FOREST ROAD PLANNING AND CONTROL BY MATHEMATICAL PROGRAMMING AND TIME STUDY ANALYSIS

by

Willbard Shoo Abeli
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This thesis submitted by Willbard S. Abeli in fulfillment of the requirements for a Ph.D. degree award has been examined by:

....
Professor D. P. Dykstra
Internal Examiner and Supervisor

....
Professor M. Saarilahti
Internal Examiner

....
Professor Dr. Hans Löffler
External Examiner

DECLARATION

I, WILLBARD SHOO ABELI, do hereby declare to the Senate of the Sokoine University of Agriculture that this thesis except where otherwise acknowledged is my original work, and it has not been submitted for a degree award in any other University.

Signature: ....

Date: 26/11/85

ABSTRACT

Planning and construction of forest roads have until very recently been geared towards establishment and tending of plantation forests. Now that most of the plantations are nearing maturity, it is essential to plan and construct proper roads for harvesting purposes.

This study was designed to develop a comprehensive methodology that can be used by forest managers in the planning and control of forest roads. Specifically, the objectives of the study were:

- To evaluate the application of mathematical programming methods to the design of safe and minimum-cost forest roads
- To test the feasibility of improving and controlling forest-road construction by mathematical programming
- To develop a comprehensive methodology for measuring and estimating production rates and costs associated with forest road construction.

To be able to plan, execute and control work operations for this study, a representative 1.3 km access road was planned and constructed at the Training Forest of the Sokoine University of Agriculture between February 1981 and May 1984. A number of mathematical models and time study methods were tested and incorporated in this study in order to develop a comprehensive methodology aimed at improving planning, control and cost estimation procedures. Based on the results of this study, the following observations and conclusions are made:

(a) With some modifications (in order to fit them to local conditions), the standard engineering procedures used for road design, layout and culvert size determination proved to be fully satisfactory for use in Tanzania.

(b) Detailed time and work analyses conducted during the study resulted in the establishment of production rates and costs of forest road construction in Tanzania.

(c) Microcomputer-based linear programming and a transportation tableau method were shown to be equivalent in determining optimal allocation of earthwork during forest road planning. Because it can be done by hand without recourse to a micro-computer, however the transportation tableau method is the one likely to be favoured and adopted by most forest projects in Tanzania.

(d) Before the actual road construction starts, the <u>distance</u> between net cut and net fill areas can be used as a surrogate for earthwork cost to determine the optimal road design. The study found that the design alternative selected on the basis of minimum earth moving <u>distance</u> was also the one that minimized earthwork <u>cost</u>.

(e) A distance of 30 m between instrument stations along the road appears to provide better accuracy for road layout and design than using either intervals of 60 m or utilizing "terrain breaks" to define instrument stations. In this study, both of these latter interval lengths provided data which tended to underestimate actual road construction cost by about 30 %.

(f) The influence of road grade and alignment on log-truck hauling speed can satisfactorily be determined through the principles of engineering mechanics. For instance, when determining the hauling speed over a given road with a good number of sharp curves, time to decelerate into and accelerate out of the curve, the designed curve speed and the maximum safe speed have to be considered.

(g) Of three road grade alternatives analysed in this study (6 %, 8 % and 10 %), the 8 % maximum grade recommended by the Forest Division was found to be the grade that minimized the total of earthwork and hauling costs.

(h) The use of the critical path method during culvert installation resulted in the reduction of total installation time by about 28 %.

(i) Road construction is an expensive undertaking, but costs can be reduced to about 1/3 to 1/2 of those estimated by road contractors if proper planning and control procedures are applied. The construction cost of this research road was found to be shs 220 587/km, with earthwork and gravelling operations taking 33 % and 30 % of the total construction costs respectively, while road alignment/clearing and culvert installation took 24 % and 13 % respectively.

In addition to these conclusions, the methodological procedures described in detail in this thesis provide a foundation on which forest managers can base decisions related to the planning and construction of forest roads.

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CHAPTER ONE

INTRODUCTION

About half of Tanzania's total land area (945 000 km^2) is classified as forest land. Within the forest land, three types of forests are distinguished: miombo woodland forests, tropical high forests and mangrove forests, and man made forests (Dykstra 1983a).

Miombo forests, which cover over 97 % of the forest land, are characterized by low stocking, low annual volume increment, unpredictable rains, a long dry season and frequent bush fires. Important hardwood timber species harvested from this forest type include *Pterocarpus angolensis*, *Dalbergia melanoxylon*, *Brachystegia* spp., *Julbernadia* spp. and *Afzelia quanzensis* (Abeli and Dykstra 1981, Dykstra 1983a).

Rainforests, comprising tropical high forests (confined mainly to mountain areas) and mangrove forests (common along the coast) have largely been set aside as water catchment areas. Where harvesting is still going on, rainforest tree species of commercial importance include *Ocotea usambarensis*, *Olea capensis*, *Khaya nyasica*, *Chlorophora excelsa* and *Brachylaena hutchinsii* (Dykstra 1983a).

There are altogether 17 industrial forest plantations in the country covering a total area of about 70 000 ha (Dykstra 1983a). Productivity in these plantations is relatively high compared to natural forests; i.e., annual volume increment is 25-35 m^3/ha/year opposed to only 2-3 m^3/ha/year in miombo forest (Abeli and Dykstra 1981, Adegbehin and Philip 1979, Dykstra 1983a). The important tree species grown in these plantations include *Pinus patula*, *Cupressus* *lusitanica*, *Pinus caribaea*, *Pinus elliottii*, *Eucalyptus* spp. and *Tectona grandis*.

A study carried out by Jaakko Pöyry (1979) showed that there is little potential for maintaining and increasing production of industrial wood from the natural forests. This is due to low stocking, unaccessibility and scatteredness of the natural forest blocks. The study therefore emphasized the need to develop and expand production of industrial timber from plantation forests.

Production and consumption of industrial softwood is estimated at 630 000 m^3 per year between 1985 and 2000 (Jaakko Pöyry 1979). Market projections for the domestic demand for all wood products suggest an increase at an average annual growth rate of 5 to 12 per cent up to the year 2000. Also some surveys have shown that with adequate industrial and logging resources, about 4-5 times the present timber could be harvested from the existing industrial plantations. However, one of the major bottlenecks on the supply side which has been singled out frequently is inadequate and poor forest roads (Forest Division 1981).

1.1 Plantation forest roads

In many of the plantation forests, forest roads are inadequate and even the few existing ones are not properly aligned, constructed or maintained (Forest Division 1981). Most of these roads were built during the plantation-establishment phase and until recently, have largely been used for silvicultural and administrative purposes. Therefore, since most plantations have reached a state of maturity, the need to improve, plan and construct proper logging roads is urgent.

In 1981, following the recommendations put forward by a Joint Tanzania/Nordic Forestry Review Mission (Forest Division 1981), a Logging and Road Building Unit (LRBU) was established. This unit, stationed at Meru forest plantation, was to take over from what used to be called the Northern Forest Road Construction Unit (NFRCU) formed in 1972 (Forest Division 1972). The main objective of NFRCU was to plan and construct forest roads in Meru, Usa, West Kilimanjaro, North Kilimanjaro and Shume forest plantations. Unfortunately, due to lack of spare parts and personnel, this unit collapsed in the mid-seventies.

Besides the LRBU appropriating the few old machines left over by the NFRCU, the unit has also acquired some new road construction machines from Sweden. So far it has constructed over 22 km of access roads in Meru forest plantation and improved over 48 km of access roads in West Kilimanjaro forest plantation (Chilembu 1984). In relation to the total length of forest roads required in all forest plantations in the country, i.e., 3000 km (Forest Division 1981), this rate of construction is very low.

Since the capacity of the LRBU is small in relation to the total road length required, there is a need for each forest plantation to plan and construct its own forest roads or encourage wood using industries to construct forest roads by giving them rebates on royalties and exclusive licences. However, whether the roads will be constructed by the forest plantations or by the wood using industries, planning and supervision of road construction has to be undertaken by personnel from the Forest Division who are conversant with road planning and construction principles.

Besides the Technical Order issued in 1971 on description and specifications for constructing main forest roads (Forest Division 1971), no detailed planning procedures, construction methodologies or cost estimation techniques have been adopted for the planning and construction of forest roads in Tanzania. Studies aimed at identifying and specifying the proper planning and construction methodologies and cost estimation are necessary. From such studies, appropriate planning methodologies and cost estimation procedures under different terrain conditions can be formulated. This will not only assist road engineers and construction personnel but will also standardize the planning technique, construction methodologies and cost estimation procedures in all forest plantations.

1.2 Planning and control of forest roads

The design of forest roads all over the world has relied largely upon the individual skill of the engineer to minimize costs. Techniques such as mass-haul (Moffit and Bouchard 1975), the transportation method of linear programming (Boughton 1967) and network analysis

(Dykstra 1984) have been developed to assist engineers in estimating construction costs and evaluating effects of design modifications on the cost function. Some of these techniques have proven dramatically successful in reducing highway construction costs but unfortunately they have not been applied in Tanzania in planning and construction of forest roads.

Since the above techniques have not been used in Tanzania, this study is designed to provide an initial evaluation of such techniques in planning and construction of forest roads. The individual techniques to be applied are not new nor is their application to road planning and construction. However, their integration into a comprehensive planning and control methodology is an important and innovative step, not only to Tanzania, but in forest road engineering generally (Dykstra 1982d).

Furthermore, the proposed use of time study methods to obtain current production and cost data on forest road construction is in itself an important undertaking. No such data exist in Tanzania at present except in the form of constructors' estimates whose reliability may be regarded as questionable at best. If planning and control of forest roads is to be undertaken seriously, then such data must be obtained. More importantly, effective procedures for data capture and analysis must be developed and documented if any planning and control methodology is to be useful to forest project engineers.

Hitherto there has been no published evidence that detailed time studies have been used to obtain data for forest road planning and control although the application of time study methods to forest operations has ranged from nursery activities and harvesting operations to the assessment of ergonomic personnel (Abeli and Dykstra 1981, Dykstra 1975, Heding and Meiludie 1979, Skaar 1973, Wittering 1973).

While it seems almost certain that time studies have been made on forest road construction activities, the fact that those studies have not been published suggest that they have not been utilized as part of a formal planning methodology. Since time study is a useful tool in measuring work so as to provide data for more effective planning of the subsequent operations, it is used in this study to:

(i) obtain accurate production and cost data for the road to be constructed

(ii) permit the development of procedures for incorporating time study methods into a formal methodology for planning and construction of forest roads.

Data for road planning and construction could be obtained from private or public road contractors. However, the reliability of such data is questionable due to poor data recording and the unwillingness of the contractors (especially the private ones) to reveal true costs. Even the data obtained from a time study of a contractor's operation is of limited use because it tells how the operation <u>was</u> done, not how it <u>might</u> or <u>ought</u> to be done. This is due to lack of influence or control over the contractor's operation. Thus, to be able to plan and control the whole operation in a more systematic and controlled manner and to obtain realistic data on both costs and production rates through time study, a representative 1.3 km forest access road was planned and constructed in this study. This road segment constituted an extension of the present access road serving the upper end of the University Training Forest near Arusha.

1.3 The objectives of the study

The primary goal of this study is to provide guidelines to forest managers on the application of improved techniques for planning and control of forest roads. The specific objectives of the study are as follows:

- To evaluate the application of mathematical programming methods to the design of safe and minimum cost forest roads.
- To test the feasibility of improving planning and control of forest road construction by mathematical programming.
- To develop a comprehensive methodology for measuring and estimating production rates and costs associated with forest road construction.

CHAPTER TWO

LITERATURE REVIEW

2.1 General

Forest roads are constructed not only to facilitate the extraction of forest products but also to provide access for the management and protection of forests (Samset 1975). Forest roads, unlike public roads or highways, are low-speed and light-traffic roads. In forestry, three classes of roads are distinguished: connecting or main roads, access roads and branch roads (Lundquist 1979).

Connecting roads are constructed to connect forest areas with utilization centres while access roads are the links between the forest and connecting roads. Access roads are either classified as main roads or secondary roads depending on the number of lanes, surface layer thickness, quality of subgrade and other factors. Since the main task of an access road is to open up the forest area, in hilly or mountainuous areas such a road is normally a "climbing" road (Dykstra 1981a). Branch roads are normally temporary roads of low standard that are used to connect individual forest compartments with access roads.

In actual practice, it is seldom possible to design a road that achieves the objectives of all concerned (Peurifoy 1979). However, it has been the goal of forest engineers to design roads that most nearly satisfy the needs of the forest and public at the lowest practical cost. For example, forest engineers have been able to develop planning techniques which are less time consuming and considerably less expensive yet quite adequate. The use of a Suunto clinometer and a levelling staff to determine height differences instead of a theodolite is just one example (Peurifoy 1979). In other words, engineers have endeavored to determine ways and means of reducing costs without unduly reducing the service which the road is intended to furnish (Dykstra 1983b).

The routing of a road is mainly determined by social-economical consideration and by the topography of the area which the road is to access. Within these limits, the choice of alignment is influenced by the design specification and construction technologies which are adopted (ILO 1979).

Construction of forest roads, especially on mountainuous areas, is costly and accounts for a significant portion of the total harvesting cost even though the initial investment is written off over many years (Boughton 1967). Because of the relatively large capital investments associated with road construction, efficient planning of forest roads is of considerable importance (Dykstra 1983b).

Road construction methods and costs vary throughout the world. Where construction is mechanized, the cost of equipment amounts to about 75-80 % of the total construction costs while labour and material costs range between 10-15 % and 5-10 % respectively (Edmonds and Howe 1980). On the other hand, where construction is semi-mechanized, the equipment cost is largely offset by the labour cost. This is the case in many developing countries where unemployment is high and low-cost manpower is plentiful. In Iran for example, Edmond and Howe (1980) found that the cost distribution for a semi-mechanized road were: labour 42.5 %, equipment 52.9 % and materials 4.6 %.

Construction of forest roads means interfering with the ecological system and the loss of productive forest area (Chandra 1982). FAO (1977) estimates that for a forest area of 1000 ha with road density of 20 m/ha, the area required for roads with a right of way width of 20 m equals 40 ha or 4 % of the total area. Inaccurate planning and construction of forest roads may contribute to land slides and soil erosion which result in silting of streams and rivers. The magnitude of erosion depends on the physical properties of the soil, vegetation cover, gradient and the length of the slope (Pearce and Stenzel 1972).

According to Gignac (1962), when planning for road construction over a new forest area, the following considerations are of primary importance:

(i) What kind of road and where to locate the road to provide the best access to the merchantable timber.

(ii) How to take the best advantage of the most favourable land features in order to keep construction and maintenance costs to a minimum.

One major objective of road planning is to find out the type of road to construct which minimizes total construction and transport costs (Chandra 1982). As one component of planning for the overall transportation system, road planning must give special attention to interactions between the transport routes and transportation equipment with the objective of minimizing the overall transport costs (Dykstra 1981a). Experience has shown that extra effort taken during the planning stage can often reduce the construction costs by one half or more (Dykstra 1983b). In other words, a good road starts with a good plan.

2.2 Road design

Before a road survey is undertaken, the standard of the road has to be specified since this will determine the limiting specifications for gradient, curvature, turnouts, surfacing, bank slopes, travel speed and the type of drainage structures (Pearce and Stenzel 1972). Also according to Pearce and Stenzel (1972), factors which influence the determination of the road standard include construction and maintenance costs, truck hauling costs and the volume of timber over which the cost of the road is to be amortized.

Because of topographic features and other obstacles, roads usually deviate from straight lines by means of curves. Although the purposes of curves are to avoid excessive earthworks, they are costly to construct and maintain. In addition, since curves hamper visibility, driving on curves is dangerous. In Canada for example, between 1955 and 1956, it was found that 50 per cent of the fatal accidents were attributed to curves (Gignac 1962).

When driving around a curve, the centrifugal force tends to push the vehicle off the road. Its magnitude depends on the radius of curvature, vehicle weight and speed. To compensate for this centrifugal force and for the off-tracking of a long vehicle's rear wheels, curves are usually super-elevated and widened. Formulas to determine the extent of curve widening have been developed by Dykstra (1983b), FAO (1977), Sessions (1976) and Smith (1973).

While it is always advisable to avoid steep gradients, occasional steep grades for short stretches are often necessary if excessive earthworks and the frequency of curves are to be minimized. For these reasons, Forest Division (1972) allows gradients up to 8 % over short distances on main access roads.

Vehicles driving on steep grades experience high running and maintenance costs (Dykstra 1981a) and run high risks of accidents and breakdowns. The amount of damage caused by trucks on steep grades increases progressively, especially on curves, while load sizes decrease progressively with a linear increase in grade (FAO 1977, Gignac 1962). Also it is known that as the road grade increases, the eroding power of water running down the road increases progressively; i.e., if the speed of running water doubles, its eroding power increases four times (Dykstra 1981a).

Roads are usually surfaced to provide traction, protection and distribute the load. The surface layer has to be thick enough to distribute the applied loads of traffic so that maximum pressure reaching the subgrade does not exceed its bearing strength. Factors which determine the thickness of surface layer include the characteristics of the subgrade, the loads to be hauled, traffic density, design speed and the surfacing materials (Cron 1975, Huang 1959, McFarlane 1963).

The ability of the surface layer to bear stress, to resist mechanical wear and to withstand climatic forces depends upon the quality and character of soil aggregate (Huang 1959). As a road surface material, coarse-grained materials (sand 0.06-2.0 mm and gravel 2.0-60 mm) are considered to be more satisfactory than the fine grained ones (silt 0.02-0.06 mm and clay less than 0.02 mm). Skaar (1972a) recommends that soil aggregate containing more than 10 % of clay or fine silt should never be used as a surface layer material. Any material of a granular nature which does not easily soften when wet or break down under traffic can be employed as a surface layer material. The soil aggregate material most commonly used as a surface layer in forest roads is a pit-run or river gravel. Pit-run gravel is a natural mixture of gravel, sand, silt and clay. It is of such quality that it is used without or with only minor processing (Huang 1959).

Water reaches a roadway principally through the precipitation of rain, by surface flow from adjacent lands, by floods in streams and by percolation through the soil. This water may cause detrimental erosion resulting in an unsightly appearance, clogging of ditches and drainage structures, and filling of streams and rivers with unwanted sediment (Akesson 1963, Pearce and Stenzel 1972).

Experience has shown that drainage is one of the most important considerations when building forest roads. Surface drainage is accomplished by constructing roads with crowned road surfaces, shoulder slopes, side slopes, water bars, intercepting ditches at the top of cut slopes, side ditches, culverts and other drainage structures (Akesson 1963, Huang 1959, Pearce and Stenzel 1972).

Most of the side ditches encountered in forest roads are of the "V" type and are constructed mainly to intercept water from the surrounding area and from the road surface. Water along these ditches is usually led away from the roadway by means of mitre drains or ditch relief culverts sited at appropriate intervals (FAO 1977).

The spacing between ditch relief culverts depends upon the gradient of the ditch and the erosion index of the soil. The size of these culverts on the other hand depends on the topography, soil type, type of forest cover and the area of the watershed. It also depends on the frequency and the magnitude of sudden rain storms and the culvert spacing (FAO 1977, Pearce and Stenzel 1972, Simmons 1979).

Three types of culverts in common use in forestry are: wooden culverts, concrete culverts and steel culverts. Wooden culverts are used on low-standard forest roads while concrete culverts are the most widely used type of culvert on forest roads. Where concrete culverts are over 60 cm in diameter, they are usually steel re-inforced (Skaar 1972b).

In order to ensure lateral road drainage, both main and secondary access roads are crowned on tangent sections and superelevated on curve points. The recommended crown slope for hard packed gravel roads is 2-3 % while for loose, coarse and rough gravel roads it is 4-5 % (FAO 1977).

2.3 Earthworks

Traditionally, heavy machines such as bulldozers have been used in clearing the right of way, earthmoving and construction of subgrades for forest roads (Caterpillar 1981). Recently, especially where environmental laws are very strong, hydraulic backhoe machines have been introduced to the forests for road construction (Stjernberg 1982).

According to Caterpillar (1981), the weight and engine power of the machines, the nature and conditions of terrain, co-efficient of traction and the type of dozer blade determine the ability of the machine to push and excavate soil. Of the two types of blades, "U" type and "Straight" type, the latter is the most preferred due to its versatility and aggressiveness in penetrating and obtaining a blade load.

Earthmoving by bulldozer tractors constitute a major cost in road construction. Although several planning techniques aimed at reducing earthwork costs have been suggested, so far there is no published evidence that they have been used in the design and construction of forest roads. Boughton (1967) observed that despite the fact that earthwork costs constitute a major cost in road construction, earthworks rarely receive any scientific planning directed towards reducing costs. He suggested that the movement of materials from a region of excess cut to regions requiring fill is a problem of allocation which could be solved by the transportation method of linear programming in order to minimize earthwork costs.

Transportation problems are special types of linear programming problems concerned with distribution of resources from supply centres called sources to receiving centres called destinations in such a way as to minimize total distribution cost (Hillier and Lieberman 1980, Phillips *et al.* 1976). Applied to earthwork minimization, the transportation problem seeks to determine the allocation of earthwork from excavation sites to fill areas so that the total cost associated with this allocation is minimized.

In transportation problems, special solution techniques which have advantages over more generalized LP solutions have been developed. As a consequence, very large transportation problems can be solved by hand without resorting to the use of computers (Dykstra 1984).

Successful management of large-scale projects like road building requires careful planning, scheduling and co-ordination of numerous inter-related activities. To aid in the accomplishment of such projects on time, formal procedures based on the use of networks and network techniques have been developed. The most prominent of these procedures are the Programme Evaluation and Review Technique (PERT) and the Critical Path Method (CPM). PERT is applied where there is uncertainty in predicting activity times while CPM is appropriate when activity times can be predicted and adjusted based on previous experience (Dykstra 1984). Because of its relative simplicity, CPM is the more commonly used technique in both construction and maintenance operations.

CPM aids in solving problems connected with project planning and control. It identifies the activities that are most likely to be the bottlenecks and reveals areas where the greatest effort should be expended in order to stay on schedule. In addition, CPM graphically portrays the inter-relationships among the activities of a project by showing the precedence relationships and the order in which activities must be performed (Dykstra 1984, Hillier and Lieberman 1980). According to Dykstra (1984), examples from the forestry literature where CPM has been applied include the planning and control of logging operations, timber sale preparation, forest road construction, sawmill modernization and slash burning.

2.4 Time study

Modern time study was originated in 1881 by Fredrick W. Taylor who used it to set work standards. Later, together with Frank Gillbreth, he devised the theory of time and motion study which gives a basis for measuring work production rates and costs over a period of time (Dykstra 1982a, Migunga and Dykstra 1983).

Time and motion studies are imperative if large and small scale planning and control of operations are to be effective. Forest operations have been widely studied (Migunga and Dykstra 1983) using either shift level time study or detailed time study. The two methods differ in purpose, methodology and in the type of analysis to which data may be subjected (Dykstra 1982a).

2.4.1 Shift level time study

Shift level time study, also known as gross data analysis, measures and records the production levels achieved by a work system, crew or machine working over a given period of time. The method is relatively cheap and requires no precise timing and trained time study analyst (Migunga and Dykstra 1983, Olsen and Kellogg 1983).

The purpose of a shift level time study is to monitor long term trends in productivity and costs and to provide information for planning and control of operations, i.e., by showing where detailed time studies are required (Anderson 1976, Dykstra 1982a, 1977). A major difficulty with this method is its requirement on lengthy study period in order to accumulate enough data to be analysed. Also, it is difficult or impossible to determine whether the results obtained from one study apply to conditions other than those investigated (Migunga and Dykstra 1983).

2.4.2 Detailed time study

Detailed time studies require the employment of a trained time study analyst. They are usually undertaken where there is a need to obtain information about work method, workers' productivity, operating costs and procedures beyond the level of detail available from shift level studies (Dykstra 1982a).

During detailed time study operations, the work cycles are broken down into individual work elements and studied independently. These work elements are made short and defined clearly so that they can be recognized by the time study personnel. Segregating work cycles into work elements helps in making comparisons between elemental times for one operation and those obtained for similar elements in another operation. It also allows the time study analyst to isolate time consuming elements which can be subjected to more comprehensive method studies (Dykstra 1982a).

Most detailed time studies in forestry are done with stop watches. Although specialized equipment such as time-lapse movie cameras (Gardner and Schillings 1969, Olsen and Kellogg 1983), video tape units (Dykstra 1982a) and special digital recording equipment (Dykstra and Howard 1980) have been used in conducting detailed time studies, the stop watch remains the most reliable, least expensive and the most flexible type of equipment to use. Depending on the purpose of the study, there are two common timing procedures used during detailed time study: <u>continuous timing</u> and <u>activity sampling</u>.

2.4.2.1 <u>Continuous timing</u>

Continuous timing, which is done continuously from the start to the end of a working shift, is the most common type of detailed time study method. Under this method, which records both regular and irregular events plus delays, two timing methods are distinguished: cumulative timing and snapback timing.

Cumulative timing keeps a record of the sequence of activities in addition to providing unbiased measurements, especially where productive and unproductive times are to be studied (Anderson 1976, Dykstra 1976, 1977, Migunga and Dykstra 1983). Its chief drawback is the fact that the elapsed time of each activity must be determined later by subtraction. This makes field checking of the sensibility of activity times difficult.

During snapback timing, the observer starts the stop watch at the beginning of each activity and at the conclusion of the activity, the elapsed time is recorded and the stop watch reset to zero (Barnes 1963, Wittering 1973). This method is less prone to record-keeping errors than cumulative timing although it does not reveal the sequence of a day's activities (Migunga and Dykstra 1983).

Snapback timing has been used in Tanzania for studying logging and log transport operations. Abeli and Dykstra (1981) and Migunga and Dykstra (1983) have used this method to develop methods of estimating production rates and costs of different cutting methods while Abeli and Dykstra (1981), Heding and Meiludie (1979) and Ole-Meiludie and Dykstra (1982) have used the method for developing nomograms and regression equations which are used for estimating skidding production. Log transport operations have also been time studied by the snapback method (Abeli and Dykstra 1981, Nyama 1985).

