the steering hydraulic system and has two body hoists invertedly mounted inside of frame rails. Hoists are two stage with power down in second stage.

Body hydraulic pump @ 2100 RPM	96 GPM (6.1 litres/sec)
Body raise time	16.3 Seconds
Body lower time	18.0 Seconds

FRAME

Fabricated full box section frame with integral front bumper and closed loop crossmember. Crossmember and torque tube connections are of high strength alloy steel castings.

STEERING

Independent hydrostatic steering with closed center steering valve, accumulator, pressure compensating piston pump, and dual double acting steering cylinders. Accumulator provides uniform steering regardless of engine speed.

Maximum Tire Steering Angle 38°

Accumulator provides temporary steering reserve if engine stalls Approximate steering reserve 2 lock to lock turns. Low pressure warning light signals operator when pressure drops below 1200 PSI (8 274 kPa).

BRAKES

Service—Independent front and rear systems actuated by single treadle with auxiliary manual control. Front brakes are dual shoe internal expanding type wedge actuated by individual air-over-oil intensifiers. Brake shoes are free floating. Operator controlled wet/dry road valve reduces front brake pressure 50% for control in slippery conditions. Rear brakes are TEREX oil-cooled, air-over-oil actuated disc brakes which provide both service and retarder braking. Completely sealed from dirt and water. Each unit is individually replaceable.

Brake Lining:

Front: Diameter	28 in. (710 mm)
Width	8 in. (203 mm)
Lining area	906 in.² (5 846 cm²)
Rear: Lining area	13,572 in.² (87 567 cm²)
Total Lining area	14,478 in.² (93 413 cm²)
Air compressor capacity	24 CFM (11 300 cm³/sec)
System Pressures:	
Air (Max.)	125 PSI (862 kPa)
Front Oil (Max.)	1,875 PSI (12 928 kPa)
Rear Oil (Max.)	750 PSI (5 171 kPa)

Emergency—Wig-wag warning in cab actuated if air pressure drops below 80 PSI (552 kPa). Front and rear automatically actuate if system air pressure reduces to 45 PSI (310 kPa).

Total safety brake air reservoir capacity 7,080 in.³ (116 020 cm³)

Parking—Spring applied oil released parking brakes are built into rear axle brake units.

TIRES AND RIMS (Tubeless)

	Rim Width
Standard Front & Rear	
24.00 - 49 (42PR) E-3	17 in. (.43 m)
Optional Front & Rear	
24.00 - 49 (42PR) E-4	17 in. (.43 m)
24.00 - 49 (48PR) E-4	17 in. (.43 m)
24.00R49 (Two Star) Radial	17 in. (.43 m)
24.00 - 51 (48PR) E-3 or E-4	17 in. (.43 m)
27 - 56.5 (42PR) E-4 (Includes One Piece Rims)	20 in. (.51 m)
27.00 - 49 (42PR) E-3	17 in. (.43 m)
27.00 - 49 (42PR) E-4	17 in. (.43 m)
27.00R49 (Two Star) Radial	17 in. (.43 m)

NOTES:

1) When body liners or optional 1 in. (25 mm) floor body is used, 24.00 - 49 (48PR) tires, or equivalent, are recommended.

2) Productivity and performance capabilities of TEREX haulers are such that under specific job conditions, the Ton-MPH (TKPH) capability of Standard or Optional tires can be exceeded. Operation above the Ton-MPH (TKPH) rating may lead to premature tire problems. TEREX recommends that the user consult the tire manufacturer, and evaluate all job conditions in order to make the proper tire selection.

TIRE SPACING

Inside Front	9 ft. 11 in. (3.02 m)
Inside Drive	5 ft. 7 in. (1.70 m)
Outside Front	14 ft. 9 in. (4.50 m)
Outside Drive	15 ft. 6 in. (4.72 m)

ELECTRICAL

24 volt. Two 5-SH 80 amp-hr (288 kC) batteries. 65 amp Leece-Neville alternator with integral transistorized voltage regulator. Master electrical disconnect switch in cab.

SERVICE DATA

	U.S. Gals.	(litres)
Engine Crankcase (Incl. Filters)	19.0	(71.9)
Transmission (Incl. Filters)	26.5	(100.3)
Cooling System	56.0	(212.0)
Steering Hyd. Tank	16.0	(60.6)
Steering Hyd. System (Total)	19.0	(71.9)
Body Hyd. Tank	72.0	(272.5)
Body Hyd. System plus Brake Cooling System	140.0	(529.9)
Planetaries (Total)	10.0	(37.9)
Differential	16.0	(60.6)
Fuel Tank	325.0	(1 230.0)
Power Take Off	1.0	(3.8)
Optional Air Conditioner (Freon)	60 oz.	(1.7 kg)

DIMENSIONS

All dimensions for fully loaded vehicles unless noted.