2.4.2.2 Activity sampling

Activity or work sampling involves observing the operation at fixed time intervals or at random intervals (Olsen and Kellogg 1983) established from a table of random numbers. Depending upon the type of work, the level of precision desired and the number of operations being timed by the observer, observations are usually recorded at intervals of 1 minute, 5 minutes or 30 minutes. Studies have shown that this method offers economy at no theoretical sacrifice of accuracy (Gardner and Schillings1969, Migunga and Dykstra 1983). Its primary disadvantage is that the data are not generally suitable for regression analysis and the study results cannot therefore be transferred easily to other sites.

2.5 Estimating machine costs

All machine parts are subject to wear or failure regardless of the care they receive. It is important that the purchaser of an implement consider the ease and speed of obtaining replacement parts before deciding on the type of an equipment or machine to buy. Usually standard equipment which can be used in various operations and its spare parts obtained easily should be preferred to specially designed machines (Peurifoy 1979).

The life of a machine depends on the operating conditions, the level of training of the operators and mechanics, the availability of replacement parts, obsolescence and climatic conditions, especially in the tropics (FAO 1977).

The cost of owning and operating a machine depends on the purchase price of the machine and delivery costs, the severity of the conditions under which it is used, the number of hours used per year, the number of years in use, the care with which the owner maintains and repairs it and the demand for used machines when sold (Peurifoy 1979).

There are several methods of determining the cost of owning and operating machines. No known method will give exact costs under all operating conditions - at best the estimate is only a close approximation of the cost (Peurifoy 1979). Perhaps the most accurate method of estimating machine cost would be to obtain cost data from previously kept records on a similar type of machine. Unfortunately such carefully kept records hardly exist and there is no assurance that a similar machine will incur similar costs, especially if the machine is used under different working conditions. As such, several standard machine costing formulas have been developed in order to estimate machine owning and operating costs (Dykstra 1982b, FAO 1977).

The cost of a machine may be divided into fixed costs and variable costs. Fixed costs include depreciation, interest, taxes and insurance costs or other costs which do not vary with the level of production. Variable costs on the other hand depend on the level of production and include costs of fuel, oil and lubricants, maintenance and repair, and tyres (FAO 1977).

Machine depreciation is of two types: physical depreciation and economic depreciation. Physical depreciation results from physical impairment of the asset due to aging, while economic depreciation results from reduced demand for the asset's services over time, often because of obsolescence (Dykstra 1982b). To compute depreciation costs, the straight-line method (Dykstra 1982b, FAO 1977) and capital recovery formula (ILO 1979) are most frequently used. The straight-line method assumes that an asset declines by a fixed amount each year while the capital recovery formula is a means of recovering the original investment on the machine by considering both depreciation and interest costs.

Interest charges on the money used in purchasing a new machine must be considered since funds tied up in the new machine could otherwise be deposited in a bank and earn regular interest (Dykstra 1982b). In forest operations analysis, interest charges on machines are based on the average annual investment (AAI) in the machine (Dykstra 1982b, FAO 1977).

Insurance is a payment intended to cover public liability, property damage or loss through theft, fire, accidents and other hazards. Its magnitude depends on the value of the machine, commonly ranging between 1 and 5 per cent of AAI (FAO 1977, Ole Meiludie 1984).

Taxes on property are fixed by the government and typically range between 0.5 and 2 per cent of AAI (Dykstra 1982b, Ole Meiludie 1984).

Usually fuel cost is estimated from log books kept for each machine.

Where records are non-existent, standard formulas are often used (FAO 1977). Fuel consumption depends on engine power and the load factor. A load factor of 50 per cent is used to estimate fuel consumption for machines operating under ordinary working conditions (FAO 1977). In case of oil and lubricants, consumption depends on the type of machine and gross engine power. For machines like tractors, trucks and skidders the rate of oil and lubricant consumption is typically about 0.0027 litres per gross kilowatt -hour (FAO 1977, Ole Meiludie 1984).

Although maintenance and repair cost of a machine actually increases with age, it is normally averaged over the machine's life and expressed on a "straight-line" basis in the same manner as depreciation. Experience has shown that maintenance and repair costs vary between 75-150 per cent of depreciation depending on the complexity of the machine, the type of work it is engaged in, its working milieu and the operator (FAO 1977, Laarman *et al.* 1981).

The cost of tyres for hauling trucks are typically accounted for on a mileage basis or on the basis of travelling hours. In forest operations, because of the different types of roads encountered, tyre costing is more commonly based on the travelling hours rather than on mileage basis (FAO 1977).

Labour engaged in operating a forest machine may either be permanent or casual labour. For permanent labour, labour costs include direct wages paid on time, task or piece work and fringe benefits. Fringe benefits are expressed as a percentage of direct wages and in the case of Tanzania, it is estimated to vary between 20 - 30 % (Dykstra 1982b), although in some cases it is higher. In the case of casual labour, a fixed amount is paid daily based either on time, task or piece work.

CHAPTER THREE

METHODOLOGY

3.1 Description of the study area

The Training Forest of the Sokoine University of Agriculture (SUA) comprises about 680 ha of softwood plantation near Arusha. The forest consists of two blocks; Narok block on the lower part and Laikinoi block on the upper part of the forest (Fig. 1). The main tree species grown here include; *Cupressus lusitanica* Mill. (50 %), *Pinus* spp. (42 %), *Eucalyptus* spp., *Grevelia robusta* A. Cunn. and *Acacia melanoxylon* R. Br.

The forest lies on the slopes of Mount Meru about 1740-2320 m above sea level. The slopes are generally steep (sometimes exceeding 20°) with dissected stream courses. The soils are volcanic, free-drained silts which become greasy during the rainy season and powdery during the dry season. The climate is very seasonal with a marked and consistent five-month dry season from June to October. Rainfall patterns also vary considerably but on average, the annual precipitation exceeds 750 mm (Division of Forestry 1979).

The main objectives of the University Training Forest are:

- To provide facilities for training students in forest practices and management
- To provide an area and facilities for forest research
- To manage the forest profitably and supply a sustained yield of forest produce to the local wood-based industries and people.

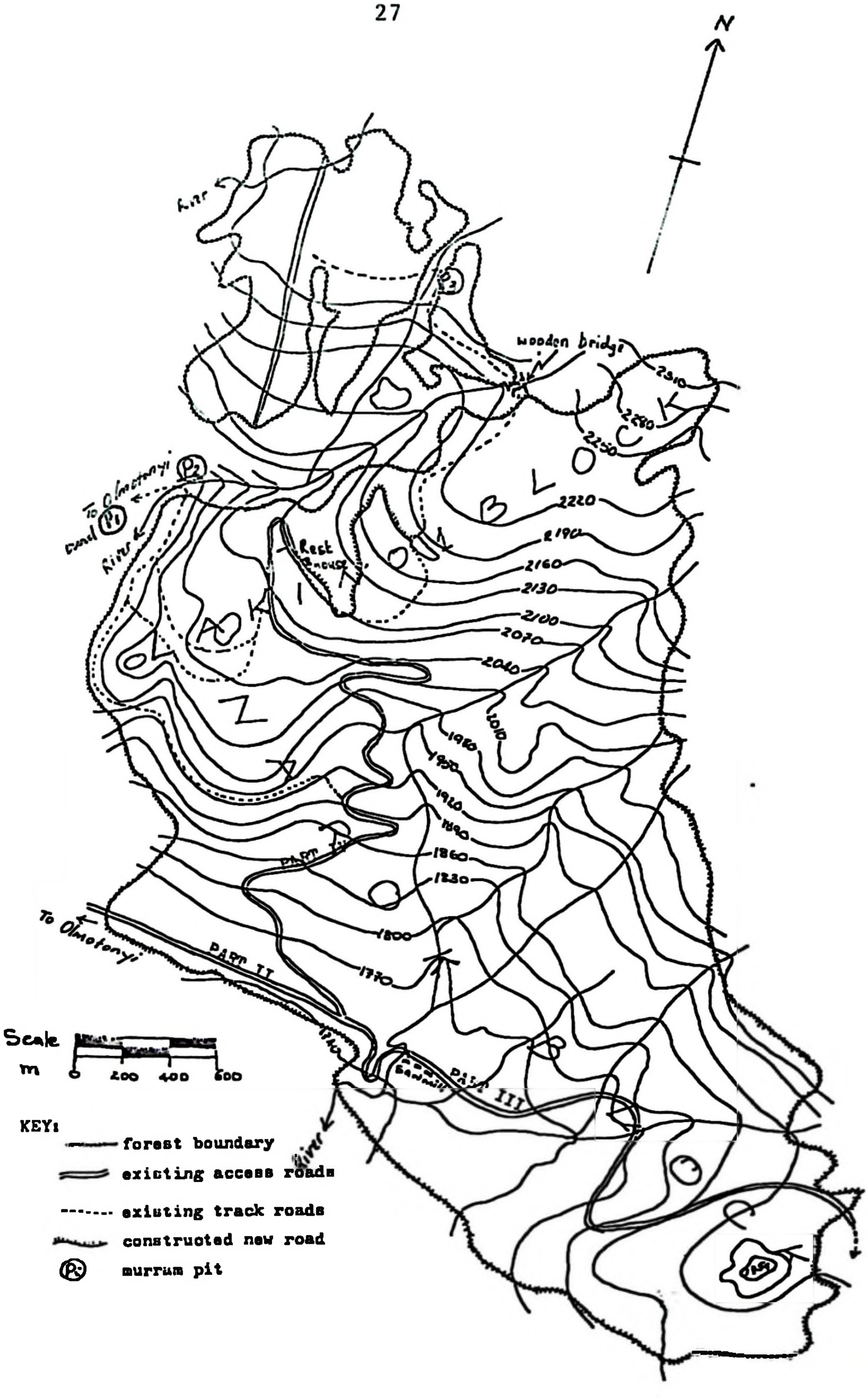


Figure 1. A contour map of the SUA Training Forest showing the existing roads, the constructed new road plus the tracks. (Sources: Survey and Mapping 1972, Division of Forestry 1979).

When the Training Forest was transferred to the Sokoine University of Agriculture from Meru Forest Project in 1978, the roads leading to and within the forest were untrafficable. The simple tracks built in the early 1960's to exploit the natural forest and to tend the young plantations provided only rudimentary access to the forest. This led to delays in thinning operations and the general exploitation of timber from this forest.

The SUA Training Forest offices are located 7 km from the forest. To be able to reach the forest and exploit timber, the University had to construct a number of access roads. Thus, between 1979 and 1981 a total of 15 km of secondary access roads were constructed through a grant from the Norwegian Agency for International Development at a cost of about shs 6.6 million or shs 440 000 per km.

Despite all the efforts by the University to cover the whole forest with access roads, the access road on the Laikinoi block ended 4 km from the edge of the forest. This meant that an area of about 183 ha with a standing tree volume of about 82 700 m^3 (Hamza 1983) was yet to be served by an access road. This prompted the idea of constructing this road as a research project in order to serve this area. Figure 1 shows a contour map of the SUA Training Forest with the existing road network, the constructed new road and tracks leading to the murram pits.

3.2 Description of the road standard

This research road being an extension of the existing Laikinoi access road (Fig. 1), the Management Board of the Training Forest decided that they should both be of the same standard. Figure 2 shows a typical cross-sectional profile of the existing Laikinoi access road. Details of the construction standards of the existing Laikinoi road are given in the following paragraphs.

3.2.1 Design gradient

The ruling or average designed gradient is 5 %. In some portions of the road, the gradients are as steep as 9 % but only for short distances. Since trucks are empty when traversing the adverse grades and loaded when going down the favourable grades, hauling speed is not limited by either gradient or engine power but rather by road alignment.

3.2.2 Design speed

For safety reasons and due to the nature of the road alignment, the Forest Manager of the Training Forest has set a maximum speed of 40 km per hour on tangent sections both when travelling empty and loaded.

3.2.3 Turnouts

Since the road has the width of only a single lane, turnouts are provided to permit vehicles to pass when they meet. The turnouts vary in length from 10-15 m depending on their location.

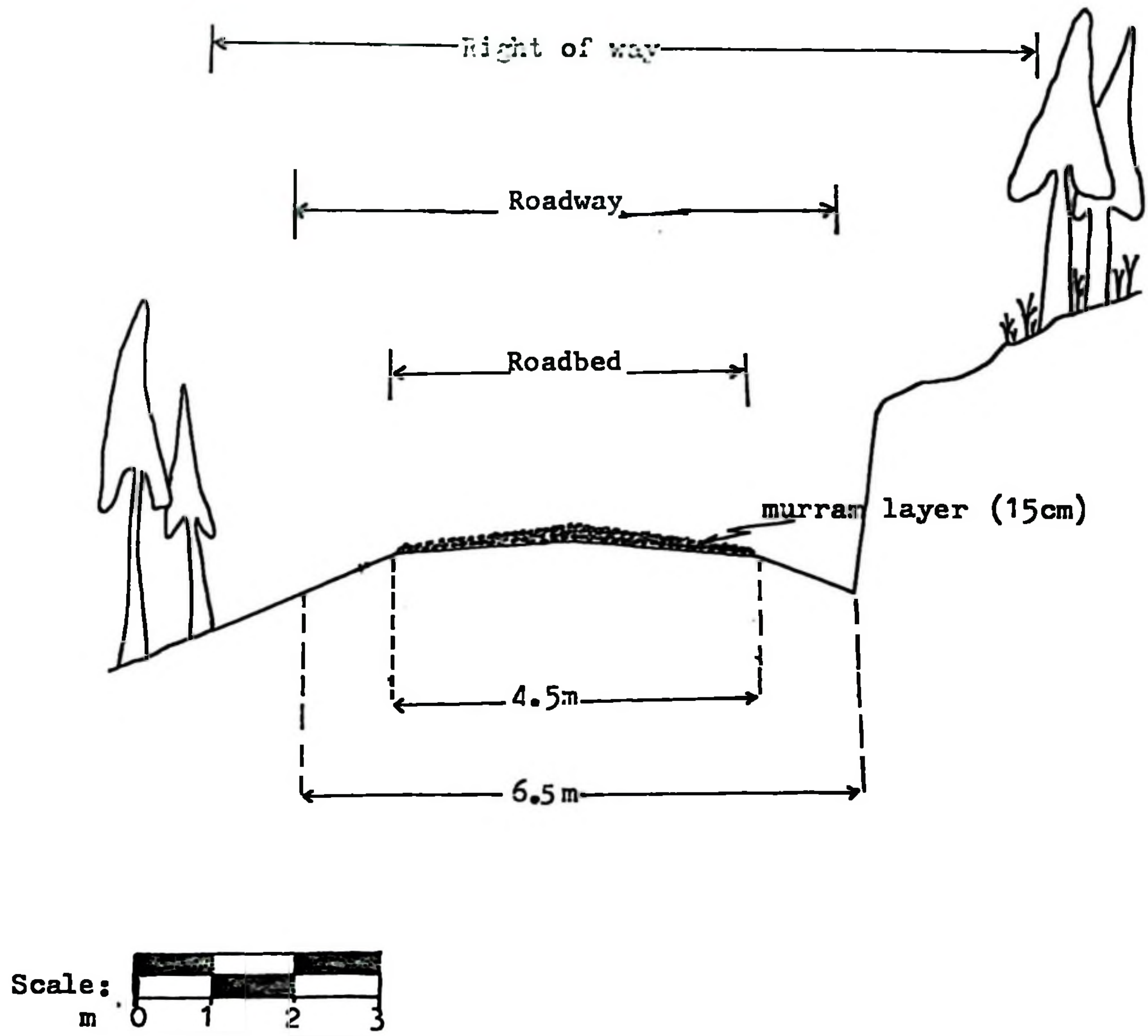


Figure 2. A cross-sectional profile of the existing Laikinoi access road.

3.2.4 Surface layer

A 15 cm layer of murram spread the whole roadway width is used as the running surface.

3.2.5 Curve widening

In order to allow for off-tracking on curves, curves have been widened using formulas developed by Smith (1973) and Sessions(1976). Figures for hauling trucks typical in North America have been used in determining the extent of curve widening. For example, curves with radii between 50-75 m require a widening of 1.5 m while those between 75-100 m require a widening of 1 m.

3.2.6 Curve radius

To be able to reach the upper part of the forest, "switchback" curves have been used on this road. The minimum curve radius on this road is 11.5 m while curves with radii more than 200 m are considered as part of tangent sections. To prevent trucks from slipping out of the road when negotiating the curves, curves have been super-elevated towards the centres.

3.2.7 Cross-drains

Ditch relief culverts or cross-drains are constructed along the road to drain off water. Two sizes of round concrete culverts used here are 45 cm and 60 cm. These are laid on a 4-5 % slope. In addition, mitre drains are dug between culvert points to drain water from the side ditches.

3.2.8 Backslopes

Backslope angles vary depending on the extent of cutting. In areas with deep cuts, the backslope angles are sharp (9:1) while in flat terrain, the backslope angles are as low as 2:1.

3.3 Description of the methodological procedure

The study was divided into two phases: the planning phase and the construction phase. The planning phase included the following operations which were done before the construction phase started:

- Reconnaissance survey of the area to be served by the new road
- Alignment of the roadline and detailed study of the longitudinal and cross-sectional profiles to obtain data for grade planning and earthwork design
- Detailed planning of road gradient, design of drainage structures and determination of curve widths and backslopes
- Calculation of earthwork volumes and development of methodology to allocate earthwork volume from cut to fill areas in order to minimize earthwork costs.

During the construction phase, activities were systematically time studied either on shift-level basis or in detail. Time study and cost data obtained were then analysed to develop a methodology to improve future planning and control of forest roads.

3.3.1 Reconnaissancesurvey of the area

The area to be served by the new road was surveyed to become familiar with the topography and ground condition. Reconnaissancesurvey helped also in locating the general route, identifying approximate location of curves and road terminus. As stated earlier, the starting point of this new road is the terminus of the existing Laikinoi access road at the Rest House (Fig. 1).

3.3.2 Road alignment

The procedure used in aligning the road is essentially that explained by Moffit and Bouchard (1975). A three-man crew consisting of an instrument man (the leader), a flagman and a helper was used for aligning the road.

On tangent sections, stations were established at intervals of 20 m while on curves, at intervals of 10 m. At each station both gradient in per cent and azimuth in degrees were measured using a Suunto clinometer and a hand compass respectively. Centreline stakes were pegged on all stations and reference pegs set on the sides. Reference stakes were to re-locate the road centreline after the centreline stakes had been removed during the construction phase.

A levelling staff plus a Suunto clinometer were used to find the ground elevation differences at 2, 4 and 6 m left and right of each station. This was necessary in order to obtain data for cross-sectional road profiles. An example of a road alignment data sheet used in the field is shown in Appendix 1.

Simple offset and tangent offset methods are the two common methods used in laying out the horizontal curves (Dykstra 1983b). The former, because of its simplicity and direct applicability in the field without preliminary design, was the one used in this study.

3.3.3 Road gradient

Since earthwork costs are inversely proportional to road grade, and road maintenance and hauling costs are directly proportional to road grade, to decide on the optimal grade which minimizes the sum of earthwork, hauling and maintenance costs, various road grades must be evaluated. In this study therefore, three road grade alternatives with maximum gradients of 6 %, 8 % and 10 % were evaluated.

Earthwork and hauling costs for each of the above road grades were determined. Procedures involved in determining the earthwork costs and hauling costs are covered later in this chapter. Within the three road grade alternatives, the one which minimizes the total construction and hauling costs was chosen for construction.

3.3.4 Drainage structures

Since no major streams or rivers cross the new road, only cross-drain culverts are required to drain water from the side ditches. To determine the size of culverts required, Talbot's formula was used in this study. This formula takes into consideration the rainfall intensity, surface run-off area and the average slope of the drainage basin, soil type and ground cover (Simmons 1979). Because of its simplicity and the ease of measuring the required variables, this formula is the most likely to be used by the forest project personnel who might view more complicated methodologies with apprehension.

For Talbot's formula to be used with the international system of units, the formula is written as follows:

$$A = \frac{C(2.47M)^{3/4}}{10.76}$$

where: A = cross-sectional area of the waterway, in m^2

C = constant factor based on soil absorptive capacity, slope and ground cover (varies from 0.2 to 1.0)

M = the area of the drainage basin, in ha

2.47 = number of acres in one hectare

10.76 = number of ft^2 in 1 m^2

The above formula is based on rainfall intensity of 100 mm/h. In a specific application, the formula has to be modified according to the actual maximum rainfall intensity expected in the study area.

Records from Olmotonyi Metereological Station show that for the past 20 years, the highest rainfall experienced in a single day was 1577 mm (Metereological Dept. 1966). Since this rainfall was measured during a period of 15 hours, rainfall intensity per hour was 105.13 mm. Because of the elevation difference of about 500 m between Olmotonyi station and the Laikinoi study area, this intensity would be expected to increase by about 10 %. Thus, the maximum rainfall intensity for this study area is taken to be 115 mm/h.

Altogether eight drainage basins were demarcated and their areas determined (Fig. 3). To determine the C factors to use with Talbot's formula, field data were collected for each drainage basin. Based on these data and making reference to Talbot's Table (Appendix 2), two C factors were chosen for computing culvert sizes:

C = 0.8 for drainage basins with steep slopes, dense canopy and less ground cover

C = 0.6 for gentle drainage basins with dense vegetation ground cover.

Figure 3 summarizes the drainage area, the C factor and the culvert size required for each drainage basin. To illustrate the determination of culvert sizes, one drainage basin (No. 2) is taken as an example.

The cross-sectional area of the waterway required for this drainage basin is:

$$A = \frac{0.8\{2.47(1.09)\}^{3/4}}{10.76}$$

$$= 0.16 \text{ m}^2$$

Knowing the cross-sectional area of the waterway, the diameter of the culvert to be used is computed as:

$$D = \sqrt{4A/\pi} \quad = \sqrt{4(0.16)/\pi} \quad = 0.45 \text{ m}$$

where: D = diameter of the culvert, in m

A = cross-sectional area of the waterway, in m^2

π = the ratio of the perimeter of a circle to its diameter = 3.14159 .....

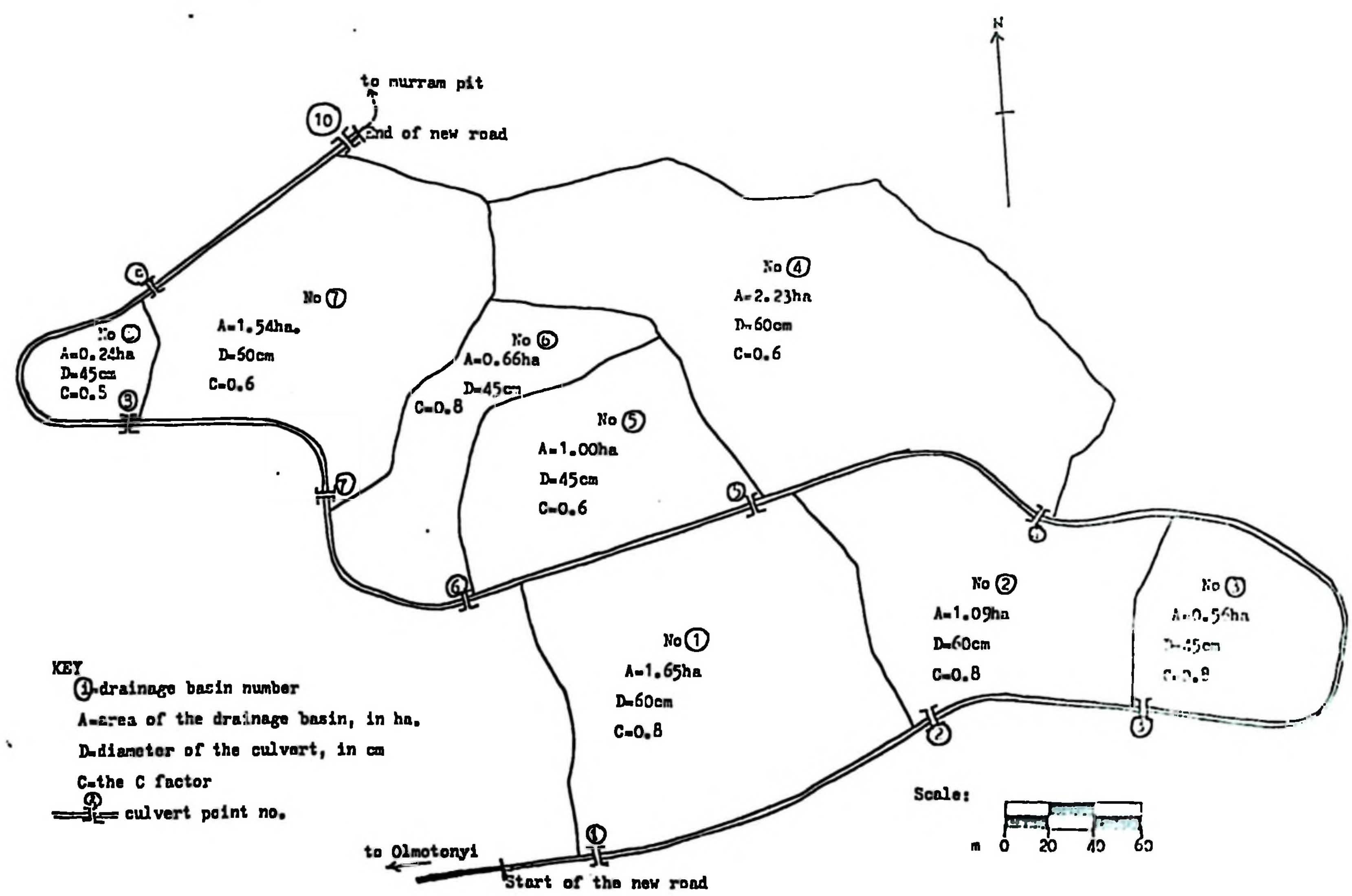


Figure 3. The drainage basins and the positions of the culvert points on the new road.

Since the above formula is based on 100 mm/h rainfall intensity and the maximum rainfall intensity expected for this area is 115 mm/h, the above culvert size is multiplied by the ratio (115/100) = 1.15; i.e., 1.15 x 0.45 = 0.52 m or 52 cm.

Only two culvert moulds were available and hence two sizes of culverts were made; 45 cm and 60 cm diameter. Where a drainage basin required a culvert size of greater than 45 cm, i.e. No. 2, a 60 cm culvert size was used and for drainage basins which required less than 45 cm, 45 cm culvert size culverts were used. No drainage basin required culvert sizes greater than 60 cm as Figure 3 indicates.

Culverts were locally moulded into 1 m sections by mixing murram and cement in the ratio of 3:1 respectively. For a 45 cm culvert, about 0.20 m^3 of murram was used while a 60 cm culvert required about 0.30 m^3 of murram. Samples of murram analysed by both sieving and hydrometer methods (Day 1965) revealed the following particle size distribution: 57 % gravel, 34 % sand, 5 % silt and 4 % clay.

3.3.5 Backslope angle

To determine the backslope angle which would minimize landslides, angle measurements were made on 48 points along the existing Laikinoi access road and the extent of landslides determined. Backslope angles were found to vary from 2:1 to 9:1 and even on steep backslopes, landslides were very minimal; i.e., less than 10 %. Thus, since the area to be served by this new road has the same soil type and experiences the same amount of rainfall, it was taken for granted that the backslopes of the two roads would behave more or less the same. Backslope angles up to 10:1 have therefore been used on this new road.

3.3.6 Curve widening

As stated in Section 3.2.5, curves on the existing Laikinoi access road have been widened based on long trucks used in North America. Since trucks to be used on this new road are not that long and no figures are given for curves with radii less than 50 m, there is a need to determine the correct curve widths which would minimize costs at the same time permit smooth passage of vehicles.

The present vehicles and the most likely to continue hauling logs from the SUA Training Forest are: 7 t Isuzu trucks (7.6 m), SISU trucks (9.2 m) and tractor-trailer combination (9.8 m). To determine the extent of curve widening required by these vehicles, the following formula developed by Smith (1973) and Sessions (1976) is used.

$$CW = R - \sqrt{R^2 - \sum_{i=1}^{n} L_i^2}$$

where: CW = required curve widening in m, for a road with a running surface width of 3 m

R = the curve radius, in m

Li = wheelbase, in m, of a vehicle or combination component i (i = 1, 2, ..., n)

n = 1 for a vehicle without a trailer; 2 for a tractor-trailer combination.

Using the above formula, the curve widening widths required on different curves for the three hauling vehicles are summarized in Table 1.

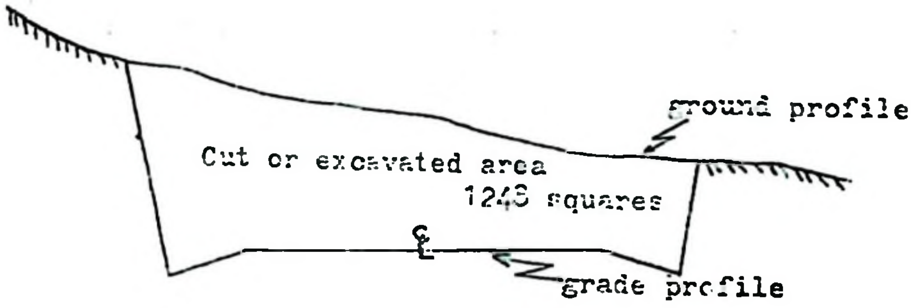
Table 1. The curve widening requirements (m) on different curve radii for the three hauling means.

Curve radius (m) \ Type of vehicle	Tractor-trailer combination	Isuzu truck	SISU truck
20.0	1.1	1.5	2.2
25.0	0.9	1.2	1.8
33.3	0.7	0.9	1.3
40.0	0.6	0.7	1.1
50.0	0.5	0.6	0.9
66.6	0.3	0.4	0.6
100.0	0.2	0.3	0.4
150.0	0.1	0.2	0.3

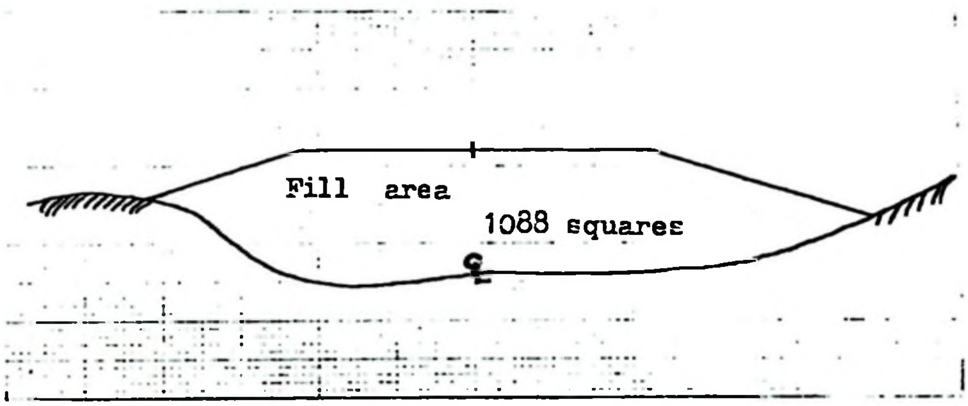
Since the SISU truck requires more curve widening than other vehicles, its column is the one selected for determining the extent of curve widening widths in this new road.

3.4 Earthwork computations

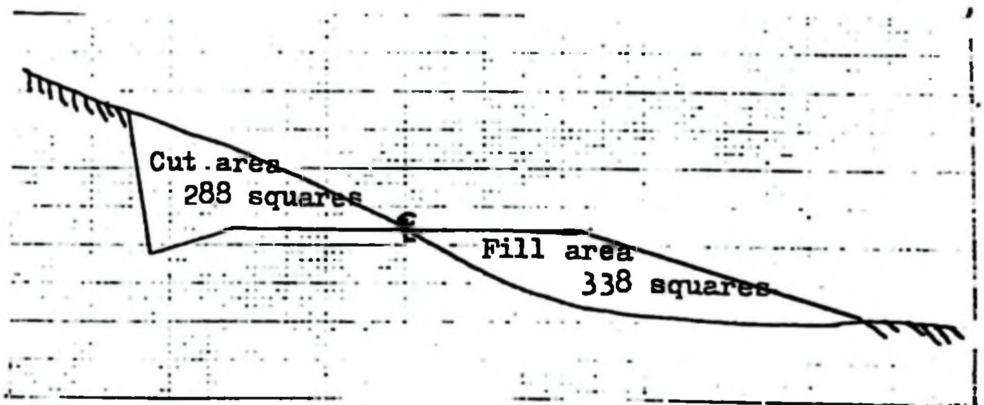
To be able to estimate the volume of soil to be excavated and filled, the end area of each station was determined first. The process involved drawing to scale the cross-sectional ground profile of each station and superimposing the cross-sectional road profile at the correct position (Fig. 4). The depth of cut or height of fill required at the centreline in order to place the road surface on the grade line was determined from the longitudinal road profiles. Figure



(a) Cut section



(b) Fill section



(c) Cut-fill section

Figure 4. Three types of cross sections encountered in this study.

4 shows the three types of cross-sectional end areas encountered in this study. It should be noted that type (c), the cut-fill section, need not have the ground line and road surface intersecting exactly at centreline as shown. A modest cut or fill at centreline would still produce a cross-section with part of the area to be excavated and part to be filled.

Normally a scale factor of 1:100 (for both vertical and horizontal axis) is used when plotting and determining the cross-sectional end areas. To determine the end area of each cross-section, the number of squares in a cross-section are counted and multiplied by a scale factor. For example, in Figure 4(a), all of the end area is to be cut away. This end area is represented by a total of 1248 small squares. Since each small square represents an area of $(0.1\ m)^2 = 0.01\ m^2$, therefore the estimated end area to be excavated is equal to $(0.01\ m^2/\text{square})(1248\ \text{squares}) = 12.48\ m^2$. The end areas for Figure 4(b) and 4(c) are computed in the similar manner.

Knowing station end areas, earthwork volumes were computed by multiplying the average end areas of adjacent stations by the distance between them. This method, called the average end area method, is equivalent to Smalian's formula for computing the wood volume of logs and is of satisfactory accuracy for most forest roads. In cases where the terrain is very irregular so that cross-sections may change from a cut section at one station to a fill section at the next station, the average end area method may be in error by as much as 16 % (Lecklinder and Lund 1971). This method was however used in lieu of a more exact method such as the prismoidal or trapezoidal methods (Moffit and Bouchard 1975) because the average end area method is easier to apply and the most likely to be preferred and used by forest project personnel in Tanzania.

Mathematically, the average end area method may be written as follows:

$$V_{i,\,i+1} = \tfrac{1}{2}(A_i + A_{i+1})L_{i,\,i+1} \qquad (1)$$

where: $V_{i,\,i+1}$ = volume of soil to be cut or filled between stations i and i+1, in m^3

A_i and A_{i+1} = end areas at station i and the adjacent station, i+1, in m^2

$L_{i,\,i+1}$ = the distance between stations i and i+1, in m.

3.4.1 Net earthwork volumes

In any particular road segment, there will either be an exact balance between the cut and fill volumes, or there will be a net cut volume (more soil will be excavated than required as fill material in the segment) or a net fill volume (more soil required as fill material than is available from excavations within the segment).

As an example, take two stations 10 m apart with cut-fill sections. Suppose the end areas for these two sections are as follows:

Station	Cut end area (m^2)	Fill end area (m^2)
720	1.53	1.26
730	2.15	0.92

Using equation (1), the total cut volume between stations 720 and 730 is equal to $0.5(1.53 + 2.15)(10) = 18.4\ m^3$. Similarly, the total fill volume between these stations is $0.5(1.26 + 0.92)(10) = 10.9\ m^3$.

When 1 m^3 of soil is excavated and compacted into a fill, it will occupy a volume less than 1 m^3; this is because some of the soil is lost when it is moved from the excavation area to the embankment and also air spaces are reduced when the soil is compacted into the fill to create a stable mass (Lecklinder and Lund 1971, Moffit and Bouchard 1975). This means that whereas the calculated fill requirement is 10.9 m^3, much more volume than this will have to be excavated and placed in the fill in order to compensate for losses and compaction. The amount of this additional volume requirement will vary according to soil type, moisture content and local construction practices (Moffit and Bouchard 1975). For the SUA Training Forest, a "compaction factor" of 0.8 has been used and adopted in this study. This means therefore, for the road segment between stations 720 and 730, the actual volume required to occupy the fill area is estimated to be $10.9/0.8 = 13.6\ m^3$.

In order to minimize construction costs, as far as possible the material excavated between stations 720 and 730 should be used to satisfy fill requirements within that segment of the road. For example, in this segment where the excavated soil material is 18.4 m^3 and the amount required as fill material is only 13.6 m^3, there will be an excess of $(18.4 - 13.6)\ m^3 = 4.8\ m^3$. This being a "net cut" segment, the excess cut material is either hauled to the nearby "net fill" segments or disposed of as sidecast material.

3.4.2 Identification of road segments

In the example of Section 3.4.1, the length of road between two adjacent stations (10 m) was used as a road segment. In general, it will prove unwieldy to use such short segments because the number of computations that must be done increases exponentially as the average segment length declines. This implies the necessity, particularly for long road projects, to increase average segment lengths in order to reduce the total number of computations required to a level that can be accommodated by forest project personnel. However, increasing the segment length will reduce the precision of the estimate of total earthwork. Thus, a balance must be struck which will provide acceptable precision and yet avoid an unnecessarily large number of computations.

In an effort to identify an average segment length which would provide this balance, three segment lengths were considered in this study.

(a) Road segments established with a target length of 30 m. Since no segment was permitted to exceed 30 m in length, such road segments are referred to as "30 m segments" even though they may on occasion be less than 30 m in length.

(b) Road segments established with a target length of 60 m. Again segment lengths may on occasion be less than 60 m in length but cannot exceed this length. Segments identified in this way are referred to as "60 m segments".

(c) Road segments established so that the end points of segments occur at points along the roadline where the original ground profile intersects the gradeline (i.e. the cut or fill at centreline is zero). This is a traditional method of identifying road segments and was used as a "control". Segments identified in this manner are referred as "terrain segments".

Figure 5 illustrates the identification of segments along a hypothetical roadline for the three alternatives. The roadline illustrated is 270 m in length, so that exactly 9 segments, each 30 m in length are established for the "30 m segment" alternative. For "60 m segments", 4 full-length segments are established but the fifth segment is only 30 m in length. Terrain segments vary in length depending upon the frequency of intersection between the ground profile and the gradeline.

Since three different road gradelines (6 %, 8 % and 10 %) are considered in this study, road segments are identified for each gradeline. Once this has been done, each segment is characterised as to whether it is a "net cut" or "net fill" segment by using the methodology of Section 3.4.1. For the present study, this means that nine separate sets of data on earthwork volumes were calculated for the three gradelines. Having done this, it was necessary to address the question of how to <u>optimally</u> allocate earthwork volumes from net cut segments to net fill segments in order to minimize earthwork costs for each of the nine combinations (three road segments and three road gradeline alternatives).

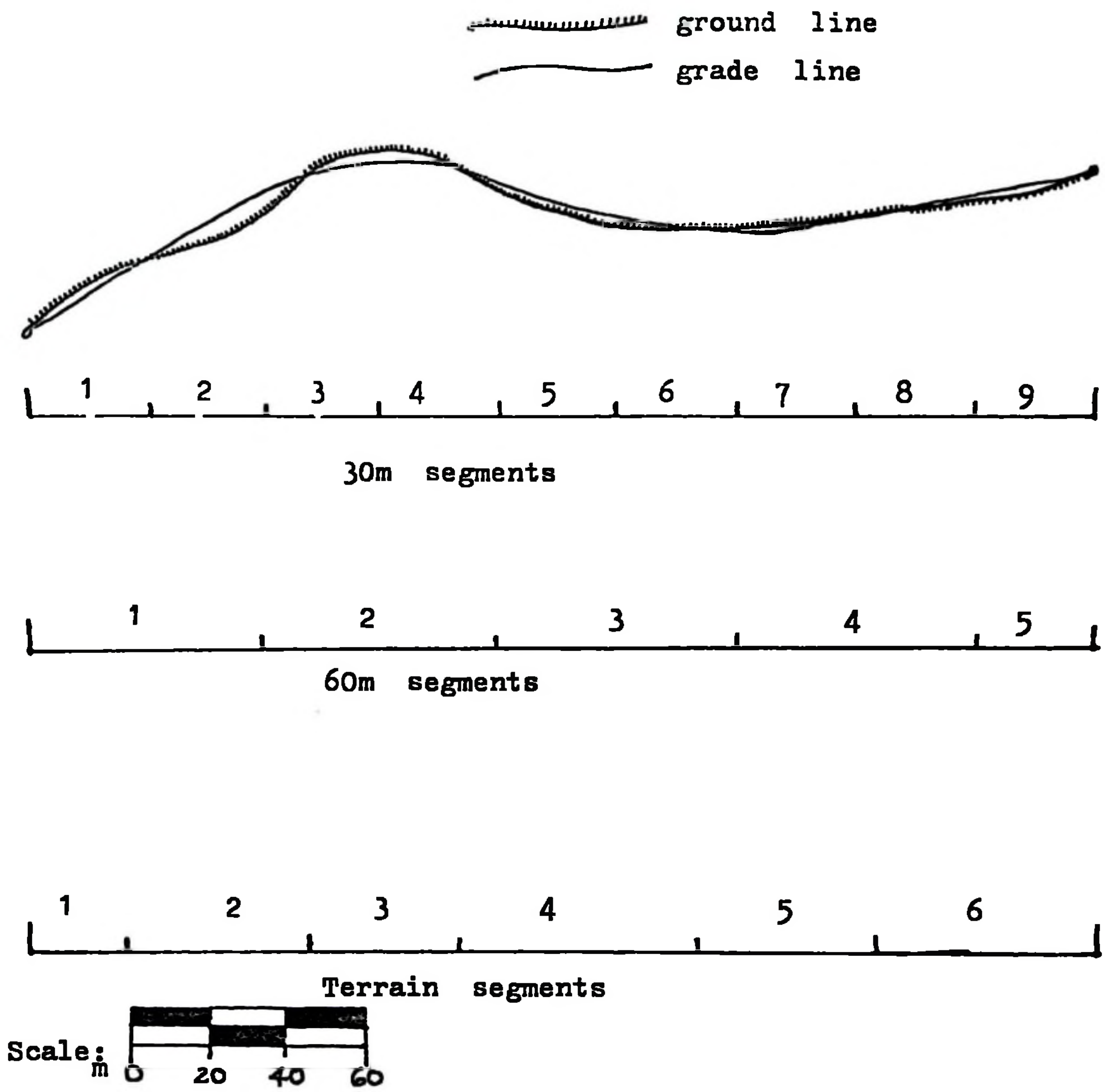


Figure 5. A hypothetical road profile illustrating the three methods of dividing the road into segments for the allocation of earthwork volumes.

3.4.3 Optimal allocation of earthwork volumes

Traditional methods used in the allocation of earthwork volumes from net cut segments to net fill segments rely heavily upon the individual skill of the engineer to minimize construction costs. Probably the most widely used of these methods is the "mass haul technique" (Lecklinder and Lund 1971, Burke 1974). This method permits the engineer to allocate earthwork volumes from net cut segments to net fill segments by determining what are called "free haul" and "overhaul" distances between segments. The engineer may then visualize, by means of a mass diagram, the effect of changes in road gradient on earthwork requirements and, indirectly, on construction costs.

The above mass haul method is however very labour intensive, requiring considerable effort to evaluate even a small number of alternative designs. In addition, its use does not necessarily produce the minimum cost design, as it depends upon qualitative interpretations by the design engineer and the degree to which the engineer is willing to test many alternative designs. Thus, since the method relies heavily on the skill and interpretive ability of the engineer, and forest projects in Tanzania do not generally have access to engineers with the skill and experience necessary to use this technique effectively, alternative methods for allocating earthwork were considered.

In recent years several authors have proposed the application of mathematical programming methods to the problem of minimizing earthwork allocation costs. Methods proposed include the transportation method of linear programming (Boughton 1967) and integer programming (Trypia 1979). The more flexible of the two methods is that of integer programming. However, there are many computational difficulties associated with its application, particularly when large-scale computer systems are not available for the calculations. The transportation method is attractive because it can be applied easily using only micro-computers or even hand calculations. For this reason it was selected as the earthwork allocation method to be used in this study.

3.4.3.1 Earthwork allocation by linear programming

Two separate solution procedures were evaluated; a micro-computer based linear programming procedure and a transportation tableau procedure suitable for hand calculations.

To illustrate the linear programming procedure, consider the data in Table 2. These data have been developed by the methods of Sections 3.4.1 and 3.4.2 for the 8 % maximum gradient along the proposed road, using 30 m segments. Part A of Table 2 summarizes data for the net cut segments, which are called "supply areas" in the terminology of transportation theory. For each supply area, the segment definition is shown (supply area 01 runs between station 60 and station 90; it thus corresponds to road segment 3, since each segment is 30 m in length). The middle of supply area 01 occurs at station 75 {(60 + 90)/2 = 75}, regardless of whether station 75 has been pegged on the ground. This mid-segment location will be used to determine the

Table 2. Net supply and demand volumes (m^3) when the road segments are 30 m for the designed 8 % road grade.

Part A. Net supply volume (m^3).

Supply area	Peg No.	Mid-segment station	Net supply (m^3)
01	60- 90	75	146.5
02	90- 120	105	216.0
03	120- 150	135	217.4
04	150- 180	165	223.7
05	180- 210	195	152.4
06	210- 240	225	174.8
07	240- 270	255	203.5
08	270- 300	285	132.0
09	390- 420	405	105.9
10	420- 450	435	104.9
11	450- 480	465	98.1
12	480- 510	495	80.1
13	510- 540	525	132.0
14	600- 630	615	64.6
15	630- 660	645	90.8
16	660- 690	675	90.5
17	720- 750	735	16.9
18	780- 810	795	9.2
19	840- 870	855	26.4
20	870- 900	885	97.6
21	930- 960	945	45.4
22	960- 990	975	70.7
23	990-1020	1005	57.9
24	1110-1140	1125	426.7
25	1140-1170	1155	143.0
26	1170-1200	1185	23.3
27	1260-1290	1275	64.5
28	1290-1300	1295	6.6
Total net supply volume (m^3)			3221.1

Part B. Net demand volume (m^3).

Demand area	Peg No	Mid-segment station	Net demand (m^3)
01	0- 30	15	18.6
02	30- 60	45	26.9
03	300- 330	315	74.5
04	330- 360	345	111.5
05	360- 390	375	9.9
06	540- 570	555	3.7
07	570- 600	585	70.9
08	690- 720	705	13.0
09	750- 780	765	13.9
10	810- 840	825	21.6
11	900- 930	915	44.5
12	1020-1050	1035	40.4
13	1050-1080	1065	198.8
14	1080-1110	1095	41.0
15	1200-1230	1215	5.4
16	1230-1260	1245	20.7
Total net demand volume (m^3)			715.3

average haul distance from supply area 01 to any of the net fill segments. The amount of the net cut volume in each supply area is shown in column 4 of part A; this is the excess quantity of soil available to be transported to net fill segments.

Part B of Table 2 shows data for net fill segments, called "demand areas" in the context of transportation theory. Demand area 01 runs between station 0 and station 30, thus corresponding to road segment 1, with a mid-segment location at station 15. The net demand for additional fill material in this segment is 18.6 m^3, which has already been adjusted for anticipated transport losses and compaction as described in Section 3.4.1.

Note that the total supply volume from part A of Table 2 is greater than the total demand volume from part B. This means that more fill material is available than is required over the entire road project. The difference between the two totals, 2505.8 m^3, is the estimated excess volume that would be sidecast if the 8 % maximum grade was adopted. Another design alternative might produce a total demand volume greater than the total supply volume. In this case, the additional fill material required would have to be excavated from "borrow pits" located along the route.

To solve the allocation problem, the data of Table 2 must be transferred into a linear programming format. The problem may then be solved numerically using a number of commercially available computer programs written for this purpose. For the present study, a locally written BASIC language program called "BigLP" was used with a Hewlett-Packard Model 85 (HP 85) micro-computer. This program was used rather than the linear programming package provided by Hewlett-Packard because the latter was unable to accommodate the relatively large problem structure of the allocation problem.

The general mathematical formulation of the earthwork allocation problem may be stated as follows:

$$\text{MINIMIZE } Z = \sum_{i=1}^{M} \sum_{j=1}^{N} Cijxij \qquad (2a)$$

subject to the constraints:

$$\sum_{j=1}^{N} xij \leq Si \qquad (i = 1, 2, \ldots, M)$$

$$\sum_{i=1}^{M} xij \geq Dj \qquad (j = 1, 2, \ldots, N)$$

$$xij \geq 0 \qquad (i = 1, 2, \ldots, M; \; j = 1, 2, \ldots, N)$$

where: Z = the value to be minimized (total construction cost associated with the allocation of soil from supply areas to demand areas)

i = an index of supply areas (i = 01, 02, ..., M = 28)

j = an index of demand areas (j = 01, 02, ..., N = 16)

Cij = the cost of excavating 1 m^3 of soil from supply area i, transporting it to demand area j, and compacting it into the fill along that road segment

xij = the total volume of soil, in m^3, to be allocated from supply area i to demand area j

Si = the net supply of soil available from supply area i

Dj = the net demand for soil in demand area j.

The variables x_{ij} are the unknown quantities in this formulation; the solution to the problem will provide a set of values for the x_{ij} such that the value of Z is minimized. The x_{ij} are thus referred to as the "decision variables".

Although the formulation specifies that the total cost of allocating soil from supply areas to demand areas is to be minimized, values for the C_{ij} were not available prior to the study; indeed, one of the objectives of the study is to provide such values from an analysis of cost data obtained during the construction of the road. Since the allocation problem had to be solved before the road could be constructed, a surrogate for the cost coefficients C_{ij} had to be selected. For this purpose the distance, in m, between the mid-station of supply area i and mid-station of demand area j was used. Thus the objective function minimizes the sum of the products of distance multiplied by volume. This sum of products can be shown to be exactly equivalent to the allocation cost if it is assumed that the unit allocation costs (C_{ij}) are directly proportional to the allocation distance. Since this proportionality is virtually always assumed in road construction cost analyses (Lecklinder and Lund 1971), the use of distance as a surrogate for unit cost will provide exactly the same results as if the unit costs themselves were used. For the purpose of this analysis, then, the C_{ij} are redefined as follows:

C_{ij} = the distance, in m, from the mid-station of supply area i to the mid-station of demand area j.

For the data of Table 2, the problem represented by the linear programming formulation requires a total of 28 x 16 = 448 decision variables and 28 + 16 = 44 constraints. A linear programming problem of this size is far too large to be solved on the HP 85 micro-computer. However, it is possible to reduce the number of decision variables significantly. Since the objective of the linear programming problem is to minimize the sum of the products of allocation distance multiplied by the volume allocated, the linear program will tend to allocate volumes to demand areas from the supply areas located closest to those demand areas. Thus, it is not necessary to consider the more distant allocations, even though they are technically possible. As an example, it is not necessary to consider the allocation of soil from supply area 01 to demand area 16, because the volume required in demand area 16 can be more economically supplied from less distant supply areas. By this reasoning it is possible to consider only a relatively small number of possible allocations (xij), so long as care is taken to ensure that this set of possible allocations includes the optimal allocations.