Wheelbase	15 ft. 0 in.	(4.57 m)
Cab Guard to Ground		
w/Body Down (Empty)	15 ft. 8 in.	(4.76 m)
w/Body Raised (Empty)	28 ft. 8 in.	(8.74 m)
Cab Roof to Ground (Empty)	14 ft. 6 in.	(4.42 m)
Loading Height		
(Empty, Standard Body)	14 ft. 3 in.	(4.34 m)
(Empty, Ore Body)	13 ft. 8 in.	(4.17 m)
Chassis Length	28 ft. 3 in.	(8.61 m)
Overall Length	34 ft. 3 in.	(10.44 m)
Overall Width		
(Rear Tires)	15 ft. 6 in.	(4.72 m)
(Platform w/extensions)	15 ft. 10 in.	(4.83 m)
(Body)	15 ft 0 in	(4.57 m)
(Platform w/o extensions)	12 ft. 6 in.	(3.81 m)
Front Axle Track	12 ft. 4 in.	(3.76 m)
Drive Axle Track	10 ft. 8 in.	(3.25 m)
Front Axle Clearance	2 ft. 4½ in	(.72 m)
Drive Axle Clearance	2 ft 3 in.	(.69 m)
Transmission Clearance	2 ft. 11 in.	(.89 m)

Body		
Length Inside	22 ft. 4 in.	(6.81 m)
Width Inside	14 ft. 0 in.	(4.27 m)
Depth Inside (Max.)	5 ft. 6 in.	(1.68 m)
Height Standard Canopy Front Rail	7 in.	(.18 m)
Dump Angles		
Floor w/Horizontal		57°
Tail Chute w/Horizontal		42°
Turning Diameter on Front Axle Track (SAE) @ 38° Steering Angle	67 ft. 0 in.	(20.42 m)
Vehicle Clearance Circle (SAE) @ 38° Steering Angle	75 ft. 0 in.	(22.86 m)

NOTE: Add 1 in. (.025 m) to all vertical dimensions for E-4 tires.

WEIGHTS (Mass)

	(lbs.)	(kg)
Net Weight (Mass) Distribution		
Front Axle	56,400	(25 600)
Rear Axle	64,400	(29 200)
Total	120,800	(54 800)
Payload	170,000	(77 100)
Gross Weight (Mass) Distribution		
Front Axle	95,700	(43 300)
Rear Axle	195,100	(88 500)
Total	290,800	(131 800)
Chassis with Hoists	92,400	(41 900)
Standard Body	28,400	(12 900)
Full Liner Weight (Optional)	16,700	(7 600)

SPECIFICATION CHANGES TO REFLECT 27.00 - 49 (42PR) E-3 TIRES

Maximum Speed @ 2100 RPM ... 38.6 MPH (62.1 km/hr)

STEERING

Maximum Tire Steering Angle ... 35°

TIRE SPACING

Inside Front	9 ft. 8 in. (2.95 m)
Inside Drive	5 ft. 5 in. (1.65 m)
Outside Front	15 ft. 0 in. (4.57 m)
Outside Drive	16 ft. 3 in. (4.95 m)

DIMENSIONS

Cab Guard to Ground		
w/Body Down (Empty)	15 ft. 10½ in.	(4.84 m)
w/Body Up (Empty)	28 ft. 10½ in.	(8.80 m)
Cab Roof to Ground	14 ft. 8½ in.	(4.48 m)
Loading Height		
(Empty, Standard Body)	14 ft. 5½ in.	(4.41 m)
(Empty, Ore Body)	13 ft. 10½ in.	(4.23 m)
Chassis Length	28 ft. 5 in.	(8.66 m)
Overall Width (Rear Tires)	16 ft. 3 in.	(4.95 m)
Front Axle Track	12 ft. 4 in.	(3.76 m)
Drive Axle Track	10 ft. 10 in.	(3.30 m)
Front Axle Clearance	2 ft. 7 in.	(0.79 m)
Drive Axle Clearance	2 ft. 5½ in.	(0.75 m)
Turning Diameter on Front Axle Track (SAE) @ 35° Steering Angle	70 ft. 8 in.	(21.54 m)
Vehicle Clearance Circle (SAE) @ 35° Steering Angle	78 ft. 8 in.	(23.98 m)

NOTE. Add 1 in (.025 m) to all vertical dimensions for E-4 tires.