For this study, the number of possible allocations considered was reduced to a manageable level by identifying, for each demand area, at least the two closest supply areas. If the two closest supply areas were not sufficient to satisfy the demand, then more distant supply areas were added as potential sources. A summary of the demand areas and their potential supply areas (in brackets) as identified for this problem is shown below:

D01 (S01, S02, S03), D02 (S01, S02, S03, S04), D03 (S05, S06,

S07, S08), D04 (S08, S09), D05 (S09, S10, S11), D06 (S12, S13, S14), D07 (S13, S14, S15), D08 (S15, S16, S17), D09 (S17, S18), D10 (S18, S19, S20), D11 (S19, S20, S21, S22), D12 (S22, S23), D13 (S23, S24), D14 (S24, S25), D15 (S25, S26, S27), D16 (S26, S27, S28).

This methodology reduced the number of decision variables to be considered from 448 to 46. Thus, the final problem has 46 decision variables and 44 constraints, a problem size which is within the capabilities of the HP 85 micro-computer.

Table 3 summarizes the linear programming problem for the 8 % maximum grade with 30 m road segments. Columns in the table are identified with decision variables (xij) and rows represent the objective function coefficients and the constraint equations. A non-blank entry at a row and column intersection is read as the product of that numerical entry multiplied by the decision variable associated with the column. Thus, each row is read as a sum of products. The objective function row, for example is read as 60 X01,01 + 90 X02,01 + 120 X03,01 + ... + 30 X27,16 + 50 X28,16. Blank entries are interpreted as zero coefficients. Thus, row S01, representing the possible allocation of earthwork volume from supply area 01 to all demand areas, is read as 1 X01,01 + 1 X01,02 $\leq$ 146.5. Similarly, row D01, representing the possible allocations of earthwork volume to demand area 01 from all supply areas is read as 1 X01,01 + 1 X02,01 + 1 X03,01 = 18.6. Notice that whereas the general formulation of demand constraints indicates that the constraints are of the type ($\geq$), Table 3 shows

Table 3. Linear programming tableau matrix.

Row Name	X0101	X0201	X0301	X0102	X0202	X0302	X0402	X0303	X0403	X0703	X0803	X0804	X0904	X0905	X1005	X1105	X1206	X1306	X1406	X1307	X1407	X1507	X1508	X1608	X1708	X1709	X1809	X1810	X1910	X2010	X1911	X2011	X2111	X2211	X2212	X2312	X2313	X2413	X2414	X2514	X2515	X2615	X2715	X2616	X2716	X2816	RHS
Obj. Fctn.	60	90	120	30	60	90	120	120	90	60	30	60	60	30	60	90	60	30	60	60	30	60	60	30	30	30	30	30	30	60	60	30	30	60	60	90	60	60	30	60	60	30	60	60	30	50	[illegible]
S01	1			1																																											≤ 140.5
S02		1			1																																										≤ 210.0
S03			1			1																																									≤ 217.4
S04							1																																								≤ 223.7
S05								1																																							≤ 152.4
S06									1																																						≤ 174.8
S07										1																																					≤ 201.5
S08											1	1																																			≤ 132.0
S09													1	1																																	≤ 105.9
S10															1																																≤ 154.9
S11																1																															≤ [illegible]
S12																	1																														≤ [illegible]
S13																		1		1																											≤ 132.0
S14																			1		1																										≤ 64.6
S15																						1	1																								≤ 90.8
S16																								1																							≤ 80.5
S17																									1	1																					≤ 36.9
S18																											1	1																			≤ 3.7
S19																													1		1																≤ 25.4
S20																														1		1															≤ 57.6
S21																																	1														≤ 45.4
S22																																		1	1												≤ 73.7
S23																																				1	1										≤ 57.9
S24																																						1	1								≤ [illegible]
S25																																								1	1						≤ 143.0
S26																																										1		1			≤ 23.3
S27																																											1		1		≤ 64.5
S28																																														1	≤ 6.6
D01	1	1	1																																												= 19.5
D02				1	1	1	1																																								= 25.9
D03								1	1	1	1																																				= 74.5
D04												1	1																																		= 111.5
D05														1	1	1																															= 9.9
D06																	1	1	1																												= 3.7
D07																				1	1	1																									= 73.9
D08																							1	1	1																						= 13.0
D09																										1	1																				= 11.9
D10																												1	1	1																	= 21.5
D11																															1	1	1	1													= 44.5
D12																																			1	1											= 43.5
D13																																					1	1									= [illegible]
D14																																							1	1							= 41.0
D15																																									1	1	1				= 5.4
D16																																												1	1	1	= [illegible]

them as equalities. This is because the "BigLP" program used to solve the problem numerically requires less computer memory for equality constraints than for inequality constraints of type (≥). Since total supply exceeds total demand for this road project, all of the demand constraints can be satisfied. And since the objective function is to be minimized, all of these constraints will be satisfied at the lower limit; that is, as equalities. Therefore the optimal solution will be identical whether the demand constraints are of the type (=) or of type (≥). Equalities have thus been used because they minimize the computer memory requirements and therefore reduce solution time.

3.4.3.1.1 Optimal linear programming solution

After the linear programming problem has been organized as shown in Table 3, it can be entered into the computer in the format required by the "BigLP" program. When the data have been entered and verified, the linear programming procedure is executed. Details of the solution procedure which uses the standard "simplex" method, are included in most introductory management science textbooks such as Dykstra (1984), Hillier and Lieberman (1980).

For the earthwork allocation problem with a 8 % maximum gradient and 30 m segments, the optimum value of the objective function (Z) was found to be 31 515 m^3m. Detailed output from the computer program is included in Appendix 3, while optimum volume allocations are summarized in Table 4. Since the total fill volume required in the 16 demand

Table 4. Summary of the optimal linear programming solution to the earthwork allocation problem for the proposed road with a 8 % maximum gradient and 30 m segments.

Supply areas	Demand areas D01	D02	D03	D04	D05	D06	D07	D08	D09	D10	D11	D12	D13	D14	D15	D16	Total vol. alloc. (m^3)	Total supply avail. (m^3)	Slack (m^3)
S01	60 18.6	30 26.9															45.5	146.5	101.0
S02																		218.0	218.0
S03																		217.4	217.4
S04																		223.7	223.7
S05																		152.4	152.4
S06																		174.8	174.8
S07																		203.5	203.5
S08			30 74.5	60 57.5													132.0	132.0	
S09				60 54.0	30 9.9												63.9	105.9	42.0
S10																		104.9	104.9
S11																		98.1	98.1
S12																		80.1	80.1
S13						30 3.7	60 6.3										10.0	132.0	122.0
S14							30 64.6										64.6	64.6	
S15																		90.8	90.8
S16								30 13.0									13.0	90.5	77.5
S17									30 13.9								13.9	16.9	3.0
S18										30 9.2							9.2	9.2	
S19										30 12.4							12.4	26.4	14.0
S20																		97.6	97.6
S21											30 44.5						44.5	45.4	0.9
S22																		70.7	70.7
S23												30 40.4	60 17.5				57.9	57.9	
S24													60 181.3	30 41.0			222.2	426.7	204.4
S25																		143.0	143.0
S26															30 5.4		5.4	23.3	17.9
S27																30 20.7	20.7	64.5	43.8
S28																		8.8	8.8
Total Demand (m^3)	18.6	26.9	74.5	111.5	9.9	3.7	70.9	13.0	13.9	21.6	44.5	40.4	198.8	41.0	5.4	20.7	715.3	3221.1	2505.8

Z = 31 515

areas is 715.3 m^3 (Table 2B), this implies that the optimal solution corresponds to an average earthwork transportation distance of 31 515/715.3 = 44.1 m for each m^3 of earthwork allocated from supply areas to demand areas.

Entries in the main part of Table 4 indicate the quantity of soil allocated from supply areas to demand areas in the optimal solution to the linear programming problem. For example, the solution indicates that 18.6 m^3 would be allocated to demand area 01 from supply area 01. In the context of the linear programming problem, this is indicated as an assignment of $X01,01 = 18.6$. Blanks in the main part of the Table correspond to combinations which no assignment is made.

Entries associated with demand areas in the row labelled "total demand " are the sums of allocations to each demand area column; by definition these must equal to the entries in column 4 of Table 2B if the problem is to be feasible. The entries in the "total volume allocated" column are the sums of allocations from each of the supply areas. This column's total will equal the total volume required over the entire road if the problem is feasible. The "total supply available" column entries correspond to the entries in column 4 of Table 2A. Entries in the "slack" column are the differences between the total supply available and the volume actually allocated from each of the supply areas. These values represent the excess fill material to be sidecast in each of the supply areas.

3.4.3.2 Earthwork allocation by the transportation tableau method

Although the linear programming method for solving the earthwork allocation problem is efficient, its use in general would require the availability of a computer. Linear programming problems can conceivably be solved by hand, but the problem of Section 3.4.3.1 would have required quite a considerable amount of computational time if it had been solved this way. The computational method is very tedious and, except for small problems, is prone to cumulative arithmetic errors (Hillier and Lieberman 1980). These considerations motivated a se arch for a reliable, intuitively appealing and computationally simple optimization technique. The transportation method proposed by Boughton (1967) appears to fulfill these requirements.

As a demonstration of the transportation method, consider the problem of Section 3.4.3.1; that is, the design alternative using a 8 % maximum gradient with 30 m road segments. Table 5 shows the distances (placed on top of each "cell" in block letters) between supply areas and demand areas for this problem, i.e., the distance from the mid-station of supply area 01 to the mid-station of demand area 01 is 60 m. The supply column indicates the quantities of excess soil available from each of the supply areas, and the demand row indicates the quantities of fill required in each of the demand areas. The "slack" column represents the difference between the volumes available from the supply areas and the total volumes actually allocated from those supply areas to various demand areas.

Notice that some of the distance entries in Table 5 are blank; thus

Table 5. A transportation tableau showing the initial feasible solution.

Supply areas	Demand areas D01	D02	D03	D04	D05	D06	D07	D08	D09	D10	D11	D12	D13	D14	D15	D16	Slack vol. (m^3)	Total supply (m^3)
S01	60 (18.6)	30 (26.9)	240	270	300	480											0 (101.0)	146.5
S02	90	60	210	240	270	450											0 (216.0)	216.0
S03	120	90	180	210	240	420											0 (217.4)	217.4
S04	150	120	150	180	210	390											0 (223.7)	223.7
S05	180	150	120	150	180	360											0 (152.4)	152.4
S06	210	180	90	120	150	330	360										0 (174.8)	174.8
S07	240	210	60	90	120	300	330										0 (203.5)	203.5
S08	270	240	30 (74.5)	60 (5.6)	90	270	300										0 (51.9)	132.0
S09	390	360	90	60 (105.9)	30	150	180										0	105.9
S10	420	390	120	90	60 (9.9)	120	240										0 (95.0)	104.9
S11	450	420	150	120	90	90	210										0 (98.1)	98.1
S12			180	150	120	60	90	210									0 (80.1)	80.1
S13			210	180	150	30 (3.7)	60	180	240								0 (128.3)	132.0
S14			240	210	180	60	30 (64.6)	90	150	210	300						0	64.6
S15			330	300	270	90	60 (6.3)	60	120	180	270	390	420				0 (84.5)	90.8
S16			360	330	300	120	90	30 (13.0)	90	150	240	360	390				0 (77.5)	90.5
S17			420	390	360	180	150	30	30 (13.8)	90	180	300	330	480			0 (5.1)	18.9
S18				450	420	240	210	90	30	30 (9.2)	120	240	270	300			0	9.2
S19				510	480	300	270	150	90	30 (12.4)	60	180	210	240			0 (14.0)	26.4
S20					510	330	300	180	120	60	30 (44.5)	150	180	210	330		0 (53.1)	97.6
S21					570	390	360	240	180	120	30	90	120	150	270		0 (65.4)	65.4
S22						420	390	270	150	60	60	60	90	120	240		0 (70.7)	70.7
S23						450	420	300	240	180	90	30 (40.4)	60 (17.5)	90	210		0	57.9
S24						570	540	420	360	300	210	90	60 (181.3)	30 (41.0)	90	120	0	204.4
S25							570	450	390	330	240	120	90	60	60	90	0 (143.0)	143.0
S26							600	480	420	360	270	150	120	90	30 (5.4)	60	0 (17.9)	23.3
S27								570	510	450	360	240	210	180	60	30 (20.7)	0 (43.8)	64.5
S28									530	470	380	260	230	200	80	50	0 (6.6)	6.6
Total Demand (m^3)	18.6	26.9	74.5	111.5	9.9	3.7	70.9	13.0	13.8	21.6	44.5	40.4	198.8	41.0	5.4	20.7	2505.8	3221.1

Z = 31 812

for example, the distance from supply area 01 to demand area 08 is not shown. Following the reasoning of Section 3.4.3.1.1, the possibility of supplying demand area 08 from supply area 01 has not been considered, because the fill requirement in that demand area can be satisfied more economically from other supply areas closer to it. However, a greater number of possible allocations is represented in Table 5 than was represented by the linear programming formulation in Table 4. This is because the computational burden imposed by the transportation method is relatively light, so that more alternatives can be considered. In fact, all possible alternatives (including, for example, the possible allocation of soil from supply area 01 to demand area 16) could be evaluated without significantly increasing the computational effort required. Altogether, the entries in Table 5 are equivalent to a linear programming problem with 240 decision variables and 44 constraints.

The transportation method can be visualized as a series of steps leading to the optimal solution. These steps are described in the discussion which follows.

Step 1

To initialize the problem, the first task is to make up a "transportation tableau" as illustrated in Table 5. Since the volume on the slack column is to be sidecast from the supply areas rather than allocated to demand areas, no distance is associated with the slack volumes, and the distance entries for cells in this column are all equal to zero.

It should be noted that the slack column is required in the transportation method described here because the procedure will work correctly only if the total supply is equal to the total demand. For this problem there is an excess supply of 2505.8 m^3. Thus, the slack column serves as an additional "demand area" which makes up for the difference between supply and demand. If demand had exceeded supply, then an additional row would have been required in place of the slack column. This row would represent the additional volume to be allocated to demand areas from a "borrow pit" located somewhere near the road project. Distances entered in the cell boxes associated with this row would be non-zero, since there would be an actual distance from the borrow pit to each of the demand areas.

In addition to the distances between the mid-stations of supply and demand areas, some of the cells in Table 5 contain circled entries. These entries represent the volume of soil to be allocated from supply areas to demand areas. Consider cell (08,03). The circled entry in this cell is 74.5 indicating an allocation of 74.5 m^3 of soil from supply area 08 to demand area 03. The method of establishing these entries is discussed in Step 2. With the exception of some additional entries that will be made in the cells and the eventual addition of one more row and one more column, Table 5 constitutes a typical transportation tableau for the earthwork allocation problem.

Step 2

Before commencing the optimization phase of the transportation method, an initial feasible solution to the allocation problem must be generated. By "feasible solution" is meant a solution (i.e., an assignment of volume allocations from supply areas to demand areas) for which all of the demand requirements are met without exceeding any of the supply limitations.

Several methods are commonly used for this purpose (Dykstra 1984, Hillier and Lieberman 1980, Phillips *et al.* 1976). A modified version of the so-called "northwest corner method" is used here because it is intuitively appealing, requires only simple calculations and could be executed by forest project personnel in Tanzania.

As the name implies, the northwest corner method begins by making an assignment to the northwest corner cell of the tableau; that is, cell (01,01). The entry to this cell is made as large as possible by checking the boundary conditions. The smaller of these two quantities limits the entry in cell (01,01),i.e., the entry is 18.6 m^3 since that is the volume of fill required by demand area 01. The remaining volume in supply area 01 (146.5 - 18.6 = 127.9 m^3) is temporarily entered in the slack column. Since the requirements of column 01 have been satisfied, that column is omitted from further consideration.

Having eliminated column 01, the northwest corner method moves to column 02. In order to have an initial solution which is closer to the optimal solution, a variant of the northwest corner method called column minima (Hadley 1962) is used. This method systematically eliminates one column at a time starting from the left to the right. Within a column, the method moves up and down searching for a cell or row with the minimum or shortest distance. In this case, therefore, an allocation of 26.9 m^3 is made to cell (01,02) since that is the cell within column 02 with the shortest distance (30 m). Since this volume is taken from supply area 01 (temporarily stored in the slack column) the amount available in supply area 01 now becomes; 127.9 - 26.9 = 101.0 m^3. This amount is again entered in the "slack" column and column 02 eliminated from further consideration.

In column 03, the cell with the minimum distance is (08,03). The total demand volume by demand area 03 is 74.5 m^3. From supply area 08, this amount is supplied to demand area 03, while the excess (132.0 - 74.5 = 57.5 m^3) is temporarily entered in the slack column. Again, column 03 is dropped from further consideration.

Since rows 02, 03, 04, 05, 06 and 07 have been skipped over, the full amount of soil available from these supply areas is entered into the "slack" column. At this point therefore, columns 01, 02 and 03 and the first eight rows have been eliminated.

The allocations continue in this manner, successively eliminating rows and columns until the demand requirements associated with all of the columns have been satisfied. Where a demand area requires supplies from more than one supply area (because the closest supply is not big enough), the column minima variant method moves to the next cell with minimum distance. For instance, to satisfy the requirements of column 04, Table 5 shows that an extra 5.6 m^3 of soil from supply area 08 had to be supplied to demand area 04 in addition to 105.9 m^3 of soil formerly supplied by supply area 09.

Step 3

After all the column requirements have been satisfied and the final slack quantities calculated, the demand column totals (including the slack column) should add up exactly to the total of the supply column. If not, then an error has been made in the allocations. When the totals add correctly, the allocations indicated by the completed tableau represent a feasible solution to the allocation problem.

One additional condition must be satisfied in order for the assignments represented by Table 5 to be considered an initial feasible solution. This condition is a technical consideration necessitated by the mathematical programming technique. Each of the rows and columns in the transportation tableau may be thought of as an equation. For instance, the entries in column 01 must add exactly to 18.6, and the entries in row 01 must add exactly to 146.5. If, in general, there are M rows (where M = the number of supply areas) and N columns (where N = the number of demand areas + 1 for the slack column), then there are M+N such equations. However, since these equations are all exact equalities, a solution can be determined by using only (M + N − 1) of them; i.e., the solution to any of the equations can be determined whenever the other (M + N − 1) entries in the tableau are known (Dykstra 1984;

Hillier and Lieberman 1980). For the problem considered here, M = 28 and N = 17. Therefore there must be 28 + 17 - 1 = 44 entries in the tableau. Since Table 5 has 44 entries and all of the demand requirements are met without exceeding the supply limitations, it can therefore be declared to represent an initial feasible solution to the earthwork transportation problem.

It should be observed that Table 5 provides support for the statement made earlier that the column minima variant of the northwest corner method used here will provide an initial feasible solution which is reasonably close to the optimal solution for the earthwork allocation problem. Notice in Table 5 that most of the cell entries lie near the diagonal of the table. Because of the way the cut and fill areas are defined, beginning at the start of the project, and because there are approximately the same number of cut areas as fill areas, the minimum distance from any supply area i to any demand area j lies near the diagonal. Therefore, the initialization method will normally produce a nearly optimal initial feasible solution. This conclusion does not apply to transportation problems in general; only to the earthwork transportation problem considered here.

Step 4

The objective of the earthwork transportation problem is to minimize the sum of all earthwork volumes multiplied by the distances over which the volumes must be transported from cut areas to fill areas. As observed for the linear programming formulation, this objective may be stated mathematically as follows:

$$\text{MINIMIZE } Z = \sum_{i=1}^{M} \sum_{j=1}^{N} Cijxij \qquad (2b)$$

where: Z = the value of the objective function associated with any solution, in m^3m

M = the number of rows in the tableau

N = the number of columns in the tableau, including the slack column

Cij = the "cost" (i.e., distance) associated with the transportation of earthwork from supply area i to demand area j, in m

xij = the entry in cell (i,j) representing the volume of earthwork allocated from supply area i to demand area j, in m^3.

The value of the objective function associated with the initial feasible solution can be determined by evaluating equation (2b). This initial value can later be compared with the optimal value of the objective function to ascertain the degree of improvement accomplished by the optimization algorithm. For the initial feasible solution in Table 5, the value of Z is as follows:

$$\begin{aligned} Z = {} & 60(18.6) + 30(26.9) + 0(101.0) + 0(216.0) + 0(217.4) + \\ & 0(223.7) + 0(152.4) + 0(174.8) + 0(203.5) + 30(74.5) + \\ & 0(51.9) + 60(105.9) + 60(5.6) + 0(95.0) + 60(9.9) + 0(98.1) \\ & + 0(80.1) + 30(3.7) + 0(128.3) + 30(64.6) + 60(6.3) + \\ & 0(84.5) + 30(13.0) + 0(77.5) + 30(13.9) + 0(3.0) + 30(9.2) + \\ & 30(12.4) + 0(14.0) + 30(44.5) + 0(53.1) + 0(45.6) + 0(70.7) \\ & + 30(40.4) + 60(17.5) + 60(181.3) + 0(204.4) + 30(41.0) + \\ & 0(102.0) + 30(5.4) + 0(17.9) + 30(20.7) + 0(43.8) + 0(6.6) \\ = {} & 31\ 812.0 \ m^3m \end{aligned}$$

Note that there are exactly (M + N - 1) = 44 elements in this equation. Since the total volume required in the 16 demand areas is 715.3 m^3, the average transportation distance represented by the 44 assignments in the initial solution is: 31 812.0/715.3 = 44.5 m.

Step 5 The optimization algorithm

Having found an initial feasible solution, the transportation simplex algorithm, also called the primal-dual transportation algorithm or MODI method (for MOdified DIstribution method) can be used to iterate to the optimal solution (Hadley 1962). In this context, the word "iterate" means "to execute a formalized solution procedure repeatedly" (Hillier and Lieberman 1980). Each transportation tableau obtained during the optimization step is referred to as an "iteration". The initial feasible solution (Table 5) is referred to as iteration 0.

To use the transportation simplex algorithm it is necessary to introduce two quantities Ui and Vj, where Ui is associated with tableau row i and Vj is associated with column j. In the context of mathematical programming, these are the dual variables associated with the row and column constraint equations. For any row i, Ui measures the marginal improvement in the value of the objective function that would result if one additional m^3 of soil were available from supply area i. For any column j, the Vj measures the marginal improvement in the objective function if one less m^3 of soil were required in demand area j.

The quantities Ui and Vj have the property that for any cell in which there is currently an assignment, say x_{ij}, the following identity holds:

$$C_{ij} = U_i + V_j \tag{3}$$

Since there are $(M + N - 1)$ cells with current assignments, there will be that many equations of the type (3). But there are $(M + N)$ unknowns in the equations (i.e., the U_i and V_j). Therefore, an arbitrary value can be assigned to any one of the unknowns without violating the equations. Having assigned this one arbitrary value, all of the remaining U_i and V_j values can be calculated from the $M + N - 1$ equations of the type (3), since the values of the C_{ij} are known for all feasible cells.

A convenient way of assigning the arbitrary value is to select the row that has the largest number of allocations, say row k, and set $U_k = 0$. From Table 5, the row with the largest number of allocations is row 01, with three allocations. Setting $U_{01} = 0$, equation (3) becomes:

$$C_{01,j} = 0 + V_j \quad \text{or} \quad V_j = C_{01,j}$$

From row 01, the values of j for which assignments have been made are j = 01, 02 and 17 (slack). Therefore the values of V_j are: $V_{01} = 60$, $V_{02} = 30$, $V_{17} = 0$. These values of V_j are used to calculate additional values for the U_i. For instance, consider column 17. Twenty-four cells have been assigned values in this column. Since $V_{17} = 0$, equation (3) becomes:

$$C_{i,17} = U_i + 0 \quad \text{or} \quad U_i = C_{i,17} - 0$$

Therefore, the Ui values that can be established knowing V17 are Ui for rows 01 to 08, 10 to 13, 15 to 17, 19 to 22 and 24 to 28. Since sidecast (Ci,17) distance is zero, all the values of Ui for the above rows are zero too; i.e., Ui = 0-0. Continuing in this manner, all of the Ui and Vj values are established. Table 6 shows the values of Ui and Vj for all rows and columns in the tableau.

The stopping rule

Before initiating the optimization algorithm, it is necessary to know how to recognize an optimal solution. The initial feasible solution may, in some instances, be optimal, so it should be tested for this condition before proceeding. The stopping rule for the transportation simplex algorithm is as follows:

> If, for every xij for which no cell assignment has been made (i.e., the cell is feasible but blank), the quantity (Cij - Ui - Vj) is greater than or equal to zero, then the current solution is optimal.

This rule follows directly from the theory of the simplex method of linear programming (Hadley 1962). The quantity (Cij - Ui - Vj) is the "simplex criterion" associated with variable xij (i.e., cell {i,j}). For a minimization problem if this quantity has a negative sign, then the value of the objective function could be improved (reduced) by making an assignment to variable xij, which currently has no assignment (the cell is blank). If the quantity is equal to zero, then no improvement in the objective function would be achieved

Table 6. A transportation tableau showing the initial feasible solution and values of ui, vj and Cij-ui-vj for the blank cells.