WEIGHTS (Mass)

	(lbs.)	(kg)
Net Weight (Mass) Distribution		
Front Axle	57,200	(25 900)
Rear Axle	66,100	(29 900)
Total	123,300	(55 800)
Payload	170,000	(77 100)
Gross Weight (Mass) Distribution		
Front Axle	96,500	(43 800)
Rear Axle	196,800	(89 100)
Total	293,300	(132 900)
Chassis with Hoists	94,900	(43 000)

APPENDIX 5B

EUC R-100

ENGINES

Make	**Detroit Diesel**	**Cummins**
Model	12V-149T	KTA-2300-C
Type	2 Cycle	4 Cycle
Aspiration	Turbocharged	Turbocharged
Rated Output (SAE)	@ 1900 rpm 783 kW (1050 bhp)	@ 2100 rpm 783 kW (1050 bhp)
Flywheel Output (SAE)	@ 1900 rpm 746 kW (1000 bhp)	@ 2100 rpm 746kW (1000 bhp)
Number Cylinders	12	12
Bore & Stroke	146 mm x 146 mm (5¾" x 5¾")	159 mm x 159 mm (6¼" x 6¼")
Displacement	29.4 lit. (1792 in³)	37.7 lit. (2300 in³)
Maximum Torque	@ 1400 rpm 3951 N•m (2915 lb-ft)	@ 1500 rpm 4095 N•m (3020 lb-ft)
Starting	Air	Air

TRANSMISSION

Allison DP-8962 planetary type, full power shift with automatic shifting. Integral torque converter with automatic lock-up in all ranges and Integral hydraulic retarder. Remote mounted, 6 forward speeds, 1 reverse.

		MAXIMUM SPEEDS			
	Gear	3.42:1 Diff.		3.15:1 Diff.	
Range	Ratio	km/h	(mph)	km/h	(mph)
1	4.24	9.41	(5.85)	10.22	(6.35)
2	2.34	17.20	(10.69)	18.68	(11.61)
3	1.70	23.60	(14.67)	25.63	(15.93)
4	1.31	30.47	(18.94)	33.08	(20.56)
5	1.00	39.90	(24.80)	43.33	(26.93)
6	.73	54.69	(33.99)	59.37	(36.90)
R	5.75	6.93	(4.31)	7.53	(4.68)

DRIVE AXLE

Full floating axle shafts, double reduction provided by Euclid Model 2650 differential and single reduction planetary with balanced life gears in each wheel.

Ratios	**Standard**	**Optional**
Differential	3.42:1	3.15:1
Planetary	7.41:1	7.41:1
Total Reduction	25.34:1	23.34:1
Maximum Speeds	**km/h**	**(mph)**
With 27.00-49 tires	54.7	(59.4)
	34.0	(36.9)
30.00-51 tires	58.2	(63.2)
	36.2	(39.3)

TIRES

Standard	**Rim Width**
Front & Rear *27.00-49 (48PR) E-3	495mm (19.5")
Optional	
Front & Rear *30.00-51 (40PR)	559mm (22.0")

*With one piece 15° drop center rim, plus tire types, treads and ply ratings.

FRAME

Box section main rails bridged by three cross members, front bumper and front suspension tube. Rails are constant taper, constructed of 689 N/mm² (100,000 psi) yield strength steel. Two rear cross members are 655 N/mm² (95,000 psi) yield strength steel castings with integral suspension and drive axle mountings. Cross member to frame rail junctions use large radii to minimize stress.

LOAD CAPACITY

	m³	(yd³)
Struck (SAE)	35.1	(45.9)
Heap 3:1	48.1	(62.9)
Heap 2:1 (SAE)	54.7	(71.6)
Field Heap	52.8	(69.0)

Optional bodies offered on request. Consult Euclid's Sales Engineering Department.

WEIGHTS

	kg	(lb)
Chassis with Hoists	51 700	(114,000)
Body	15 800	(34,900)
Net Weight	67 500	(148,900)
Front Axle	32 500	(71,700)
Rear Axle	35 000	(77,200)
Payload	90 700	(200,000)
Gross Weight	158 200	(348,900)
Front Axle	53 400	(117,700)
Rear Axle	104 800	(231,200)
Options:		
Body Rock Liner, Complete (3/8" floor, 1/4" side, 3/8" corner, 3/8" top rail, 1/2" end protection)	4 055	(8,940)
Tires:		
Extra Rock Tread		
27.00-49 (48) E-4	1 149	(2,532)
30.00-51 (40) E-4	4 387	(9,672)