Supply areas	Demand areas																Slack vol (m^3)	Total supply (m^3)	u_i
	D01	D02	D03	D04	D05	D06	D07	D08	D09	D10	D11	D12	D13	D14	D15	D16			
S01	60 (18.6)	30 (26.9)	240 +210	270	300	480											0 (101.0)	148.5	+0
S02	90 +30	60 +30	210 +180	240	270	450											0 (216.0)	216.0	+0
S03	120	90 +60	180 +150	210 +150	240	420											0 (217.4)	217.4	+0
S04	150	120 +90	150 +120	180 +120	210	390											0 (223.7)	223.7	+0
S05	180	150 +120	120 +90	150 +90	180	360											0 (152.4)	152.4	+0
S06	210	180 +150	90 +60	120 +60	150 +120	330	360										0 (174.8)	174.8	+0
S07	240	210 +180	60 +30	90 +30	120 +90	300	330										0 (203.5)	203.5	+0
S08	270	240 +210	30 (74.5)	60 (5.6) ⊕	90 +60	270 +240	300										0 (51.9) ⊖	132.0	+0
S09	390	360	90 +60	60 (105.9) ⊖	30 ⊕ -30	150 +120	180										0	105.9	+0
S10	420	390	120 +90	90 +30	60 (9.9) ⊖	120 +90	240										0 (95.0) ⊕	104.9	+0
S11	450	420	150 +120	120 +60	90 +60	90 +60	210										0 (98.1)	98.1	+0
S12			180 +150	150 +90	120 +90	60 +30	90	210									0 (80.1)	80.1	+0
S13			210	180 +120	150 +120	30 (3.7)	60 +0	180 +150	240								0 (128.3)	132.0	+0
S14			240	210	180 +180	60 +60	30 (64.6)	90 +90	150	210	300						0	84.6	-30
S15			330	300	270 +240	90 +60	60 (6.3)	60 +30	120	180	270	390	420				0 (84.5)	90.8	+0
S16			360	330	300 +270	120 +90	90 +30	30 (13.0)	90 +60	150	240	360	390				0 (77.5)	90.5	+0
S17			420	390	360 +330	180 +150	150 +90	30 +0	30 (13.9)	90 +60	180 +150	300	330	480			0 (5.0)	18.9	+0
S18				450	420	240	210 +150	90 +60	30 +0	30 (9.2)	120 +90	240	270	300			0	9.2	+0
S19				510	480	300	270	150 +120	90 +60	30 (12.4)	60 +30	180 +150	210	240			0 (14.0)	26.4	+0
S20					510	330	300	180	120 +90	60 +30	30 (44.5)	150 +120	180 +120	210	330		0 (53.1)	97.6	+0
S21					570	390	360	240	180 +120	120 +90	30 +0	90 +60	120 +60	150	270		0 (45.4)	45.4	+0
S22						420	390	270	150 +120	60 +30	60 +30	60 +30	90 +30	120	240		0 (70.7)	70.7	+0
S23						450	420	300	240	180 +120	90 +60	30 (40.4)	60 (17.5)	90 +60	210		0	57.9	+0
S24						570	540	420	360	300	210 +180	90 +60	60 (181.3)	30 (41.0)	90 +60	120	0 (204.4)	425.7	+0
S25							570	450	390	330	240	120 +90	90 +30	60 +30	60 +30	90 +60	0 (143.0)	143.0	+0
S26							600	480	420	360	270	150	120 +60	90 +60	30 (5.4)	60 +30	0 (17.9)	23.3	+0
S27								570	510	450	360	240	210	180 +150	60 +30	30 (20.7)	0 (43.8)	84.5	+0
S28									530	470	380	260	230	200	80 +50	50 +20	0 (6.6)	6.6	+0
Total Demand (m^3)	18.6	26.9	74.5	111.5	9.9	3.7	70.9	13.0	13.9	21.6	44.5	40.4	198.8	41.0	5.4	20.7	2505.8	3221.1	
V_j	+60	+30	+30	+60	+60	+30	+60	+30	+30	+30	+30	+30	+60	+30	+30	+30	+0		

Z = 31 812

by making an assignment to xij (although the objective function would not increase, either). However, if the quantity is greater than zero, then the value of the objective function would be increased if an assignment were made to xij. Clearly this would be undesirable in the context of a minimization problem. Therefore, when all blank cells have zero or positive values for the quantity (Cij - Ui - Vj), a solution providing the minimum value of the objective function has been found.

Step 5.1

Enter the value of (Cij - Ui - Vj) in each blank cell, (i.e., without a circled assignment) representing a feasible allocation of soil from supply area i to demand area j, as shown in Table 6. Apply the stopping rule. If it is satisfied, STOP; the optimal solution has been found. Otherwise, go to Step 5.2. Note that for Table 6, the stopping rule is not satisfied; the initial feasible solution is therefore not optimal.

Step 5.2

Choose the blank cell for which the value of (Cij - Ui - Vj) is most negative. An assignment will be made in this cell. In the event of a tie, make an arbitrary choice. For example in Table 6, the cell chosen is (09,05), for which the value (C09,05 - U09 - V05 or 30 - 0 - 60) = -30 is more negative than any other (Cij - Ui - Vj) value associated with the blank cells. Go to Step 5.3.

Step 5.3

Increase the value of the entering variable xij (which in the example is X09,05) as much as possible from zero. To determine the maximum quantity by which xij can be increased while ensuring that the fill requirements and supply limitations are observed, first place a "+" in cell (i,j). Move horizontally in the same row until a cell with a circled value (i.e., an assignment) is encountered. Call this cell (i,q). If, in column q, there is at least one cell in another row having a circled value, then place a "-" in cell (i,q). If not, continue moving horizontally until such a cell with a circled entry is encountered. If the initial feasible solution has been established correctly, with (M + N - 1) assignments, then there will always be at least one cell (i,q) in row i meeting these requirements. In the example in Table 6, the cell satisfying these conditions is cell (09,04).

At cell (i,q), make a right-angle turn and move vertically in column q until a cell with a circled value is encountered. Call this cell (p,q). If, in row p, there is at least one cell in another column having a circled value, then place a "+" in cell (p,q). If not, continue moving vertically until such a cell is encountered. In the example in Table 6, the cell satisfying these requirements is cell (08,04).

Continue in this manner until a complete path has been traced, ending back at the beginning cell (i,j). Notice that the path starts in a "blank" cell (without a circled value) but then proceeds in subsequent steps to consider only "non-blank" cells (those with circled values). For this reason it is sometimes called the "stepping stone" procedure.

The "+" and "-" signs alternate, and indicate that the current assignment is to be either increased (+) or decreased (-). To illustrate, consider the path in Table 6. The "+" in cell (09,05) indicates that this value is to be increased from zero. The "-" in cell (09,04) indicates that the present assignment of 105.9 is to be reduced. This is necessary if the row constraint is to be observed. That is, if soil is to be moved from supply area 09 into demand area 05, then some of the soil presently allocated from supply area 09 into demand area 04 must be diverted to demand area 05. Similar interpretations hold for cells (08,04), (08,17), (10,17) and (10,05), the other components of the path.

Step 5.4

Having ascertained the path of cells whose values are to be adjusted, identify the cell in this path, from among those in which "-" signs have been entered, having the smallest present assignment. In Table 6, this is cell (10,05), with a present assignment of 9.9. Make an assignment equal to this value in cell (i,j). Then adjust the assignments in the remaining cells in the path by an equal amount, according to the signs that have been entered in the cells. After following this procedure, the six cells in the path in Table 6 have the following assignments:

(09,05) : 9.9

(09,04) : 96.0

(08,04) : 15.5

(08,17) : 42.0

(10,17) : 104.9

(10,05) : 0.0 (blank)

Enter the new values in a new tableau, together with all of the assignments that remain unchanged. Circle all of the non-blank assignments. This is done in Table 7.

After completing Step 5.4, return to Step 5.1 and execute this step for the next tableau. If the stopping rule is satisfied, then the optimal solution has been found. If not, continue executing Steps 5.1-5.4 until the optimal solution is found. For the example problem, when the stopping rule is applied to the solution in Table 7, it is found that all of the (Cij - Ui - Vj) values in the blank cells are greater than or equal to zero. Thus, Table 7 represents the optimal solution, for which the assignments of soil from supply areas to demand areas are indicated by the circled quantitites in the cells. The fact that only one iteration was required to find the optimal solution verifies the claim that the initial feasible solution was reasonably close to the optimal solution.

The optimal value of the objective function can be determined by executing equation (2b) for the optimal earthwork assignments. This

Table 7. A transportation tableau showing the optimal solution after the first iteration.

Supply areas	Demand areas																Slack vol. (m^3)	Total supply (m^3)	u_i
	D01	D02	D03	D04	D05	D06	D07	D08	D09	D10	D11	D12	D13	D14	D15	D16			
S01	60 (18.6)	30 (26.9)	240 +210	270	300	480											0 (101.0)	148.5	+0
S02	90 +30	60 +30	210 +180	240	270	450											0 (216.0)	216.0	+0
S03	120	90 +60	180 +150	210 +150	240	420											0 (217.4)	217.4	+0
S04	150	120 +90	150 +120	180 +120	210	390											0 (223.7)	223.7	+0
S05	100	150 +120	120 +90	150 +90	180	360											0 (152.4)	152.4	+0
S06	210	180 +150	90 +60	120 +60	150 +120	330	360										0 (174.8)	174.8	+0
S07	240	210 +180	60 +30	90 +30	120 +90	300	330										0 (203.5)	203.5	+0
S08	270	240 +210	30 (74.5)	60 (15.5)	90 +60	270 +240	300										0 (42.0)	132.0	+0
S09	390	360	90 +60	60 (96.0)	30 (9.9)	150 +120	180										0	105.9	+0
S10	420	390	120 +90	90 +30	60 +30	120 +90	240										0 (104.9)	104.9	+0
S11	450	420	150 +120	120 +60	90 +60	90 +60	210										0 (98.1)	98.1	+0
S12			180 +150	150 +90	120 +90	60 +30	90	210									0 (80.1)	80.1	+0
S13			210	180 +120	150 +120	30 (3.7)	60 +0	180 +150	240								0 (128.3)	132.0	+0
S14			240	210	180 +180	60 +60	30 (84.6)	90 +90	150	210	300						0	84.6	-30
S15			330	300	270 +240	90 +60	60 (6.3)	60 +30	120	180	270	390	420				0 (84.5)	90.8	+0
S16			360	330	300 +270	120 +90	90 +30	30 (13.0)	90 +60	150	240	360	390				0 (77.5)	90.5	+0
S17			420	390	360 +330	180 +150	150 +90	30 +0	30 (13.9)	90 +60	180 +150	300	330	480			0 (5.0)	18.9	+0
S18				450	420	240	210 +150	90 +60	30 +0	30 (9.2)	120 +90	240	270	300			0	9.2	+0
S19				510	480	300	270	150 +120	90 +60	30 (12.4)	60 +30	180 +150	210	240			0 (14.0)	26.4	+0
S20					510	330	300	180	120 +90	60 +30	30 (44.5)	150 +120	180 +120	210	330		0 (53.1)	97.6	+0
S21					570	390	360	240	180 +120	120 +90	30 +0	90 +60	120 +60	150	270		0 (45.4)	45.4	+0
S22						420	390	270	150 +120	60 +30	60 +30	60 +30	90 +30	120	240		0 (70.7)	70.7	+0
S23						450	420	300	240	180 +120	90 +60	30 (40.4)	60 (17.5)	90 +60	210		0	57.9	+0
S24						570	540	420	360	300	210 +180	90 +60	60 (181.3)	30 (41.0)	90 +60	120	0 (204.4)	426.7	+0
S25							570	450	390	330	240	120 +90	90 +30	60 +30	60 +30	90 +60	0 (143.0)	143.0	+0
S26							600	480	420	360	270	150	120 +60	90 +60	30 (5.4)	60 +30	0 (17.9)	23.3	+0
S27								570	510	450	360	240	210	180 +150	60 +30	30 (20.7)	0 (43.8)	64.5	+0
S28									530	470	380	280	230	200	80 +50	50 +20	0 (8.6)	8.6	+0
Total Demand (m^3)	18.6	26.9	74.5	111.5	9.9	3.7	70.9	13.0	13.9	21.6	44.5	40.4	198.8	41.0	5.4	20.7	2505.8	3221.1	
V_j	+60	+30	+30	+60	+30	+30	+60	+30	+30	+30	+30	+30	+60	+30	+30	+30	+0		

Z = 31 515

produces a value of $Z = 31\ 515.0\ m^3m$, representing a reduction of 1 % as compared to the initial feasible solution. Since the total fill requirement for the 16 fill demand areas is equal to 715.3 m^3, this means that the average transportation distance per m^3 is given by:

$$\frac{31\ 515.0\ m^3m}{715.3\ m^3} = 44.1\ m$$

The optimal solution thus represents an average reduction in earthwork transportation of 0.4 m for each m^3 transported, as compared to the initial feasible solution.

3.5 Time study, production and cost data

The objectives of the time study phase of the research were as follows:

(a) To obtain accurate production and cost data during the construction of this new road

(b) To develop procedures for incorporating time study methods into a formal methodology for planning and constructing forest roads.

In order to simplify data collection and to facilitate data interpretation and analysis, the operation was segregated into sub-operations which were time-studied independently. These sub-operations were then either time-studied in detail or on a shift-level basis (gross-data analysis). In addition to time study data, informations on the type of machinery, labour, costs, work place condition and production were also collected for each sub-operation.

3.5.1 Gross data analysis

Mandays and machine hours were the units of time used during the gross-data analysis. The gross-data analysis method was used on those sub-operations considered to be less critical and mostly done on a task work basis. Total mandays on the following sub-operations were recorded: road survey, road alignment, clearing the right of way (tree felling), moulding culverts, digging mitre drains and clearing the utility road leading to the murram pit. For sub-operations like skidding logs out of the roadline, digging stumps on the roadline, grading the road leading to the murram pit, and excavation of murram, total machine hours were recorded. Appendix 4 shows a data form used in the field for recording gross data for various activities.

3.5.2 Detailed time study and work productivity

Detailed time study was used on four sub-operations: earthwork, road grading, murraming or gravelling and culvert installation. These sub-operations were considered to be critical; that is, improvements in the working methodology could substantially reduce total road construction cost.

During the detailed time study, the snap-back method of timing was employed, using a stop watch calibrated in minutes and centiminutes. To permit the identification and analysis of important and time consuming activities, each sub-operation was segregated into work elements. At the start of each work element, the stop watch was started from zero and left to run until the end of the element. At the conclusion of the element, the elapsed time was recorded and the stop watch "snapped back" to zero ready to record the subsequent work element. As a control, where activities took long duration, a wrist watch with a second hand was used.

Recorded times were segregated into work place time, effective working time and delay time. Usually work place time comprises all of the time spent at the work place. This includes the effective time, delay time and scheduled breaks. Effective time is the actual time spent in doing productive work, excluding all delays. Delays are interruptions that interfere with the continuity of work. Delay time was divided into necessary delay time and unnecessary delay time. Necessary delay time included unavoidable interruptions due to the nature of work, while unnecessary delays are those that could be avoided if proper working methodologies were followed. In this study, due to constant supervision, the amount of delay time recorded was very small. This means therefore, only effective working time was used in estimating productivity while in case of production costs, work place time minus scheduled lunch break time was used in estimating production cost. It is important to note that, delay times recorded in this study do not include machinery breakdown or down time. Since breakdowns took a long time to repair, machinery breakdown times are treated separately from the work place delay times. In this study for example, machinery breakdown time was recorded in terms of days rather than minutes or hours.

During earthwork and grading sub-operations, travel times, break time, effective working time and delay times were recorded as shown in Appendix 5. In addition to recording time, data on productivity, costs and working methods were also recorded. Productivity in these sub-operations for example was measured as the quantity of soil excavated, transported and compacted into fill areas per unit time.

To measure earthwork or grading production rates, cross-sectional profiles were made at the end of each day along the stations where work was in progress. The methodology is illustrated as follows:

Station No.	End area scheduled to be cut, m^2	End area cut during the day, m^2	End area scheduled to be filled, m^2	End area filled during the day, m^2
40	0.50	0.28	3.06	1.35
60	2.41	1.19	3.22	1.49

For each station, the end-area was determined before and after the sub-operation. For the above example, at Station 40, out of 0.5 m^2 end area scheduled to be cut, only 0.28 m^2 was cut that day. Using the average end-area formula, the day's volume excavated between stations 40 and 60 is estimated as: $0.5(0.28 + 1.19)(20) = 14.7\ m^3$. Similarly, the day's volume filled between stations 40 and 60 is: $0.5(1.35 + 1.49)(20) = 28.4\ m^3$. A data sheet used in the field for recording data to be used later in determining earthwork productivity is shown in Appendix 6.

Road gravelling or murraming was split into measurable work elements

(Appendix 7) in order to facilitate data collection. Effective working times, delay times and break times were recorded and one round-trip journey formed a work cycle. A work cycle consisted of the following work elements: loading, travelling loaded, unloading and travelling empty.

Loading effective time included the time required to position a truck in the right position plus the actual loading activity. Loading delay time included time required to adjust the releasing chain or removing big stones loaded by mistake. The volume of murram loaded in m^3 was measured as an independent variable by counting the number of back-hoe buckets, i.e., 1 bucket is equal to 0.87 m^3 of murram.

Travelling loaded effective time was the time taken to travel from the murram pit to the road site. Delay time under this activity involved one vehicle side parking to let the other pass when the two vehicles met. Hauling distance in km was measured as an independent variable.

Unloading effective time was the time required to tip-off murram to the road and to make a U-turn ready for the next trip. Unloading delay time included time to release the stuck chain and stones from the truck deck.

Travelling empty effective time was time taken to travel from the road site to the murram pit. Distance travelled from the road site to the murram pit was measured as an independent variable (by recording odometer readings).

Lunch break time was time spent in taking lunch plus a short rest in the forest.

For the culvert installation sub-operation, detailed time studies were used to obtain data pursuant to streamlining the installation methods. Altogether 10 concrete culverts were installed along the road and the first 8 were subjected to detailed time and methods analysis. To be able to collect data and analyse them, the sub-operation was segregated into various work elements. A data form showing these work elements plus other data for this phase of the study is shown in Appendix 8.

Data collected from the installed 8 culverts were analysed first before time requirements for the final two culverts were estimated. Critical path networks were then developed for these two culverts. Incorporated into these networks was the actual operating experience from the preceding installations plus methodological improvements introduced by studying the "bottlenecks" which occured during the earlier installations.

From the critical path networks, critical activities were isolated and schedules to be followed for the installation of the remaining two culverts were drawn. Detailed time studies were made of these two installations and deviations from scheduled times were analysed.

3.5.3 Cost data

For each sub-operation, cost data on labour and equipment (machines or tools) were collected or analysed. Labour was categorized into permanent labour and casual labour.

3.5.3.1 Permanent labour

Permanent labour includes foreman, tractor drivers, chockermen, and a few tree fellers who are paid on a monthly basis. Besides their basic salaries, these workers enjoy the following fringe benefits: free medical services, paid annual leave, transport fare during annual leave, paid sick leave, insurance, food supplies and housing. Cost data for these permanent labour were obtained from the monthly payrolls.

3.5.3.2 Casual labour

Casual labourers are those employed daily on a task work basis. That is, each day they are assigned a specific amount of work, and when this is completed, they are free to leave the workplace. The daily pay for casual labour in this study is shs 23.05 per manday. [1US $ = 12 T.Shs.]

3.5.3.3 Equipment costs

Machine or tool specifications, purchase price and operating costs were obtained from the records maintained by the management of the SUA Training Forest. Where cost records were not available for a specific item, estimated costs were derived using standard engineering economy procedures.

Equipment used in this project whose costs have been computed are:

(i) A County 1164 tractor with a dozer front blade and a back-hoe. This tractor was used for earthmoving, uprooting stumps, preparing the site during construction of the wooden bridge and occasionally for loading murram onto trucks. The cost of owning and running this machine per hour is shown in Appendix 9.

(ii) Ford 6600 tractors with skidding winches and trailers for skidding logs out of the roadline and hauling logs to the wooden bridge. Appendix 9 summarizes the cost of owning and running this type of tractor per hour.

(iii) Isuzu trucks for hauling murram from the murram pit to the road site. Costs per hour are shown in Appendix 10.

(iv) The D6 Caterpillar for clearing the road leading to the murram pit and opening the murram pit.

(v) The D4 Caterpillar for forming the road body.

(vi) A motor grader for grading the road and forming side ditches.

(vii) A 215 Caterpillar Excavator for digging and loading murram onto trucks. The costs of owning and running the above four Caterpillar machines per hour are shown in Appendix 11.

(viii) Crosscut saws and axes for clearing the right of way of standing timber.

(ix) Hand tools like spades, mattocks and hoes for digging stumps, mitre drains, culvert channels and spreading murram on the road surface.

(x) Bush-knives (pangas) for clearing grass and shrubs during the surveying operation.

(xi) Surveying equipment like a Suunto clinometer, compass, ranging poles, leveling staff and wire tape.

Cost data maintained by the Training Forest on items like purchase prices, fuel, oil, insurance and taxes were used to determine the cost of owning and operating the equipment. Other cost items like depreciation and interest, maintenance and repair, tyres etc. whose records were not available or not properly kept were estimated as follows:

(i) Depreciation and interest

As a means of recovering the original investment on the equipment, depreciation and interest costs on these equipment were computed.

Depreciation: the straight line method, which assumes that the value of the equipment declines by a fixed amount each year, was used in computing the depreciation cost. At the end of the equipment's economic life, the salvage value of the equipment was taken to be 10 % of the original purchase price.

Annual depreciation cost by the straight-line method was computed as follows (Dykstra 1982c):

$$D = \frac{P - s}{L}$$

where: D = annual depreciation, in shillings/year

P = purchase price of the equipment, in shillings

s = salvage value of the equipment, in shillings

L = anticipated service life of the equipment, in years.

Interest:

A common method for estimating interest and other costs for engineering analysis makes use of the "average annual investment" (AAI). AAI may be computed as follows (Dykstra 1982c):

$$AAI = \frac{P + s + D}{2}$$

where AAI = average annual investment cost, in shillings.

Knowing the government interest rate, the annual interest charge on the equipment was then estimated as follows:

$$I = (AAI)(i)$$

where: I = estimated annual interest charge, in shillings

i = interest rate expressed as a decimal (the present government interest rate is 8 %; therefore, $i = 0.08$).

(ii) Maintenance and repair

Cost records on machine maintenance and repairs have not been properly kept and therefore they had to be estimated. Elsewhere it has been estimated that annual maintenance and repairs account to about 100 % of the annual depreciation cost (Laarman *et al.* 1981, Peurifoy 1979). Thus, in this study, due to terrain difficulties and the exorbitant high prices of spare parts, annual maintenance and repair costs are assumed to equal the annual depreciation costs; i.e.,

$$MR = D$$

where: MR = annual maintenance and repair costs, in shillings

D = annual depreciation costs, in shillings.

(iii) Tyre cost

Tyre cost per year was estimated as:

$$T = (St)(Pt)$$

where: T = tyre cost per year

St = number of sets of tyres required per year

Pt = price of a set of tyres.

3.6 Data analysis

3.6.1 Statistical analysis

Normally the main objective of data analysis is to extract the main features of relationships that are hidden or implied in an accumulated mass of data (Dykstra 1975). Before analysing the data, dependent and independent variables in each sub-operation were identified and described. Depending on whether the time study data collected were through shift level or detailed time studies, qualitative or detailed statistical analyses were made.

Since the main purpose of analysing gross data is to provide a descriptive summary useful for estimating production rates and costs of future operations, summary statistics and tables were compiled for these data. For detailed time study data, least square regression methods were used to fit mathematical equations to the observed data. Graphical models which can be used by forest project personnel to estimate production rates and costs were also developed. The results of these analyses were then combined into a comprehensive methodology for estimating production rates and costs of future forest road construction projects.

In the regression analysis, the collected data were treated as independent random observations. Equations were computed by least squares to explain the quantitative relationships among the dependent variables (time) and the independent variables (measured attributes such as stump diameter, hauling distance, hauled volume and road gradient). Regression coefficients were tested for statistical significance and confidence limits derived. The sizes of samples required before conclusions could be drawn from the statistical analyses were also determined (Freese 1967).

3.6.2 Critical path analysis

The purpose of critical path analysis is to identify activities which must be completed on time if the duration of a project, defined as an orderly sequence of activities, is to be completed at the scheduled time. The activities so identified are referred to as "critical activities" and the sequence of critical activities in a project is called the "critical path". Efforts to effect methodological improvements in order to reduce project duration are then concentrated on the critical activities, since improvements in non-critical activities will not affect project duration.

In this study, critical path analysis was used to plan and control culvert installation sub-operations in an effort to reduce installation time and costs. Data collected from time studies during the installation of the first eight culverts were used to estimate time requirements for each activity relating to the installation of the remaining two culverts. Critical path networks were developed for these installations and information from the analysis was used to control the actual installation of the remaining two culverts. Detailed time studies made during the installation of the remaining two culverts were analysed for the variations between scheduled times and the actual times.

3.7 Grade optimization

Earthwork costs, which have been found to vary with the designed road gradient (Byrne *et al.* 1960), represent a considerable fraction of the total road construction costs, especially in mountainuous areas. Unlike road maintenance and hauling costs which are incurred repeatedly over the life of a road (Byrne *et al.* 1960), earthwork costs are one-time investments. In order to minimize earthwork costs, the quantity of earthworks have to be minimized. This implies maximizing road gradient. Maximizing road gradient on the other hand will tend to increase road maintenance and hauling costs; over time, the total increase in these costs may offset the advantage of minimum earthwork costs. Therefore, if total costs associated with the road are to be minimized, road maintenance and hauling costs must be considered in addition to earthwork cost.

To determine the optimal road grade for the road project considered in this study, three road grade alternatives: 6 %, 8 % and 10 % were tested for optimality. The optimal road grade is the one which minimizes the sum of earthwork costs, hauling costs and road maintenance costs. Although road maintenance cost varies with gradient, it is not included in the testing of optimality in this study due to the small range of road grades (6-10 %) analysed here and the lack of reliable road maintenance cost data. Thus, only the sums of earthwork costs and hauling costs of the above three road grade alternatives are worked out and the one with minimum value is considered to be the optimal grade.

3.7.1 Vehicle hauling costs

In order to estimate the relationship between vehicle operating costs and road grade, it was assumed that fuel consumption was the only component of hauling cost which varies with road grade. As long as the road grades are not extreme (Yamamoto 1981), this assumption holds; on very steep grades, however, engine and tyre lives are reduced and maintenance costs associated with the brakes and drivetrain increase.

Fuel consumption varies with gross power developed by the engine. On adverse grades, fuel consumption increases because more power is required to climb the grade while on favourable grades, fuel consumption increases also because the engine works to retard speed through the use of lower gears.

To estimate fuel consumption per hour when gross engine power (PG) is known, the following formula is used (Dykstra 1981b):

$$F = \frac{(R)(PG)(L)}{100W} \tag{4}$$

where: F = fuel consumed, in l/h

R = specific fuel consumption rate, in kg/kWh

PG = gross engine power available from the truck engine, in kW

L = fuel consumption load factor, in %

W = unit mass of fuel, in kg/l.

The unit mass of diesel fuel is taken as 0.84 kg/l and the specific fuel consumption rate for diesel engines averages approximately 0.23 kg/kWh (Dykstra 1981b).

The fuel consumption load factor, which depends on the severity of service to which the engine is applied, measures the actual fuel consumption over a route as a percentage of the maximum capacity of the engine to burn fuel (Dykstra 1981b). It is expressed as:

$$L = (P/PG)100 \tag{5}$$

where: P = <u>actual gross</u> engine power required to negotiate a particular grade at the designed speed, in kW.

Usually PG is given in the manufacturer's specifications or estimated as PG = Pn/0.9 in case the net engine power rating (Pn) is specified. Pn is the power available at the flywheel when the engine is operating with all accessories like fan, alternator, air compressor, fuel and water pumps attached to the engine (Dykstra 1981b, FAO 1977).

When a vehicle is negotiating a grade, enough power has to be produced to overcome motion resistance at the desired travel speed. The total power required by a vehicle to drive up the grade is called actual gross engine power (P) and is estimated as:

$$P = PN/0.9 \tag{6}$$

PN is the actual net engine power required to overcome motion resistance on a particular grade at a specified speed. PN depends on the road grade, vehicle weight, road standard and travel speed. Some power is lost in the transmission of mechanical energy from the engine's crankshaft to the driving axle. Since only a fraction of the actual net engine power (PN), about 85 %, reaches the driving axle, the drive-train efficiency (E) of trucks is taken as 0.85 although it ranges from 0.75 to 0.95 (McNally 1975). Power available on the driving axle (PD) is usually estimated as:

$$PD = (PN)(E) \quad = \quad 0.85\ PN \tag{7}$$

To determine the actual net engine power (PN) required to negotiate each of the three road grade alternatives, i.e., 6 %, 8 % and 10 %, the following assumptions were made:

- The influence of air resistance was negligible since the maximum allowable speed set for SUA Training Forest is 40 km/h. Normally air resistance is significant only on highways where speeds are over 50 km/h (McNally 1975).

- Altitude would not significantly influence the engine power since the operating altitude is only 2000 m above sea level (McNally 1975).

- Rolling and grade resistances were assumed to be the major motion-resistance forces to be overcome when driving up the grade.

Rolling resistance (RR) is defined as a measure of the retarding effect of the road to forward movement of the vehicle. On a given road grade, RR is computed as (Dykstra 1981c):

$$RRG = (u + aV)\ W \cos\theta \qquad (8)$$

where: RRG = rolling resistance on a given grade, in N

W = gross vehicle weight, in N

u = coefficient of rolling resistance, in N/N of W

V = velocity, in km/h

a = 0.00006 N.h/km per N of W (if V is in m/s, then a = 0.00022 N.s/m per N of W)

θ = the grade angle in degrees.

Grade resistance (GR) is the force required to overcome the force of gravity when climbing a grade. GR in newtons is expressed as (Dykstra 1981c):

$$GR = W \sin\theta \qquad (9)$$

For a truck to overcome motion resistance when negotiating a given grade at a specified speed, the engine has to develop enough power. Since power is defined as force x velocity, PN required to overcome motion resistance at a specified grade and designated speed is determined as follows (Dykstra 1981c):

$$PN = (RRC + GR)V/(3600\ E) \qquad (10)$$

where: 3600 = factor to convert from N.km/h to kW

Before determining the net engine power (PN) required to overcome motion resistances when negotiating the three road grade alternatives i.e., 6 %, 8 % and 10 %, the anticipated average travel speeds (V) were computed. When computing average speeds, the following observations were made on this particular road:

- trucks will be travelling empty when traversing the adverse grade and when loaded, they will be on a favourable grade
- the road alignment consists of a considerable number of sharp curves which are likely to limit the truck's speed.

3.7.1.1 <u>The average travel speeds</u>

3.7.1.1.1 <u>Empty truck</u>

Since trucks will be travelling empty when negotiating the adverse grade, it is likely that power will not be the limiting factor. Also, despite the maximum allowable speed in SUA Training Forest being 40 km/h, it will be very difficult to maintain this speed due to the alignment of this particular road. Byrne *et al.* (1960) and

Dykstra (1981d) have shown that where speed is limited by alignment, the effects of deceleration into curves and acceleration out of them have to be considered when computing the average speed. The road under study being a single-lane access road with side ditches, data for such roads from Byrne *et al.* (1960) and Dykstra (1981d), were fitted to the following log-linear equation:

$$Vc = -25.8 + 12.0 \log_e(R) \quad (r^2 = 0.99) \tag{11}$$

where: Vc = average travel speed on a curve of radius R, in km/h

R = radius of curvature, in m

$\log_e(R)$ = natural logarithm of R

r^2 = coefficient of determination.

By use of equation (11), the average travel speeds on various curves encountered on this particular road are computed and the results summarized in Table 8.

Table 8. Estimated average speeds on curves of different radii.

Curve radius R (m)	Average speed V (km/h)	Average speed V (m/s)
20	10	2.78
25	13	3.61
33.3	16	4.44
40	19	5.28
50	21	5.83
66.6	25	6.94
100	30	8.33
136	33	9.17

To be able to determine the overall average speed on this road, the weighted average speeds on both tangent sections and on curves had to be computed. Again, although the maximum allowable speed is 40 km/h, due to road alignment, considerations had to be given to time and distance required to decelerate into curves and accelerate out of these curves when computing the average speeds on tangents (Byrne *et al.* 1960, Dykstra 1981d). Principles of determining deceleration and acceleration times are discussed in the following paragraphs.

(i) Deceleration

Force is defined as mass multiplied by acceleration. Since deceleration is negative acceleration, the force required to brake a vehicle decelerating into a curve is found as:

$$F = md$$

where: F = braking force required to decelerate a vehicle, in N

m = mass of the vehicle, in kg

d = deceleration rate, in m/s^2

If W = vehicle weight, in N, and g = the constant of gravitational acceleration (9.81 m/s^2) then $m = W/g$ and

$$F = \frac{Wd}{g} \quad \text{or} \quad \frac{F}{W} = \frac{d}{g}$$

The coefficient of friction (or traction) between the tyres of the braking vehicle and the road surface is by definition: $f = F/W$

Thus, $f = d/g$ or $d = fg$

For a vehicle decelerated from a speed of $V1$ m/s to a speed of $V2$ m/s, the distance travelled during deceleration is given by McLean and Nelson (1962):

$$D = \frac{V1^2 - V2^2}{2d} = \frac{V1^2 - V2^2}{2fg} \tag{12}$$

If the rate of deceleration is constant, then the product of this rate times deceleration time must be equal to the change in speed:

$$V1 - V2 = dt = fgt$$

so that

$$t = \frac{V1 - V2}{fg} \tag{13}$$

Equations (12) and (13) provide a model for calculating the distance and time required for a vehicle to decelerate from an initial speed to a lower speed. It is important to notice that vehicle weight has no effect on deceleration distance or time (assuming that the brakes are properly adjusted and that the vehicle is not overloaded with respect to brake capacity). However, the weight of the vehicle does affect the total time required for the vehicle to slow down after the driver depresses the brake pedal (Fitch 1976). In any vehicle there is a lag between the time the braking mechanism is activated and the time sufficient pressure has built up in the braking system for the braking force to take effect. A heavier vehicle will require a greater breaking force, which implies higher pressure in the braking system and thus, a longer lag time. However, lag time is not of interest in the present analysis because during the lag time, the vehicle continues to travel at the initial speed, V1. The driver will presumably begin braking sufficiently in advance of the curve so that the lag time will have lapsed and the vehicle will have decelerated to curve speed prior to entering the curve.

(ii) Acceleration

Acceleration of a vehicle coming out of a curve is governed by the same principles as deceleration going into the curve, beginning with the definition of force,

$$F = ma$$

where: a = acceleration, in m/s^2.

since $m = W/g$, $a = gF/W$.

In this case F, the force available to accelerate the mass of the vehicle, is limited by the power the vehicle's engine can develop. From equations (8) and (9), on an adverse grade the force required to negotiate the grade may be expressed as

$$FG = RRG + GR$$

If the maximum available net engine power is Pn, the drive-train efficiency is E and the speed on the curve is V1, then the excess power available to provide an accelerating force is given by:

$$EPn - FGV1$$

since FGV1 is the power required to negotiate the grade at speed V1.

Furthermore, by the relationship between force and power, the accelerating force F is:

$$F = \frac{EPn - FGV1}{V1}$$

Therefore, if Pn is in watts, F, FG and W are in newtons, and V1 is in m/s, then

$$a = \frac{g(EPn - FGV1)}{V1W}$$

or, substituting from equation (8),

$$a = \{9.81(EPn - V1(u + 0.00022V1)W \cos \theta - V1\ W \sin \theta)\}/V1W \quad (14)$$

Then, time to accelerate from V1 to V2 is given by:

$$V2 - V1 = at \quad \text{or} \quad t = (V2 - V1)/a \quad (15)$$

Distance travelled during acceleration is

$$D = (V2^2 - V1^2)/2a \quad (16)$$

In the SUA Training Forest, the most common type of truck which is likely to be used for hauling logs is a 7 t Isuzu truck. The net power rating (Pn) of this truck is 92 kW and it weighs approximately 42 200 N when empty and 97 200 N when loaded. Its drive-train efficiency (E) is assumed to be 0.85, while the coefficient of friction (or traction) between the tyres and the road surface is estimated to be 0.3, following McNally (1975). Due to the type of murram used on this road (sandy-gravel), the unit rolling resistance (u) is assumed to be 0.02 (McNally 1975). The lengths and gradients of tangent sections and the curves encountered on this research road are shown in Figure 6.

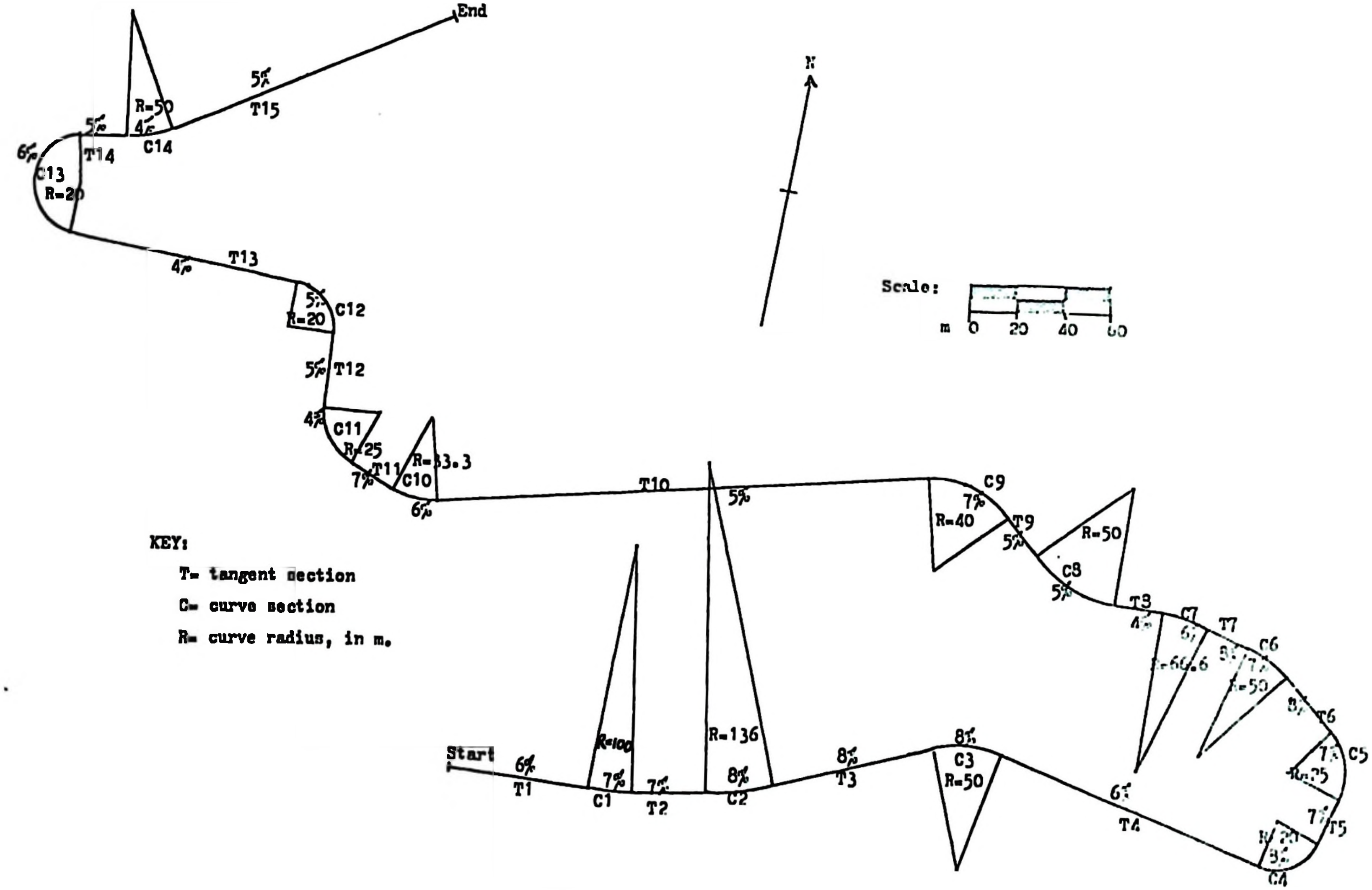


Figure 6. Tangent and curve sections encountered on the new road.
Grades shown are for the alternative with a maximum grade of 8 %.

Although the maximum allowable speed on tangents is 40 km/h, this is true only up to a certain grade, beyond which the grade will limit the travel speed. When equation (10) is used, the maximum grade at which the unloaded Isuzu truck is able to travel at a speed of 40 km/h is 15 %, beyond which, the grade starts limiting the speed. For example, when the road grade is 16 %, the maximum speed is only 38 km/h. Since there are no sections of the road (among the selected three road grades 6 %, 8 % and 10 %) which are steeper than 15 %, for the situation considered in this analysis, the road grade will not limit the truck's travel speed when travelling empty.

To find the truck's overall average speed when travelling empty over this research road, weighted average speeds on tangents and on curves were computed. For example, to determine the average speed on the first tangent section (T1 in Figure 6), it was assumed that the truck will be at a maximum speed when entering the new road and before negotiating the first curve (C1), it will have to decelerate from 40 to 30 km/h or from 11.11 to 8.33 m/s (Table 8). From equation (12), the distance travelled during the deceleration period would thus be:

$$D = \frac{11.11^2 - 8.33^2}{(2)(0.3)(9.81)} = 9.2 \text{ m}$$

and from equation (13), the time taken would be:

$$t = \frac{11.11 - 8.33}{(0.3)(9.81)} = 0.9 \text{ s}$$

Since the tangent section is 60 m (T1 = 60 m), the vehicle will travel 50.8 m (60 - 9.2 m) at a speed of 40 km/h and 9.2 m at an average speed of 35 km/h $\{(40 + 30 \text{ km/h})/2\}$. Usually during deceleration or acceleration, the average speed will be the average of the speed on the adjacent curves and the maximum allowable speed. The weighted average speed on T1 will therefore be

$$\{(50.8 \text{ m})(40 \text{ km/h}) + (9.2 \text{ m})(35.0 \text{ km/h})\}/60 \text{ m} = 39.2 \text{ km/h}$$

From Table 8, speed on C1 (100 m) is 30 km/h. Moving out of C1, the vehicle has to accelerate from 30 to 40 km/h or from 8.33 to 11.11 m/s. Acceleration "a" is computed by use of equation (14) as follows:

$$a = [(9.81)\{(0.85)(92\ 000) - 8.33(0.02 + 0.00022(8.33))42200$$
$$\cos(\tan^{-1}(0.07)) - 8.33(42200) \sin(\tan^{-1}(0.07))\}]/$$
$$8.33(42200)$$
$$= 1.47 \text{ m/s}^2$$

The distance travelled on T2 before the maximum speed is attained is computed using equation (16):

$$D = (11.11^2 - 8.33^2)/(2)(1.47) = 18.4 \text{ m}$$

and by use of equation (15), time taken during the acceleration period is:

$$t = (11.11 - 8.33)/1.47 = 1.9 \text{ sec.}$$

The average speed when accelerating from 30 to 40 km/h (C1 to T2) will be 35 km/h. But, before negotiating the second curve (C2), the truck has to decelerate from the maximum speed to curve speed (i.e., from 40 to 33 km/h). Time and distance required when decelerating from 40 to 33 km/h are computed as:

$$D = (11.11^2 - 9.17^2)/(2)(0.3)(9.81) = 6.7 \text{ m}$$

$$t = (11.11 - 9.17)/(0.3)(9.81) = 0.7 \text{ sec.}$$

The average speed when decelerating from 40 to 33 km/h will be 36.5 km/h. Since T2 is 30 m in length, it means therefore, the length of this section covered at a maximum speed of 40 km/h is 30 - 18.4 - 6.7 = 4.9 m. The weighted average speed when travelling over T2 then becomes:

$$\{(18.4 \text{ m})(35 \text{ km/h}) + (4.9 \text{ m})(40 \text{ km/h}) + (6.7 \text{ m})(36.5 \text{ km/h})\}/30 \text{ m}$$
$$= 36.2 \text{ km/h.}$$

The same procedure is used for the remainder of the road except where the tangent sections are short (≦ 30 m) and the adjacent curves are sharp (≦ 50 m). Where that is the case, the average tangent speed is the average speeds of the adjacent curves. In such case, distance do not allow the driver to accelerate from the curve speed to the maximum speed and then decelerate to the average speed of the next curve. For instance, the average speed of T5 is found by taking the average speeds on C4 and C5; i.e., (10 + 13)/2 = 11.5 km/h. Other tangent sections whose average speeds have been computed similarly are T7, T8, T9, T11, T12 and T14. Table 9 summarizes the weighted average speeds and time taken on different curves and tangent sections when travelling empty and loaded.

The overall average truck speed when travelling empty is found by dividing the total distance (km) by the total travel time (h); i.e.,

$$\left[\frac{1300 \text{ m}}{229.0 \text{ s}}\right]\left[\frac{\text{km}}{1000 \text{ m}}\right]\left[\frac{3600 \text{ s}}{\text{h}}\right] = 20.4 \text{ km/h}$$

Table 9. Average travelling times and speeds on tangents and curve sections when empty and loaded.

Section No.	Peg No.	Tangent (T) Curve (C) Section No.	Distance (m)	Average speed (km/h)		Travelling time (s)		Accelerating time (s)		Decelerating time (s)		Total time (s)	
				Empty	Loaded	Empty	Loaded	Empty	Loaded	Empty	Loaded	Empty	Loaded
01	0- 60	T1	60	39.2	38.3	5.5	5.6	-	2.1	0.9	-	6.4	7.7
02	60- 80	C1	20	30.0	30.0	2.4	2.4	-	-	-	-	2.4	2.4
03	80- 110	T2	30	36.2	31.5	3.0	3.4	1.9	-	0.7	-	5.6	3.4
04	110- 140	C2	30	33.0	33.0	3.3	3.3	-	-	-	-	3.3	3.4
05	140- 210	T3	70	36.7	33.8	6.9	7.5	1.9	2.1	1.8	0.2	10.6	9.8
06	210- 240	C3	30	21.0	21.0	5.1	5.1	-	-	-	-	5.1	5.1
07	240- 360	T4	120	35.0	36.6	12.3	11.8	2.2	2.6	2.8	1.8	17.3	16.2
08	360- 390	C4	30	10.0	10.0	10.8	10.8	-	-	-	-	10.8	10.8
09	390- 410	T5	20	11.5	11.5	6.3	6.3	-	-	-	-	6.3	6.3
10	410- 440	C5	30	13.0	13.0	8.3	8.3	-	-	-	-	8.3	8.3
11	440- 470	T6	30	17.0	17.0	6.4	6.4	-	-	-	-	6.4	6.4
12	470- 490	C6	20	21.0	21.0	3.4	3.4	-	-	-	-	3.4	3.4
13	490- 510	T7	20	23.5	23.5	3.1	3.1	-	-	-	-	3.1	3.1
14	510- 530	C7	20	25.0	25.0	2.9	2.9	-	-	-	-	2.9	2.9
15	530- 550	T8	20	23.5	23.5	3.1	3.1	-	-	-	-	3.1	3.1
16	550- 590	C8	40	21.0	21.0	6.9	6.9	-	-	-	-	6.9	6.9
17	590- 610	T9	20	20.0	20.0	3.6	3.6	-	-	-	-	3.6	3.6
18	610- 630	C9	20	19.0	19.0	3.8	3.8	-	-	-	-	3.8	3.8
19	630- 840	T10	210	38.9	37.8	19.4	20.0	2.1	3.2	2.3	2.0	23.8	25.2
20	840- 860	C10	20	16.0	16.0	4.5	4.5	-	-	-	-	4.5	4.5
21	860- 880	T11	20	14.5	14.5	5.0	5.0	-	-	-	-	5.0	5.0
22	880- 910	C11	30	13.0	13.0	8.3	8.3	-	-	-	-	8.3	8.3
23	910- 940	T12	30	11.5	11.5	9.4	9.4	-	-	-	-	9.4	9.4
24	940- 970	C12	30	10.0	10.0	10.8	10.8	-	-	-	-	10.8	10.8
25	970-1070	T13	100	37.5	34.2	9.6	10.5	1.4	2.7	2.8	2.8	13.8	16.0
26	1070-1130	C13	60	10.0	10.0	21.6	21.6	-	-	-	-	21.6	21.6
27	1130-1150	T14	20	15.5	15.5	4.6	4.6	-	-	-	-	4.6	4.6
28	1150-1170	C14	20	21.0	21.0	3.4	3.4	-	-	-	-	3.4	3.4
29	1170-1300	T15	130	37.9	38.9	12.3	12.0	2.2	-	-	1.8	14.5	13.8
TOTAL												229.0	229.1

3.7.1.1.2 Loaded truck

To determine the overall average vehicle speed when the loaded truck is travelling on the favourable grade, the same procedure used in Section 3.7.1.1.1 of computing the average speeds on tangents and on curves is used. As when the vehicle is travelling on adverse grade, the maximum descending speed on tangent sections is 40 km/h. However, when descending, the grade limits the maximum descending speed especially at lower grades. An empirical formula by Byrne *et al.* (1960) (which is used here in determining the maximum descending speeds on various grades) shows that the maximum grade at which the truck can descend at a speed of 40 km/h is 6 %. For steeper grades, vehicle speeds are limited by road grades. Table 10 shows the maximum descending speeds on different grades as derived from the formula of Byrne *et al.* (1960).

$$V = 390/(3 - G) \tag{17}$$

where: V = vehicle speed on a favourable grade, in km/h

G = the favourable road gradient, in % ($G < 0$).

Table 10. Estimated maximum speeds when travelling on a favourable grade with a load.

Descending grade (%)	Maximum speed (km/h)
-4	55.7
-5	48.8
-6	43.3
-7	39.0
-8	35.5
-9	32.5
-10	27.9

Speed limited by grade

When a vehicle is accelerating out of a curve while descending the grade, in addition to engine power, the grade force will also assist the acceleration. Rolling resistance on the other hand will tend to retard acceleration and thus, the excess power available for acceleration is:

$$EPn - V1RRG + V1GR$$

and

$$a = [(9.81)\{EPn - V1(u + 0.00022\ V1)W \cos \theta + V1W \sin \theta\}]/V1W \quad (18)$$

The descending speeds on all tangent sections and on curves are computed and compared with the maximum permitted speeds. The ones with lower values are selected and used in computing the overall weighted average speed.

The gross vehicle weight when loaded is estimated to be 97 200 N. Starting from the upper end of the road, (Fig. 6), the average speeds on tangents and on curves are computed as discussed below.

To determine the average speed on T15 (Fig. 6), the assumption is made that the vehicle will be at a maximum speed (40 km/h) when descending T15 (a -5 % grade). To be able to negotiate curve C14, the vehicle will have to decelerate from 40 to 21 km/h (Table 8). Distance and time taken during the deceleration process are estimated by equations (12) and (13) respectively.

$$D = (11.11^2 - 5.83^2)/(2)(0.3)(9.81) = 15.2 \text{ m}$$

$$t = (11.11 - 5.83)/(0.3)(9.81) = 1.8 \text{ s}$$

For a distance of 15.2 m, the truck will be travelling at an average speed of (40 + 21)/2 = 30.5 km/h. For the rest of the section, i.e., 130.0 m − 15.2 m = 114.8 m, the truck will be travelling at a maximum speed of 40 km/h. Therefore, the weighted average speed on tangent T15 is:

$$\{(15.2 \text{ m})(30.5 \text{ km/h}) + (114.8 \text{ m})(40 \text{ km/h})\}/130 \text{ m} = 38.9 \text{ km/h}$$

The maximum safe speed when negotiating a 50 m radius curve is 21 km/h (Table 8). Therefore, the maximum speed on C14 is 21 km/h. Again, since the distance between C13 and C14 is only 20 m and the curve radii are 20 and 50 m respectively, the average speed on T14 is (21 + 10)/2 = 15.5 km/h.