EUC R-100

STANDARD EQUIPMENT

General

- Air Cleaner Guards
- Air Horns, Dual
- Body Down Indicator, Mechanical
- Body Prop Cable
- Cast Tow Hooks
- Differential Ratio, 3.42:1 (34.0 MPH)
- Exhaust Heated Body (Closed Loop)
- Fan Guard
- Fully Hydraulic Brake System
- Ground Level Air Start Charge Line
- Guard Rails Around Platform
- Hydraulic Retarder
- Mirrors, Right and Left Side
- Moisture Ejector (Air Reservoir)
- Mud Flaps
- Operator Arm Guard
- Radiator Grille Guard
- Reverse Alarm
- Rock Ejector Bars
- Supplementary Accumulator Steering

Cab

- Ash Tray
- Automatic Shift
- Cab Interior Light
- Cigar Lighter
- Fold-down Service Tray
- Full Electrical Terminal Block
- Hand Control Valve for Rear Brakes
- Heater and Defroster
- Operator Seat, Air Ride
- Operator Seat Belt
- Passenger Seat
- Rubber Floor Mat
- Sun Visor
- Tilt Steering Wheel
- Tinted Glass, All Windows
- Windshield Washer
- Windshield Wiper

Gauges and Indicators

- Air Cleaner Restriction Gauge
- Air Pressure Gauge
- Brake/Steering Pressure Gauge
- Clutch Pressure Gauge
- Converter Lock-Up Indicator Light
- Converter Oil Temperature Gauge
- Coolant Temperature Gauge
- Engine Oil Pressure Gauge
- Gauge Lights Switch
- High Beam Indicator Light
- Hourmeter
- Hydraulic Filter Restriction Indicator Light
- Parking Brake Indicator Light
- Rear Brake Malfunction Indicator Light
- Speedometer, Digital Readout
- Steer System Malfunction Indicator Light
- Steering Filter Restriction Indicator Light
- Tachometer, Digital Readout
- Voltmeter

Vehicle Lights

- Back-Up Light
- Clearance Lights
- Engine Compartment Light
- Headlights, four
- Dual Combination Stop and Tail Lights
- Turn Signals and Four-way Flasher

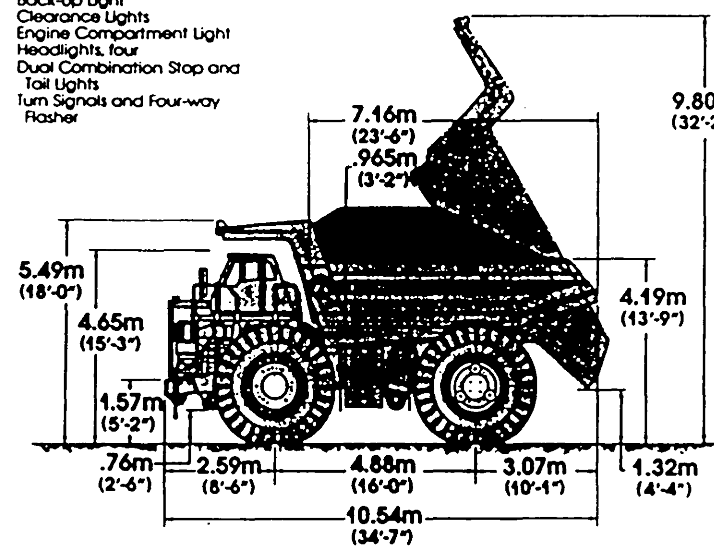
OPTIONAL EQUIPMENT

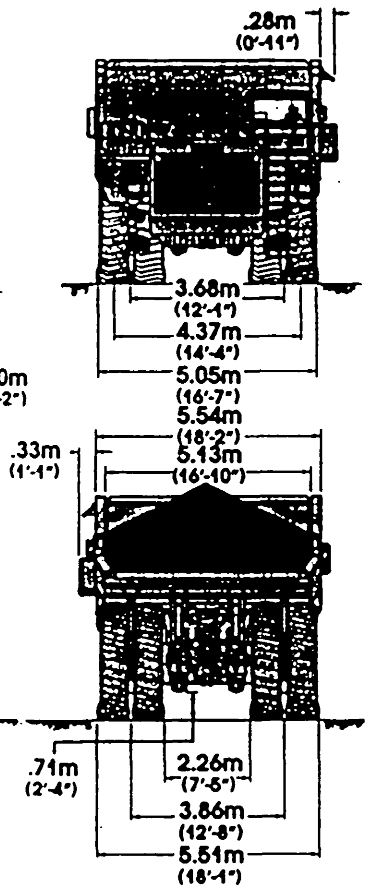
- Air Conditioner
- Air Dryer
- Alarm System, Four Function (low oil pressure, high coolant temperature, low coolant level, high conv. temp.)
- Alcohol Vaporizer
- Body Rock Liners
- Canopy Spill Guard Extension
- Centralized Lube
- Centralized Service
- Cold Starting Aid
- Differential, Optional 3.15:1 Ratio
- Differential, No Spin
- Electrical Start
- Fast Fueling System (Wiggins)
- Fuel Gauge, Cab or Tank Mounted
- Hubodometer
- Kim Hotstart
- Lube System, Automatic
- Radiator Shutters
- Range Indicator Light Assembly
- Tachograph, 24 Hour Recording
- Thermatic Fan

Standard and optional equipment may vary from country to country.

Special options provided on request. Consult Euclid Sales Engineering Department.

Product improvement is a continuing Euclid project. Therefore, all specifications are subject to change without notice.





Note: Dimensions shown are for standard empty vehicle equipped with 27.00-49 tires.

The Euclid Field Heap illustrated above maintains a 2:1 heap ratio from the floor/tail chute junction to the peak of the load profile. The SAE 2:1 heap ratio is actually a 1:1 heap ratio from floor/tail junction to the top body edge, then switches to a 2:1 heap ratio to the load peak. The Euclid field heap is more representative of field loading practices and payload distribution. Euclid body capacity ratings are based on the field heap philosophy.

EUC R-120M

ENGINES

Make	Detroit Diesel	Cummins
Model	12V-149TI8	KTA-2300-C
Type	2 Cycle	4 Cycle
Aspiration	Turbo-Charged	Turbo-Charged
Rated Output (SAE)	@ 1900 rpm 895 kW (1200 bhp)	@ 2100 rpm 895 kW (1200 bhp)
Flywheel Output (SAE)	@ 1900 rpm 783 kW (1050 bhp)	@ 2100 rpm 783 kW (1050 bhp)
Number Cylinders	12	12
Bore & Stroke	146mm x 146mm (5¾" x 5¾")	159mm x 159mm (6¼" x 6¼")
Displacement	29.3 litres (1788 in³)	37.7 litres (2300 in³)
Maximum Torque	@ 1600 rpm 4670 N•m (3445 lb-ft)	@ 1500 rpm 4475 N•m (3300 lb-ft)
Starting	Air	Air

TRANSMISSION

Allison CLBT-9680. Planetary type, full power shift with automatic shifting. Integral torque converter with automatic lock-up in all ranges and integral hydraulic retarder. Remote mounted. 6 forward speeds, 1 reverse. Shift range indicator is standard.

Maximum Speeds @ 2100 RPM Governed Engine Speed.

Range	Gear Ratio	Standard 1.56:1 Diff. 17.06:1 Plan. km/h	(mph)	Optional 1.56:1 Diff. 21.44:1 Plan. km/h	(mph)	Optional 1.56:1 Diff. 14.23:1 Plan. km/h	(mph)
1	4.24	10.02	(6.23)	7.55	(4.96)	12.02	(7.47)
2	3.05	13.90	(8.64)	11.07	(6.88)	16.67	(10.36)
3	2.32	18.33	(11.39)	14.58	(9.06)	21.98	(13.66)
4	1.67	25.34	(15.75)	20.16	(12.53)	30.38	(18.88)
5	1.00	42.49	(26.41)	33.81	(21.01)	50.94	(31.66)
6	.72	57.31	(35.62)	45.60	(28.34)	68.70	(42.70)
R	4.13	10.30	(6.40)	8.19	(5.09)	12.34	(7.67)

DRIVE AXLE

Full floating axle shafts, reduction provided by Euclid Model 2655 differential and dual path planetary with balanced life gearing in each wheel. Modular differential sealed to maintain oil separate from rest of drive axle.

Ratios	Standard	Optional	Optional
Differential	1.56:1	1.56:1	1.56:1
Planetary	17.06:1	21.44:1	14.23:1
Total Reduction	26.61:1	33.45:1	22.20:1
Maximum Speeds	km/h (mph)	km/h (mph)	km/h (mph)
With 30.00-51 tires	57.31 (35.62)	45.60 (28.34)	68.70 (42.70)
With 33.00-51 tires	60.40 (37.54)	48.06 (29.87)	72.41 (45.00)

The R-120M's advanced rear axle design continues the Euclid tradition of performance, durability and ease of service. The D-ring configuration of the axle housing exhibits a greater strength-to-weight ratio than typical A-frame designs. The differential is three-point mounted between the axle housing frame rails, isolating it from bending and torsional stresses. Modular in design, the differential is a completely sealed unit with its own oil capacity. The assembly can be removed in about one-half the time required for a conventional gear set. A change-out differential can then be quickly installed while the original unit is reconditioned under controlled shop conditions.

The R-120M utilizes a coupled planetary system with two sets of gears in each wheel. Each set of gears helps drive its respective wheel, effectively sharing torque loads. This concept keeps individual gear loading to a minimum, thereby promoting longer component lives.