On C13, the maximum safe speed is 10 km/h (Table 8). After negotiating C13, the vehicle has to accelerate from the curve speed to the maximum permitted speed (40 km/h) when it enters T13. Before determining distance and time required to accelerate from 10.0 km/h to 40 km/h, equation (18) is used to find the rate of acceleration:

$$a = [9.81\{(0.85)(92000) - 2.78(0.02 + 0.00022(2.78))\ 97200 \cos(\tan^{-1}(0.04)) + 2.78(97200) \sin(\tan^{-1}(0.04))\}]/\{2.78(97200)\}$$

$$= 3.03 \text{ m/s}^2$$

Given acceleration "a", then

$$D = (11.11^2 - 2.78^2)/(2)(3.03) = 19.1 \text{ m}$$

and $t = (11.11 - 2.78)/3.03 = 2.7 \text{ s}$

When accelerating from 10.0 to 40.0 km/h (over a distance of 19.1 m) the average speed will be {(10.0 + 40.0)/2} = 25 km/h. Again, before negotiating C12, this truck will have to decelerate from the maximum speed to the curve speed i.e., 40.0 to 10.0 km/h. During this process (whereby the average speed is 25 km/h) the distance covered will be:

$$D = (11.11^2 - 2.78^2)/(2)(0.3)(9.81) = 19.7 \text{ m}$$

Since the total length of T13 is 100 m and 19.1 + 19.7 m of this length is covered at a speed of 25 km/h, it means therefore, the rest, 100 - (19.1 + 19.7) = 61.2 m, is covered at the maximum speed of 40 km/h. Thus, the weighted average speed on tangent T13 is:

$$\{(19.1 \text{ m})(25.0 \text{ km/h}) + (61.2 \text{ m})(40.0 \text{ km/h}) + (19.7 \text{ m})(25.0 \text{ km/h})\}/ 100 \text{ m} = 34.2 \text{ km/h}.$$

Average speeds for the rest of the tangent and curve points are computed in the same manner except where the descending grades on tangents are 7 % or more. As an example, the weighted average speed for T3, which is 8 %, is computed as follows:

From Table 10, the maximum descending speed for an 8 % grade is 35.5. Thus, after negotiating C3 (Fig. 6), the truck will have to accelerate from 21 km/h (Table 10) to 35.5 km/h instead of 40 km/h. The average speed when accelerating out of C3 will then be (21.0 + 35.5)/2 = 28.3 km/h, while the rate of acceleration will be:

$$a = [9.81\{(0.85)(92000) - 5.83(0.02 + (0.00022)(5.83))\ 97200 \cos(\tan^{-1}(0.08)) + 5.83(97200) \sin(\tan^{-1}(0.08))\}]/(5.83)(97200) = 1.9 \text{ m/s}^2$$

Given the rate of acceleration "a", distance travelled is:

$$D = (9.86^2 - 5.83^2)/(2)(1.9) = 16.6 \text{ m}$$

From the tangent speed of 35.5 km/h (when travelling over T3), the driver has to decelerate to 33 km/h in order to be able to negotiate C2. During this process, the average speed will be (35.5 + 33.0)/2 = 34.3 km/h and the distance covered at this speed will be:

$$D = (9.86^2 - 9.17^2)/(2)(0.3)(9.81) = 2.2 \text{ m}$$

Since for a distance of 16.6 m (in T3) the truck will be at a speed of 28.3 km/h and 34.3 km/h for a distance of 2.2 m, it means for the rest of the section i.e., 70 - 16.6 - 2.2 = 51.2 m, the truck will be travelling at a maximum speed of 35.5 km/h. Therefore, the weighted average speed for T3 becomes:

$$\{(16.6 \text{ m})(28.3 \text{ km/h}) + (51.2 \text{ m})(35.5 \text{ km/h}) + (2.2 \text{ m})(34.3 \text{ km/h})\}/70 \text{ m} = 33.8 \text{ km/h}$$

Table 9 summarizes the truck's average speeds and time taken when descending the designed 8 % maximum grade with a load. From Table 9 the average speed when descending with a load is therefore:

$$\left[\frac{1300 \text{ m}}{229.1 \text{ s}}\right]\left[\frac{\text{km}}{1000 \text{ m}}\right]\left[\frac{3600 \text{ s}}{\text{h}}\right] = 20.4 \text{ km/h}$$

3.7.1.2 <u>Required net power (PN) when empty</u>

As computed in Section 3.7.1.1.1, the weighted average speed when travelling empty over the designed 8 % road grade is 20.4 km/h. The overall average net engine power required by the empty Isuzu truck to climb the designed maximum road grade (8 %) based on equation (10) is therefore:

$$PN = [\{0.02 + 0.00006(20.4)\}\ 42200(0.997) + 42200(0.078)]\ 20.4/3600(0.85)$$

$$= 27.9 \text{ kW}$$

3.7.1.3 Required net power (PN) when loaded

The average descending speed when the loaded truck is going downhill is 20.4 km/h (3.7.1.1.2). When a vehicle is going downhill, less power is required since both gravity and grade assist in acceleration. According to Yamamoto (1981), the power required in going downhill is about 40 % of the power required by the same truck with the same load when going uphill at the same grade.

In order to find the overall average net power (PN) required to go downhill, the PN which would have been required if the loaded truck (97 200 N) was to go uphill is computed first by use of equation (10):

$$PN = [\{0.02 + 0.00006(20.4)\}\ 97200(0.997) + 97200(0.078)]\ 20.4/3600(0.85)$$

$$= 64.3 \text{ kW}$$

If the power required to go downhill is 40 % of the power required to go uphill (Yamamoto 1981), then the loaded truck will need a net power of: 64.3 x 0.4 = 25.7 kW.

3.7.1.4 Fuel consumption on adverse grade

As stated in Section 3.7.1, P (gross engine power required to climb a grade) is used to compute the fuel consumption load factor (L). Since the overall average net engine power (PN) required to climb the adverse grade is known (PN = 27.9 kW), P is estimated as PN/0.9 = 31.0 kW (assuming that net power is 90 % of gross power). Also, since the flywheel engine power (Pn) is given as 92 kW, gross power (PG) is estimated to be: 92/0.9 = 102 kW. Thus, the load factor (L) when going uphill is:

$$L = (31)(100)/102 = 30.4\ \%$$

Since all the variables required by equation (4) are now known, the average fuel consumption per hour when travelling empty on an 8 % adverse grade is estimated as:

$$\frac{(0.23\ \text{kg/kWh})(102\ \text{kW})(30.4)}{(100)(0.84\ \text{kg/l})} = 8.5\ \text{l/h}$$

3.7.1.5 Fuel consumption on favourable grade

Since the overall average PN required to go downhill is 25.7 kW, P is estimated as PN/0.9 or 25.7/0.9 = 28.6 kW. Given PG (102 kW), the load factor on an 8 % favourable grade becomes:

$$L = \frac{(28.6)(100)}{102} = 28.0\ \%$$

Thus, the average fuel consumption per hour on the favourable grade is:

$$\frac{(0.23\ \text{kg/kWh})(102\ \text{kW})(28.0)}{(100)(0.84\ \text{kg/l})} = 7.8\ \text{l/h}$$

3.7.1.6 Fuel cost

Since a litre of diesel costs shs 5.00/l, fuel cost per hour when travelling on the 8 % adverse grade is:

shs 5.00/l x 8.5 l/h = shs 42.50/h

and when travelling on the favourable grade,

shs 5.00/l x 7.8 l/h = shs 39.00/h.

The average fuel cost per hour when travelling empty and loaded is therefore shs (42.50 + 39.00)/2 = shs 40.75/h.

3.7.1.7 Truck owning and running costs per hour

Only fuel cost is assumed to be affected by road grade (Yamamoto 1981). Other costs for the 7 t Isuzu truck are computed as shown in Appendix 10. By adding the average fuel cost (shs 40.75/h) to the hourly computed costs on depreciation, interest, insurance, tyres, labour, vehicle maintenance and repair as shown in Appendix 10, the total travelling cost is determined. For example, when driving over the designed 8 % road grade, the cost of owning and running the 7 t Isuzu truck is:

shs 40.75 + 102.50 = shs 143.25/h.

For the other two road grade alternatives, i.e., 6 % and 10 %, vehicle owning and running costs were computed in the same way. Table 11 summarizes the calculated costs of owning and running a vehicle on these two road grade designs. Note that the cost for the

8 % design gradient (shs 143.25/h) falls between the costs for the other two gradients.

Table 11. Vehicle owning and running costs on 6 % and 10 % road grades.

Road grade		6 %		10 %	
Empty (E) or Loaded (L)		E	L	E	L
Average speed,	in km/h	20.5	20.4	20.2	20.5
Required net power (PN),	in kW	23.2	21.3	35.0	32.7
Load factor,	in %	25.3	23.2	39.2	35.6
Fuel consumption,	in l/h	7.1	6.5	10.9	9.9
Fuel cost,	shs/h	35.50	32.50	54.50	49.90
Average fuel cost,	shs/h	34.00		52.00	
Other truck costs,	shs/h	102.50		102.50	
Total owning and running cost,	in shs/h	136.50		154.50	

3.7.1.8 Total hauling cost

Vehicle hauling costs over a given road grade depend on the road standard, vehicle speed, load volume and the different grade sections encountered along the route. The analysis has so far assumed that the entire road grade is a single grade; e.g., all the road sections (tangent and curve sections) have a grade of 8 % and the hauling speed irrespective of whether the vehicle is empty or loaded or whether it is on a tangent or on a curve is the calculated average; i.e., 20.4 km/h. Such assumptions can be misleading when it comes to computing actual hauling costs.

To determine actual hauling costs over a forest road, specific details of the route (curvature and gradient along each section of road) should be considered. For the designed 8 % road grade, Figure 6 shows grades along the different road sections while Table 9 shows the computed speeds when travelling empty and loaded over these road sections.

For each of the three road grade designs discussed in this study, total unit hauling cost, which is expressed as the cost of hauling 1 m^3 of logs over a 1 km distance was determined as follows (Dykstra 1982d):

$$CH = [\sum_{k=1}^{K} \{(Ct)(Lk)/(V) \ (SLk + SEk)/(2)\}] / \sum_{k=1}^{K} Lk \qquad (19)$$

where: CH = total unit hauling cost, in shs/m^3km

K = total number of road sections considered (K = 29 for the 8 % design alternative shown in Figure 6 and Table 9)

Ct = total owning and running cost per hour, in shs/h

Lk = the length of road section k, in m

V = the average load volume per trip, in m^3

SLk = the average hauling speed over section k when the truck is loaded, in km/h

SEk = the average speed over section k when the truck is empty, in km/h.

Since the cost of owning and running a vehicle (Ct) will vary from one road grade alternative to another and even within the same road grade design, computations to determine Ct over different road sections were carried out using the procedure described in Sections 3.7.1.2 to

3.7.1.7. As an illustration, Ct when hauling over tangent section T1 for the 8 % design alternative (Table 9) is computed as follows:

From Figure 6, the grade on section T1 is 6 % while the computed travel speed when empty is 39.2 km/h (Table 9).

By means of equation (10), the net power (PN) required by the Isuzu 7 t truck to negotiate a 6 % adverse grade at a speed of 39.2 km/h when empty is:

$$PN = \frac{[\{(0.02) + 0.00006(39.2)\}\ 42200(0.998) + 42200(0.060)]39.2}{3600(0.85)}$$

$$PN = 44.5 \text{ kW}$$

The net engine power required by the Isuzu 7 t truck to descend this section of the road at a hauling speed of 38.3 km/h when loaded is:

$$PN = \frac{[\{(0.02) + 0.00006(38.3)\}\ 97200(0.998) + 97200(0.060)]38.3}{3600(0.85)}$$

$$PN = 100.1 \text{ kW}$$

Since the truck is going downhill, the net power will be approximately (100.1)(0.4) = 40.0 kW.

The load factor when going uphill (empty) is:

$$\{(44.5 \text{ kW}/0.9)(100)\}/102 \text{ kW} = 48.5 \%$$

and therefore computed fuel consumption is:

$$\frac{(0.23 \text{ kg/kWh})(102 \text{ kW})(48.5)}{(100)(0.84 \text{ kg/l})} = 13.6 \text{ l/h}$$

Similarly, the load factor when going downhill (loaded) is:

$$\{(44.0 \text{ kW}/0.9)(100)\}/102 \text{ kW} = 43.6 \%$$

Fuel consumption is therefore:

$$\frac{(0.23 \text{ kg/kWh})(102 \text{ kW})(43.6)}{(100)(0.84 \text{ kg/l})} = 12.2 \text{ l/h}$$

Average fuel consumption over the road section is {(13.2 + 12.2)/2} = 12.9 l/h. The average cost of fuel when travelling over T1 is: 12.9 l/h x shs 5.00/l = shs 64.50/h, while the cost of owning and running a vehicle (Ct) over T1 is shs 64.50 + shs 102.50 = shs 167.00/h. This value is entered in the fifth column of Table 12. Ct values for the remaining sections of the road are calculated similarly and are shown in the Table.

Table 12 summarizes the procedure used to calculate hauling costs for all the road sections encountered on the designed 8 % road grade alternative.

Since the total length of these road sections ($\sum_{k=1}^{29} Lk$) is 1300 m, the average cost of hauling 1 m^3 of logs over 1 km distance is:

$$\frac{\text{shs } 1141.30 \text{ m/m}^3\text{km}}{1300 \text{ m}} = \text{shs } 0.88/\text{m}^3\text{km}$$

The total hauling cost for the other two design grades, 6 % and 10 %, are computed in the same way. Since the road alignment is the same for all three grade alternatives, the same road sections as indicated in Table 9 are used. Appendices 12 and 13 show the computed total hauling costs over the different road sections for the designed 6 % and 10 % road grades respectively. From Appendix 12, the average unit

Table 12. Estimating total hauling cost over the designed 8 road grade based on equation (19).

Section No.	Name of section	Length of section, Lk m	Average gradient %	Computed Ct shs/h	S = (Lk + Lk)/2 km/h	(V)(S) m^3km/h	$\frac{(Ct)(Lk)}{(V)(S)}$ shs m/m^3km
01	T1	60	6	167.00	38.80	271.60	36.90
02	C1	20	8	162.80	30.00	210.00	15.50
03	T2	30	7	165.45	33.60	235.20	21.10
04	C2	30	8	168.75	33.00	231.00	21.90
05	T3	70	8	173.65	35.25	246.75	49.25
06	C3	30	8	144.50	21.00	147.00	29.50
07	T4	120	6	160.50	35.80	250.60	77.05
08	C4	30	8	122.25	10.00	70.00	52.40
09	T5	20	7	123.50	11.50	80.50	30.70
10	C5	30	7	126.00	13.00	91.00	41.55
11	T6	30	8	136.50	17.00	119.00	34.40
12	C6	20	7	141.50	21.00	147.00	19.25
13	T7	20	8	149.50	23.50	164.50	18.20
14	C7	20	6	143.50	25.00	175.00	16.40
15	T8	20	4	131.60	23.50	164.50	16.00
16	C8	40	5	132.65	21.00	147.00	36.10
17	T9	20	5	132.00	20.00	140.00	18.85
18	C9	20	7	137.40	19.00	133.00	20.65
19	T10	210	5	158.40	38.35	268.45	123.90
20	C10	20	6	127.25	16.00	112.00	[illegible]
21	T11	20	7	129.00	14.50	101.50	25.40
22	C11	30	4	118.40	13.00	91.00	39.05
23	T12	30	5	118.00	11.50	80.50	44.00
24	C12	30	5	116.75	10.00	70.00	50.05
25	T13	100	4	147.50	35.85	250.95	58.80
26	C13	60	6	121.85	10.00	70.00	104.45
27	T14	20	5	124.50	15.50	108.50	22.95
28	C14	20	4	128.40	21.00	147.00	17.45
29	T15	130	5	158.40	38.40	268.80	76.85
Total							1141.30

hauling cost for the 6 % road grade is:

$$\frac{\text{shs } 1139.50 \text{ m/m}^3\text{km}}{1300 \text{ m}} = \text{shs } 0.87/\text{m}^3\text{km}$$

while for the designed 10 % grade (Appendix 13) it is:

$$\frac{\text{shs } 1165.90 \text{ m/m}^3\text{km}}{1300 \text{ m}} = \text{shs } 0.90/\text{m}^3\text{km}$$

Recent studies carried out in Tanzania have reported log hauling costs for 7-8 t trucks of shs 0.40/m^{3}km (Saria 1977), shs 1.47/m^{3}km (Abeli and Dykstra 1981), shs 1.28/m^{3}km (Nyama 1985) and shs 0.85/m^{3}km (Kilango 1984). The range of costs reported reflects differences in vehicle type, road gradient and alignment, road standard, and the volume of logs hauled per load. However, the method of calculating costs used in this study provides estimates that appear to be fully satisfactory in the context of these reported costs.

3.7.2 Earthwork costs

During earthwork operations, three work elements were identified:

- cutting and filling soil on the embarkment of the same road segment
- sidecasting the excess cut material
- hauling soil from net cut areas to net fill areas.

To compute the actual earthwork costs (CE), costs incurred on these work elements were estimated by use of the following formulas (Dykstra 1982d):

$$CW = \sum_{i=1}^{M} (C)(Vii)(Dii)/(T)(Li) + \sum_{j=1}^{N} (C)(Vjj)(Djj)/(T)(Lj) \quad (20)$$

$$CF = \sum_{j=1}^{N} \sum_{i=1}^{M} (C)(Vij)(Dij)/(T)(Lj) \quad (21)$$

$$CS = \sum_{i=1}^{M} (C)(Vis)(Dis)/(T)(Li) \quad (22)$$

where: CW = the cost of excavating fill material in each road segment and compacting it on the embarkment of the same road segment, in shs/m^3km

CF = the cost of excavating fill material from the net cut areas and hauling it to the net fill areas, in shs/m^3km

CS = the cost of excavating and sidecasting the excess soil from the net cut areas, in shs/m^3km.

Individual elements in the equations are defined as:

M = total number of net cut areas

N = total number of net fill areas

i = an index of net cut area (i = 1, 2, ..., M)

j = an index of net fill area (j = 1, 2, ..., N)

C = the cost of excavating and hauling 1 m^3 of fill material for 1 m distance, in shs/m^3m

Vii = the volume of soil excavated on cut area i and compacted on the embarkment of the same cut area i, in m^3

Vjj = the volume of soil excavated in fill area j and compacted on the embarkment of the same fill area j, in m^3

Vij = the volume of soil excavated in cut area i and hauled to fill area j as fill material, in m^3

Vis = the volume of soil excavated in cut area i and sidecast as waste material, in m^3

Dii, Djj, Dij and Dis = as defined above, but instead of volume (V), in this case it is hauling distance, in m

T = the volume of timber scheduled to be hauled over this road during its lifetime, in m^3

Li and Lj = the length of cut area i and fill area j respectively, in km.

To estimate total earthwork cost, the above three earthwork cost elements are combined i.e., CE = CW + CF + CS. Total earthwork cost (CE) is expressed in shs/m^3km in order to permit direct comparison with hauling cost.

About 82 700 m^3 of logs are expected to be hauled over this road which will serve about 30 compartments of both *Pinus patula* and *Cupressus lusitanica* (Hamza 1983). The average earthmoving distance (within the same road segment) is taken to be half the segment's length while the average sidecast distance is estimated to be 5 metres.

Each of the three equations presented above requires knowing the cost of excavating and hauling a cubic metre of soil 1 m distance (C) in shs/m^3m. No such data were available prior to this study. As stated earlier, one objective of this study is to analyse construction costs and make available such data to the forest managers or any personnel engaged in forest road construction. Thus, to know the excavation cost per unit distance (C), it was necessary to carry out a time study of the earthwork operation. From the data collected during the earthwork operation, it was then possible to calculate the excavation cost per m^3 per 1 metre distance. Results of these calculations are given in Chapter 4.

Equations (20), (21) and (22) also require knowledge of the volume of soil excavated and filled on the embarkment of the same segment, volume of soil moved from net cut areas to net fill areas and sidecast volume. For each road grade design, these volumes were determined. Table 13 shows the volumes of soil excavated and the respective hauling distances to different areas for the designed 8 % road grade. Also shown in the last column is the product of: volume x hauling distances. Volumes of soil excavated and hauled to different sections of the road for 6 % and 10 % road grades were determined in the similar manner.

3.7.3 Optimal grade

An optimum road grade is one corresponding to the minimum total earthwork (CE) and hauling costs(CH). By summing equations (19), (20), (21) and (22) i.e., CH + CW + CF + CS, the road grade with the lowest total cost was considered to be an optimum road grade.

3.8 Field schedule

Table 14 shows a time schedule and the order in which different activities were undertaken. Altogether the project took about 1210 days or 3.3 years to accomplish. However, from Table 14, only 313 days or 25 % of this time was effectively spent on the road. Major reasons for such a short road length taking that long include:

- three lengthy delays; two due to breakdowns of the County tractor used for earthwork and one due to problems in

Table 15. Volumes of soil excavated and moved to different sections of the road for the 8 % grade design divided into 30 m working segments. Numbers shown in parenthesis are average hauling distance, in m.

Road segment	Peg No.	Hauled volume within the segment (m^3) V_{ii}, V_{jj}	Hauled volume from net cut to net fill areas (m^3) V_{ij}	Sidecast volume (m^3) V_{is}	Product of volume and hauling distance (m^3m)
1	0- 30	19.4 (15)		-	291.00
2	30- 60	51.4 (15)	-	-	771.00
3	60- 90	14.8 (15)	18.6(60) + 26.9(30)	101.9(5)	2 654.50
4	90- 120	-	-	216.0(5)	1 080.00
5	120- 150	-	-	217.4(5)	1 087.00
6	150- 180	-	-	223.7(5)	1 118.50
7	180- 210	-	-	152.4(5)	762.00
8	210- 240	-	-	174.8(5)	874.00
9	240- 270	-	-	203.5(5)	1 017.50
10	270- 300	29.5 (15)	74.5(30) + 15.5(60)	47.0(5)	3 842.50
11	300- 330	46.5 (15)	-	-	697.50
12	330- 360	14.8 (15)	-	-	222.00
13	360- 390	59.7 (15)	-	-	895.50
14	390- 420	-	96.0(60) + 9.9(40)	-	6 057.00
15	420- 450	3.0 (15)	-	99.9(5)	574.50
16	450- 480	-	-	98.1(5)	490.50
17	480- 510	-	-	80.1(5)	400.00
18	510- 540	-	3.7(30)	128.3(5)	752.50
19	540- 570	34.1 (15)	-	-	511.50
20	570- 600	9.9 (15)	-	-	148.50
21	600- 630	2.6 (15)	64.6(30)	-	1 977.00
22	630- 660	-	6.3(60)	84.5(5)	800.50
23	660- 690	5.3 (15)	13.0(30)	77.5(5)	857.00
24	690- 720	28.2 (15)	-	-	423.00
25	720- 750	27.0 (15)	13.9(30)	3.0(5)	837.00
26	750- 780	37.7 (15)	-	-	565.50
27	780- 810	27.9 (15)	9.2(30)	-	694.50
28	810- 840	34.9 (15)	-	-	523.50
29	840- 870	41.5 (15)	12.4(30)	14.0(5)	1 064.50
30	870 900	-	44.5(30)	53.1(5)	1 600.00
31	900- 930	28.9 (15)	-	-	433.50
32	930- 960	44.6 (15)	-	45.4(5)	896.00
33	960- 990	3.9 (15)	-	70.8(5)	412.50
34	990-1020	8.6 (15)	40.4(30) + 17.5(60)	-	2 391.00
35	1020-1050	17.5 (15)	-	-	262.50
36	1050-1080	1.1 (15)	-	-	16.50
37	1080-1110	106.3 (15)	-	-	1 594.50
38	1110-1140	-	181.3(60) + 41.0(30)	204.4(5)	13 130.00
39	1140-1170	11.9 (15)	-	143.0(5)	893.00
40	1170-1200	14.7 (15)	5.4(30)	17.9(5)	472.00
41	1200-1230	44.1 (15)		-	661.50
42	1230-1260	30.3 (15)	-	-	454.50
43	1260-1290	-	20.7(35)	43.8(5)	943.50
44	1290-1300	7.0 (5)	-	6.6(5)	68.00
					56 220.00 m^3m
Total		809 m^3	715.3 m^3	2507.1 m^3	4 031.4 m^3
%		20	18	62	100

Table 14. A summary of activities and their durations from 1981 to 1984.
Numbers shown below are work days.

Activity \ Year	1981												1982					1983								1984					Total days per activity
Month	J	F	M	A	M	J	J	A	S	O	N	D	J	1*	A	S	2*	J	F	M	3*	S	O	N	D	J	F	M	A	M	
Road survey		3																													3
Road alignment		26																													26
Clearing right of way				38																											38
Log skidding							45																								45
Removing stumps							33		14																						47
Moulding culverts												30																			30
Earth moving												12				6			19												37
Road grading																				4						3					7
Clearing the way to murram pit																						23				6	5				34
Opening and digging murram																							3			6	5				14
Hauling murram																										4	7				11
Installation of culverts																														21	21

Notes: 1* - Breakdown of the earthmoving tractor (January-August)
2* - Awaiting delivery of spares (October-January)
3* - Awaiting delivery of diesel and cement (March-September)

obtaining fuel for the construction machines and cement to install culverts (Table 14)

- shorter intervals, spaced throughout the project in which work was interrupted because of academic schedules and bad weather. For instance, during the rainy seasons, construction activities were suspended due to slippery and mud.

CHAPTER FOUR

RESULTS, DISCUSSION AND RECOMMENDATIONS

This chapter is divided into the following sections: routing and clearing the roadline, earthwork, gravelling and culvert installation. In each section the activities relating to the broad task considered in the section are analysed and discussed independently.

For each activity, three analysis are made: qualitative analysis, statistical analysis or the use of mathematical models or functions and time study analysis to estimate production rates and thirdly, economic analysis.

In this study, unless where stated, the cost figures used are June 1984 government controlled prices. For simple calculations, a hand calculator was used while statistical analysis, regression analysis and linear programming analysis were performed on a Hewlett Packard (HP 85) micro-computer at the Faculty of Forestry, Sokoine University of Agriculture, Morogoro.