The R-120M is available in three distinct gear ratios allowing it to be tuned to a specific haulage application.

TIRES

Standard	Rim Width
Front & Rear 30.00-51 (46PR) E-4	559mm (22.0")
Optional	
Front & Rear 33.00-51 (50PR) E-4	610mm (24.0")

Plus tire types, treads and ply ratings

LOAD CAPACITY

	m³	(yd³)
Struck (SAE)	42.5	(55.6)
Heap 3:1	58.0	(75.9)
Heap 2:1 (SAE)	65.4	(85.5)
Euclid Field Heap	63.3	(82.8)

Optional bodies offered on request. Consult your nearest Euclid Distributor.

WEIGHTS

	kg	(lb)
Net Weight	77 840	(171.600)
Front Axle	36 920	(81.400)
Rear Axle	40 920	(90.200)
Payload	108 860	(240.000)
Gross Weight	186 690	(411.600)
Front Axle	62 230	(137.200)
Rear Axle	124 470	(274.400)

Options:	Additional Weight	
Body Rock Liners, Complete: 19mm (¾") floor, 10mm (⅜") sides, front and top rails, 6mm (¼") canopy and corners	8 046	(17.700)
Tires: (6) 33.00-51 (50PR) E-4 Extra Rock Tread	3 710	(8.178)

STEERING

Closed center full time hydrostatic power steering using two double acting cylinders, tie rod, piston type pump and combined brake/steering system reservoir. Accumulator provides supplementary steering.

Steering Angle .... 42°
Turning Circle (SAE) .... 24.5m (80'-6")
Steering Pump Output .... [illegible]
System Relief Pressure .... 17 237 kPa (2500 psi)

EUC R-120M

STANDARD EQUIPMENT

General

Air Cleaner Guards
Air Horns, Dual
Body Down Indicator, Mechanical
Body Prop Cable
Cast Tow Hooks
Exhaust Heated Body (Closed Loop)
Fan Guard
Fully Hydraulic Brake System
Ground Level Air Start Charge Line
Guard Rails Around Platform
Hydraulic Retarder
Mirrors, Right and Left Side
Moisture Ejector (Air Reservoir)
Mud Flaps
Operator Arm Guard
Radiator Grille Guard
Reverse Alarm
Rock Ejector Bars
Supplementary Accumulator Steering
Differential Ratio 1.56:1
Planetaries Ratio 17.06:1

Cab

Ash Tray
Automatic Shift
Cab Interior Light
Cigar Lighter
Fold-down Service Tray
Full Electrical Terminal Block
Hand Control Valve for Rear Brakes
Heater and Defroster
Operator Seat, Air Ride
Operator Seat Belt
Passenger Seat
Rubber Floor Mat
Sun Visor
Tilt Steering Wheel
Tinted Glass, All Windows
Windshield Washer
Windshield Wiper

Gauges and Indicators

Air Cleaner Restriction Gauge
Air Pressure Gauge
Brake/Steering Pressure Gauge
Clutch Pressure Gauge
Converter Lock-Up Indicator Light
Converter Oil Temperature Gauge
Coolant Temperature Gauge
Engine Oil Pressure Gauge
Gauge Lights Switch
High Beam Indicator Light
Hourmeter
Hydraulic Filter Restriction Indicator Light
Parking Brake Indicator Light
Range Indicator Light Assembly
Rear Brake Malfunction Indicator Light
Speedometer, Digital Readout
Steer System Malfunction Indicator Light
Steering Filter Restriction Indicator Light
Tachometer, Digital Readout
Voltmeter

Vehicle Lights

Back-Up Light
Clearance Lights
Dual Combination Stop and Tail Lights
Engine Compartment Light
Headlights, four
Turn Signals and Four-way Flasher

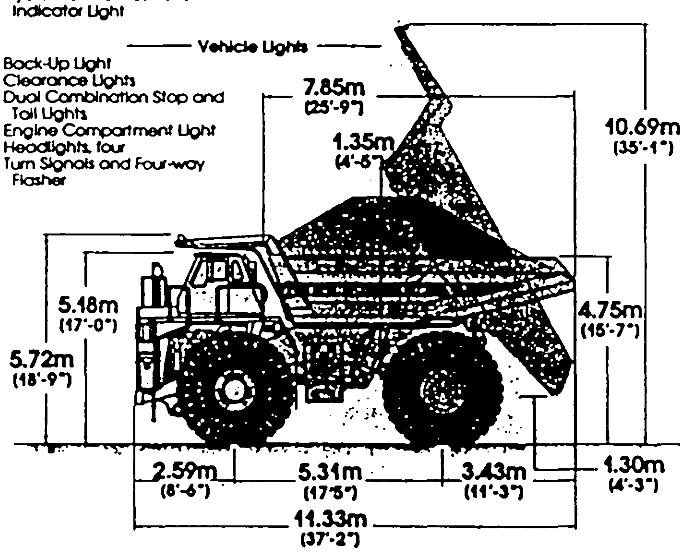
OPTIONAL EQUIPMENT

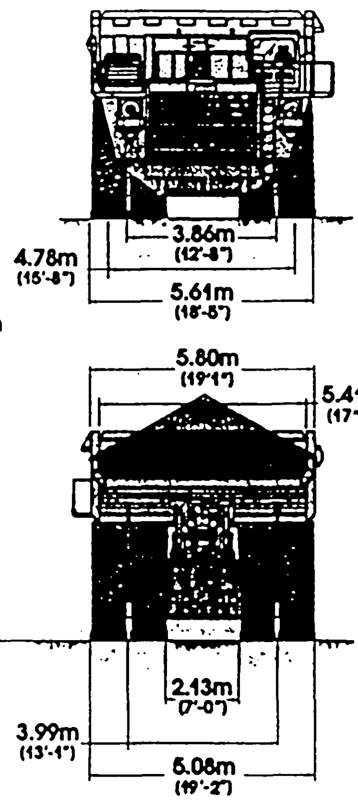
Air Conditioner
Air Dryer
Alarm System, Four Function (low oil pressure, high coolant temperature, low coolant level, high conv. temp.)
Alcohol Vaporizer
Body Rock Liners
Canopy Spill Guard Extension
Centralized Lube
Centralized Service
Cold Starting Aid
Fast Fueling System (Wiggins)
Fuel Gauge, Cab or Tank Mounted
Hubodometer
Kim Hotstart
Lube System Automatic
Planetaries Ratio 14.23:1
Planetaries Ratio 21.44:1
Radiator Shutters
Tachograph, 24 Hour Recording
Thermatic Fan

Standard and optional equipment may vary from country to country.

Special options list and other literature is available from your nearest Euclid Distributor.

Product improvement is a continuing Euclid project. Therefore, all specifications are subject to change without notice.





Note: Illustration may include optional equipment.

The Euclid Field Heap illustrated in the side view above maintains a 2:1 heap ratio from the floor/tail chute junction to the peak of the load profile. The SAE 2:1 heap ratio is actually a 1:1 heap ratio from floor/tail junction to the top body edge, then switches to a 2:1 heap ratio to the load peak. The Euclid field heap is more representative of field loading practices and payload distribution. Euclid body capacity ratings are based on the field heap philosophy.

APPENDIX 6

METHODS FOR DEPRECIATION

For economy study purposes the requirements of a depreciation method are somewhat different. The following are the most common used method of depreciation.

(1) The Straight Line Method.

The straight line method of computing depreciation assumes that the loss in value is directly proportional to the age of the structure. Thus with this formula if

L = useful life of the structure in years

C = the original cost

d = the annual cost of depreciation

S = the scrap value at the end of the life of the structure

D_n = depreciation up to age n years;

then

$$d = \frac{C - S}{L}$$

$$D_n = \frac{n(C - S)}{L}$$

This method of computing depreciation is more widely used and it does not take into account interest, operation and maintenance costs, or profits.

(2) Declining Balance Method

In this method it is assumed that the annual cost of depreciation is a fixed percentage of the salvage value at the beginning of the year. The ratio of the depreciation in any one year to the book value at the beginning of that year is constant throughout the life of the asset and is designated by 'k'. Thus

Depreciation during the nth year :

$$d_n = (C_{n-1}) * k$$

Where

$$k = 1 - \sqrt[n]{C_n/C}$$

(3) The Sum-of-the-Years'-Digits Method

In order to obtain the depreciation charge in any year of life by this method, the digits corresponding to the number of each year of life are listed in reverse order. The sum of these digits then is determined. The depreciation factor for any year is the reverse digit for that year divided by the sum of the digits. The general expression for the annual cost of depreciation for any year 'n', is as follows :

$$d_n = (C - S) * \frac{2(L - n + 1)}{L(L + 1)}$$

(4) The Sinking Fund Formula

The sinking fund formula assumes that a sinking fund is established in which funds will accumulate for replacement purposes. The total depreciation that has taken place up to any given time is assumed to be equal to the accumulated value of the sinking fund at that time. In this manner the capital is preserved. With this formula, if the estimated life, scrap value, and interest rate on the sinking fund are known, a uniform yearly deposit can be computed. Thus

$$d = (C - S)(A/F, i\%, L)$$

$$D_n = (C - S)\frac{(A/F, i\%, L)}{(A/F, i\%, n)}$$

Where (A/F,i%,L) and (A/F,i%,n) are known as Sinking Fund Factors for corresponding years of L and n and is expressed as follows :

$$\frac{1}{(1 + i)^N - 1}$$

in which i = interest rate in percent

N = number of interest periods

APPENDIX 7

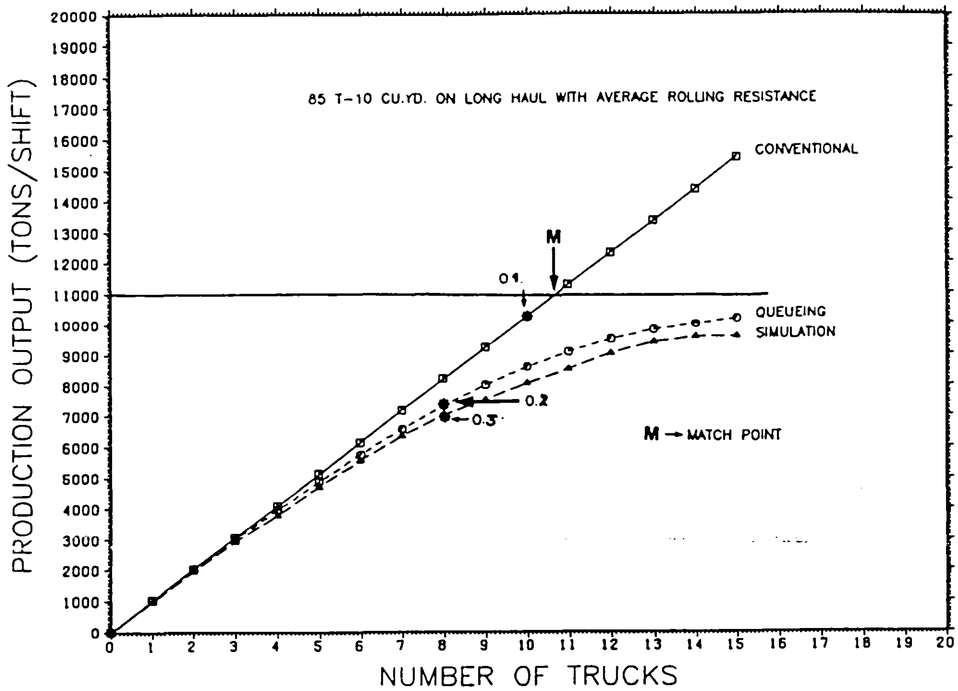


FIG.1 METHOD COMPARISON ON PRODUCTION

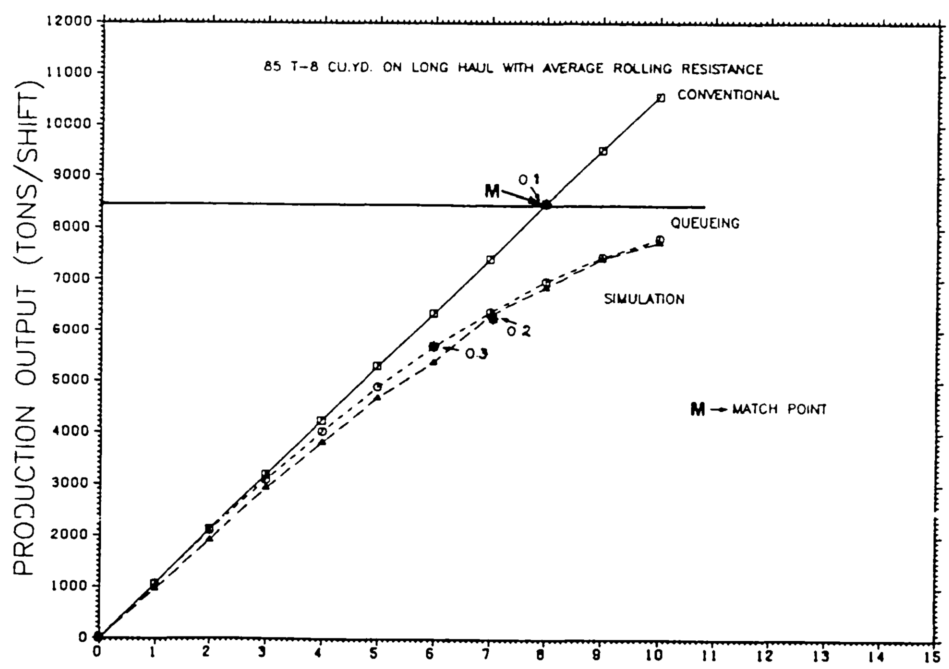


FIG.2 METHOD COMPARISON ON PRODUCTION