4.1 Routing and clearing the roadline

Activities studied under this sub-heading include: road survey, road alignment, tree felling, log skidding and stump removal. All these activities were time studied by a shift level time study method. Table 15 gives a summary of the activities and the estimated road lengths in km accomplished in each activity.

Table 15. Summary of activities in routing and clearing of the roadline.

Activity	No. of days	Tools/equipment used	Total mandays	Total machine hours	Road length accomplished
Road survey	3	Clinometer, wire tape, ranging poles, bush knife	6	-	2.50 km
Road alignment	26	Levelling staff, compass, pegs and road survey tools	78	-	1.85 km
Felling trees	38	Cross-cut saws and axes	320	-	1.35 km
Skidding logs	45	Ford 6600 tractor plus skidding accessories	-	141.2	1.35 km
Digging and uprooting stumps	47	Hoes, axes and County 1164 tractor	226	67.7	1.30 km

Since a number of alternative routes were surveyed in order to locate the best route, a total of 2.50 km of roadline were surveyed. A total length of 1.85 km was aligned but due to time and economic constraints encountered later on, only 1.35 km of the aligned road was cleared and 1.30 km were prepared for final construction.

In order to provide access to the upper end of the SUA Training Forest, the new road was joined to an old track (which leads to the end of the forest) at peg no. 1300 (Figure 1). Thus, from the Rest House to peg no. 1300 or 1.3 km, the roadline was uprooted of all stumps, excavated, graded and later gravelled.

Production rates for activities like tree felling, log skidding and stump removal were derived from the daily records kept by the management of the Training Forest.

Five felling crews were employed to clear a 12-16 m wide "right of way". A day's task per felling crew was 14 trees while for the skidding crew it was 80 logs per day.

Before uprooting the stumps by use of the back hoe on the County tractor (Plate 1), most of the stumps were loosened by digging around them manually. A day's task for the 7 labourers engaged in digging the stumps was 13 stumps per day per person, while the number of stumps uprooted by the County varied from one day to another depending on the size of the stumps encountered, the terrain slope and the ground roughness. Data from a detailed time study conducted on 41 stumps uprooted by the County tractor and later analysed on a HP 85 microcomputer (Appendix 14) showed the following relationship between uprooting time and stump size:

$$y = -0.699 + 0.338X \qquad n = 41, \; R^2 = 0.177 \qquad (23)$$

where: y = uprooting time, in min

X = stump diameter, in cm.

When the null hypothesis, HO:b1 = 0, was tested using an F test at 0.01 probability level, it was concluded that a statistical relationship between stump diameter and uprooting time exist (Appendix 14). Figure

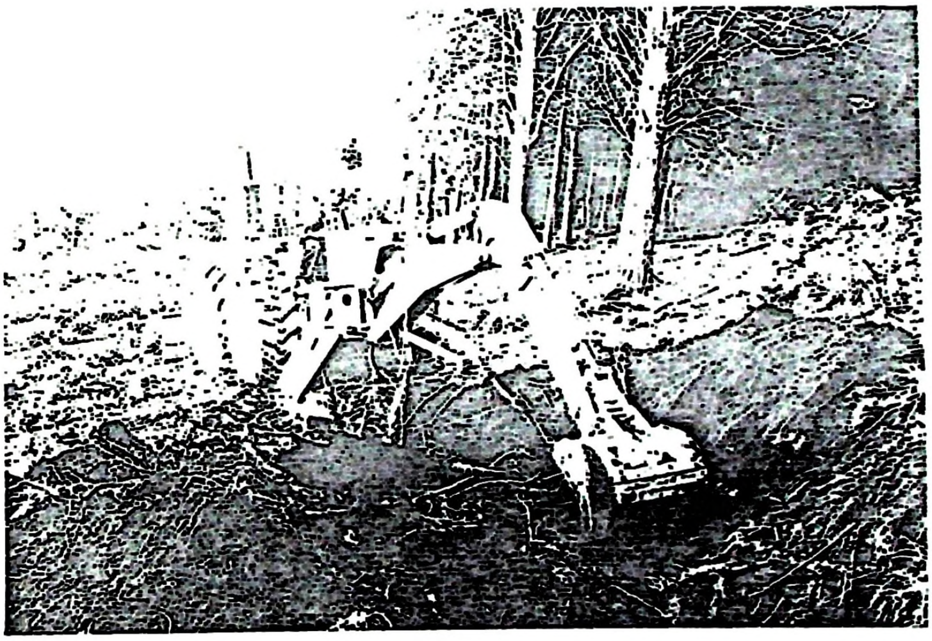


Plate 1. Uprooting stumps by use of a back hoe of the County 1164 tractor.

7 shows a linear relationship between uprooting time and stump size based on the data collected.

From the data analysis, it was observed that about 30 % of the uprooting time was spent moving from one stump to another and positioning the tractor at the right level position while the actual stumping took 70 % of the total time. Again, although it is recommended to use manual labour to uproot stumps when constructing forest roads, results of this study show that for large deep rooted stumps (as encountered here) the job becomes heavy and laborious.

In this study, the author personally surveyed, aligned and supervised all the subsequent operations which followed until the road was ready. In actual practice, such tasks could be undertaken by any forestry graduate. Thus, to be realistic, road survey, road alignment and supervision costs in this study are based on a monthly salary of a B.Sc. forestry graduate i.e. shs 2 200.00.

Total costs incurred for supervision, transport of workers to the forest and provision of lunches in the forest are summed up in this study as overhead costs. To all permanent workers, drivers and their helpers (unless where stated) 40 % fringe benefits have been added to their gross salaries in order to find the true labour costs.

Costs incurred on each activity are analysed and discussed in the following paragraphs.

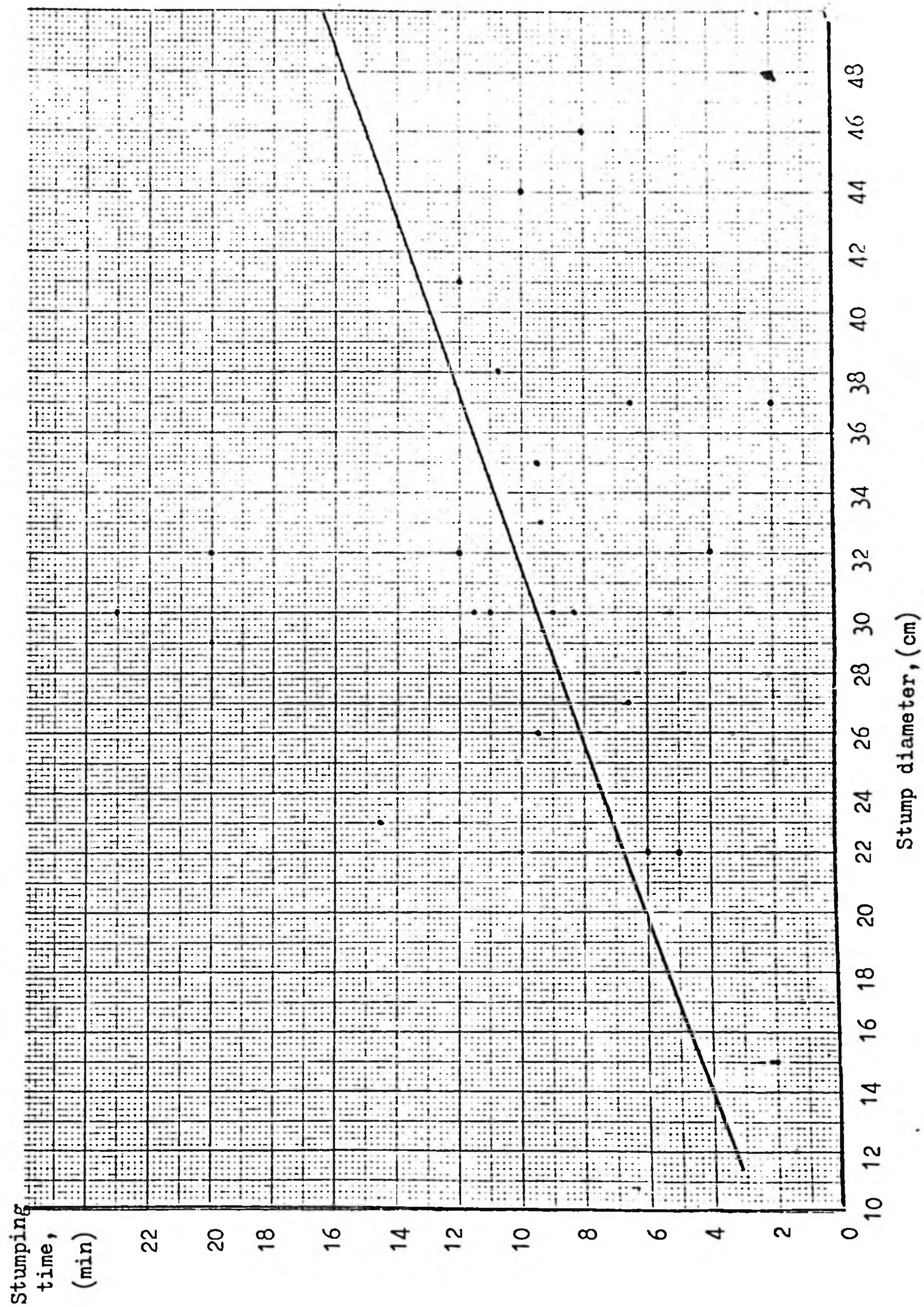


Figure 7. The relationship between uprooting time and stump diameter.

4.1.1 Road survey and alignment

In computing the cost of equipment used during survey, and road alignment activities, it was assumed that the number of days available for such work annually would be 150. Since a three-man crew did both activities, the total cost incurred on these activities is summarized below as follows:

Cost item	Total cost in shs
Equipment:	
Equipment valued at shs 2800 depreciated over 10 years with no salvage value.	
Interest rate 8 % and no insurance or taxes paid	
|({2800/10} + {(2800 + 280)/2}0.08)29 days|/150 days ...	77.95
Labour:	
Headman (see supervisor's cost under overhead costs)	
Helpers: shs 600/month x 1.40 x 2 x 29 days/25 days ...	1 948.80
Overhead:	
Supervisor: shs 2200/month x 1.40/30 days = shs 102.65/day	
Transport: shs 5.20/km x 30 km = shs 156.00/day	
Lunch:	
Maize flour: 1 kg @ shs 5.00 = shs 5.00	
Sugar: ½ kg @ shs 12.00 = " 6.00	
Cooking pot: Depreciation cost = " 1.50	
Labour (cook) @ shs 23.05 = " 23.05	
Sub-total for overhead costs: shs 294.20/day x 29 days ...	8 531.80
Total cost ...	10 558.55

4.1.2 Tree felling

About 1265 m^3 of standing trees were cleared from the right of way using 5 felling crews. The same felling gear used by Migunga and Dykstra (1983) was used in this study. Assuming that the felling gear is depreciated over 3 years and is used in cutting operations for a period of 200 days a year, the felling cost was computed as shown below:

Cost item		Total cost in shs
Depreciation and interest:		
|5({2100/3} + {(2100 + 700)/2}.08)38 days|/200 days	...	771.40
Maintenance and repair:		
5 x 100 % of depreciation	...	665.00
Labour:		
Foreman: shs 740/month x 1.40/25 days x 38 days	...	1 367.50
Labourers: shs 23.05/day x 320 mandays	...	7 376.00
Overhead :		
No transport costs since workers walked to the forest.		
Supervision undertaken by the Foreman		
Meals in the forest: shs 40.55/day x 38 days	...	1 540.90
Total cost	...	11 719.90

4.1.3 Log skidding

The cost of owning and running a Ford 6600 tractor is estimated to be shs 147.30 per hour (Appendix 9). Since a total of 141.2 machine hours (Table 15) were spent on skidding logs from the roadline, the total skidding cost is therefore:

shs 147.30/h x 141.2 h = shs 20 798.75

4.1.4 Stump removal

Assuming that the hoes and axes used during manual digging are depreciated within a year or 200 working days, the cost incurred on manual digging is:

Cost item		Total cost in shs
Depreciation:		
7 axes @ shs 117.00 x 33 days/200 days	....	135.15
7 hoes @ shs 60.00 x 33 days/200 days	...	69.30
Interest:		
Axes: $\left|\left\{\frac{(7 \times 117) + 135.15}{2}\right\} 0.08 \times 33 \text{ days}\right| / 200 \text{ days}$	..	6.30
Hoes: $\left|\left\{\frac{(7 \times 60) + 69.30}{2}\right\} 0.08 \times 33 \text{ days}\right| / 200 \text{ days}$	...	3.25
Maintenance:		
100 % of depreciation: shs 135.15 + shs 69.30	...	204.15
Labour:		
shs 23.05/day x 226 mandays	...	5 209.30
Total cost	...	5 627.45

The cost of owning and operating a County 1164 tractor is estimated to be shs 309.85 per hour (Appendix 9). Since a total of 67.7 machine hours were spent on uprooting stumps, it means therefore:

shs 309.85/h x 67.7 h = shs 20 976.85

were incurred on uprooting the stumps mechanically.

When the two stump costs i.e., manual and machine costs are summed up together, the total stump cost becomes:

shs 5627.45 + shs 20976.85 = shs 26 604.30

or an average of shs 35.50 per stump for a total of about 750 stumps uprooted along the roadline.

4.1.5 Cost summary

Activity		Total cost in shs
Road survey and alignment	...	10 558.55
Tree felling	...	11 719.90
Log skidding	...	20 798.75
Uprooting stumps	...	26 604.30
Total cost	...	69 681.50

From the above cost summary, the cost of surveying, aligning and clearing the roadline ready for excavation is shs 69 681.50 or shs 53.60 per running metre. Also it is clear that those activities which used machines were relatively more expensive than those performed manually. For example, the County tractor consumed about 80 % of the stump costs as opposed to only 20 % spent on manual labour.

Road survey and road alignment which normally must be carried out by competent road engineers accounted for only 15 % of the total routing and clearing costs.

4.2 Earthworks

Under earthwork operation, the following work activities are analysed and discussed separately: earth moving (cutting and filling) and road grading.

4.2.1 Earth moving

As stated in Sections 3.4.3.1 and 3.4.3.2, for the three road grade alternatives, mathematical computations were carried out in an effort to determine the minimum-cost road grade to construct. Optimal solutions when using the transportation tableau method and micro-computer based linear programming method were found to be the same despite their basic variables varying. Appendix 15(a)-(h) show the optimal earthwork allocations by means of transportation tableau method for the different road designs analysed in this study. A summary of these 8 road design alternatives plus the designed and constructed 8 % (30 m road segments) is presented in Table 16.

Due to the lack of excavation cost data at the onset of this study, distance was used as a surrogate for transportation cost in computing the optimal solutions. Among the 9 solutions presented here, the road grade alternative with the minimum objective-function value, or minimum average moving distance, was picked for construction. As is evident in Table 16, this road grade alternative is the one with a maximum grade of 8 % and 30 m working segments.

Table 16. The optimal solutions for the 9 construction alternatives analysed in this study.

Grade alternative (%)	Length of road segment (m)	Total net supply volume (m^3)	Total net fill volume (m^3)	Optimal solution Min Z-value (m^3m)	Average distance per m^3 (m)
6	30	24 797	2 184	197 322	90
	60	24 738	1 340	197 739	148
	Terrain	24 896	1 494	192 498	129
8	30	3 221	715	31 515	44
	60	3 081	584	34 761	60
	Terrain	3 260	825	38 322	46
10	30	2 866	2 796	570 322	204
	60	2 558	2 524	660 558	262
	Terrain	2 828	2 665	608 252	228

Later in this Section, (after knowing the excavation cost per unit distance {C}), the optimal road grade which minimizes both earthwork and hauling costs is determined.

Of the two methods; i.e., the transportation tableau and the micro-computer based linear programming method, which one to use will depend on the availability of a computer and the size of the problem. Where computers are not available (like in forest projects) and the length of the road to be constructed is short with few alternative designs, the use of the transportation tableau method has proven to be fully satisfactory. On the other hand, where a computer is accessible and the road to be constructed is long with many alternative design constructions, the use of a micro-computer based linear programming method is advocated in order to cut down computation time and human errors.

Two earth moving machines were used for cutting and filling the road-line. These machine were: a County 1164 four-wheel tractor with a back hoe and a front dozer plate (Plate 2) and a D4D Caterpillar track type tractor. The former did cutting, filling and sidecasting on the first 520 m of the roadline while the rest of the roadline was done by a D4 bulldozer. Plate 2 shows a County tractor moving excess cut material from a net cut area to a net fill area. Production rates for the two machines are summarized on Tables 17A and 17B

Because of the topographic nature of the area on which this road is constructed (Figure 1), most of the roadline required cutting rather than filling. Cutting in some sections was as deep as 3 metres and most of the excavated material was sidecast (62 %). Less than one-fifth (18 %) of the excavated material was hauled to areas requiring fill materials (Table 13) while the remaining 20 % of excavated material was compacted into the fill portion of the cross-section from which it was taken.

The average production rate for the County tractor was 30.0 m^3/h while that of the D4 was 43.3 m^3/h (Table 17). Since these two machines worked on two different road sections and were operated by different drivers, making productivity comparisons between the two machines is inappropriate. However, when compared with studies in other countries

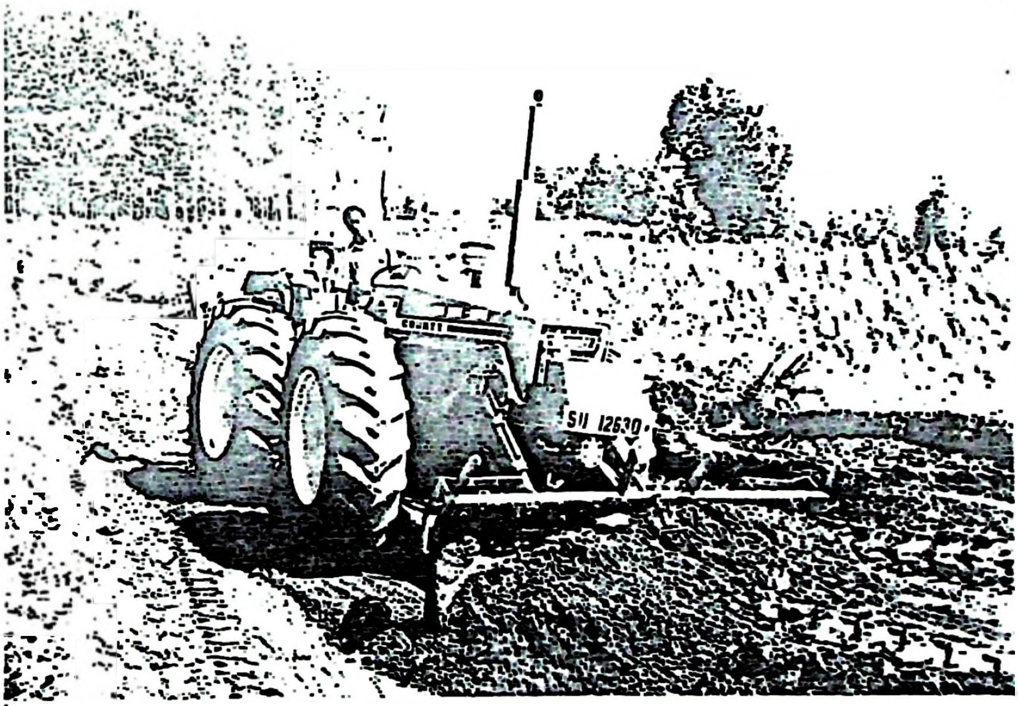


Plate 2. Excavating and moving soil by use of a dozer plate of the County 1164 tractor.

Table 17A Production rates when using a County tractor for cutting and filling the roadline.

Day No.	Total work place time (h)	Effective working time (h)	Necessary delay + stumping time (h)	Lunch break (h)	Cut volume (m^3)	Fill volume (m^3)	Length of road worked per day (m)	Volume cut/filled per hour (m^3/h)	Distance worked per hour (m/h)
1	5.85	4.08	1.20	0.57	240.0	-	100	[illegible]	[illegible]
2	5.28	4.58	0.17	0.53	103.0	-	70	[illegible]	15.0
3	6.57	6.05	-	0.52	104.0	-	40	17.2	6.6
4	6.37	5.70	-	0.67	68.3	-	40	11.9	7.0
5	5.04	4.53	-	0.51	159.8	-	90	35.3	19.9
6	6.71	4.00	1.03	1.68	109.1	-	50	27.3	12.5
7	5.18	4.55	-	0.63	127.8	-	70	28.1	15.4
8	5.90	5.27	-	0.63	157.4	-	50	29.9	9.5
9	5.89	5.32	-	0.57	65.6	-	40	12.3	7.5
10	6.19	4.48	1.03	0.68	112.6	-	40	25.1	8.9
11	6.15	5.43	-	0.72	114.9	-	70	21.2	12.9
12	5.63	4.73	-	0.90	124.9	29.8	140	32.7	29.6
13	5.42	4.67	-	0.75	114.0	113.9	100	48.8	21.4
14	6.00	5.08	-	0.92	152.0	-	100	30.0	19.7
15	4.17	3.67	-	0.50	110.0	-	50	30.0	13.6
16	4.57	3.57	-	1.00	74.1	42.2	50	32.6	14.0
17	4.33	3.50	-	0.83	84.2	74.9	60	45.5	17.1
18	3.41	2.83	-	0.58	77.4	54.9	100	46.7	31.8
19	4.15	2.42	0.86	0.87	29.8	28.8	70	23.9	20.5
20	5.38	3.55	1.10	0.73	94.6	19.6	90	32.2	25.4
21	5.50	4.64	0.18	0.68	82.4	40.3	80	30.6	17.2
22	4.17	3.40	-	0.77	73.4	33.9	80	31.5	23.6
23	4.27	3.57	0.03	0.67	73.0	90.44	50	45.8	14.0
24	8.13	6.74	0.57	0.82	163.4	33.6	90	29.0	13.0
25	7.56	5.94	0.20	1.42	160.6	3.0	100	27.5	16.8
26	8.90	6.30	1.30	1.30	190.8	12.4	100	32.3	15.9
Total	146.72	118.6	7.67	20.45	3544.79		1920	-	-
Average %	100 %	81 %	5 %	14 %	-	-	-	30.0	16.2

Table 17B. Production rates when using a D4 Caterpillar for cutting and filling.

Day No.	Total work place time (h)	Effective excavation time (h)	Necessary delay (h)	Scheduled break + travelling time (h)	Unnecessary delay (h)	Length of road worked per day (m)	Cut volume/day (m^3)	Fill volume/day (m^3)	Average cut/fill volume/hour (m^3/h)	Length of road worked per hour (m/h)
1	3.98	3.31	0.17	0.50	-	90	93.2	93.3	56.3	27.2
2	9.08	8.38	-	0.50	2.0	150	248.9	105.1	43.4	17.9
3	8.97	7.39	1.08	0.50	-	150	166.5	138.6	41.3	20.3
4	8.75	7.65	0.60	0.50	-	180	308.5	176.7	63.4	23.5
5	9.15	6.87	0.78	0.50	1.0	140	199.0	173.7	54.3	20.4
6	3.65	2.84	0.33	0.50	-	130	44.9	46.5	32.2	45.0
7	8.00	6.80	0.60	0.60	-	80	199.9	99.6	44.0	11.8
8	9.97	8.83	0.53	0.60	-	120	140.2	168.8	35.0	13.6
9	8.92	7.78	-	1.14	-	180	229.3	144.5	48.2	23.0
10	9.68	8.54	-	0.60	0.53	80	208.9	-	24.5	10.5
11	8.47	7.37	0.50	0.60	-	170	192.5	113.1	41.5	23.1
Total	88.62	75.76	4.59	6.54	1.73	1470	3291.7		-	-
Average %	100 %	86 %	5 %	7 %	2 %				43.3	19.4

using a D4 tractor, the production rates observed in this study are slightly higher for earthwork volume production but somewhat less for distance production. Table 18 shows earthwork production rates in different parts of the world when using a D4 tractor for forming the road body.

Table 18. Earth moving production rates in different parts of the world.

Country	Type of machine	Production rates	
		Distance (m/h)	Volume (m^3/h)
Tanzania (this study)	County 1164 tractor	16	30.0
	D4 Caterpillar	19	43.3
Philippines (Deepak 1978)	D4 Caterpillar	25	25.0
	D4 Caterpillar	50	15.0
Iran (Edmonds and Howe 1980)	D4 Caterpillar	20	27.5
U.S.A. (Caterpillar 1981)	D4 Caterpillar	30	150.0
	D4 Caterpillar	45	100.0

The production figures given by Caterpillar Company (Cat. 1981) are comparatively high since they are based on ideal working conditions; i.e., the machine is in good working order, the driver or operator is skilled and the working terrain is considered to be favourable.

Although further studies on the use of the County tractor for road building ought to be carried out before conclusions are drawn, from the data collected in this study, it is clear that the County tractor can be a satisfactory substitute for the D4, especially in those forest projects unable to buy separate skidding and road building tractors.

Unavailability of foreign currency to buy construction machines, some government policies of encouraging labour intensive operations and sometimes easy terrain to work with, lead to construction of roads manually. Where roads have been constructed manually, production rates per manday have ranged from 2.8 m^3/h in Afghanistan, 3.0 m^3/h in Mexico to 3.3 m^3/h in Philippines (Deepak 1978, Edmonds and Howe 1980). Whether it would have been more economical to employ 11 or 15 people to substitute the County tractor and a D4 Caterpillar in this project respectively, it is a question which requires further investigation.

The use of manual labour instead of the above machines would definately require more supervision, more construction days and consequently more overhead costs. On the other hand, employment of manual labour would not only cut down on the dependence of foreign currency but would also create rural employment and minimize delays caused by machine breakdowns. For instance, when the County tractor broke down (Table 14), it resulted to 100 % delay of the operation. Alternatively, if 11 labourers were employed to dig the road manually, one person falling sick would have resulted to a delay of only 9 % of the total construction time.

The cost of owning and running a County 1164 tractor is estimated to be shs 309.85 per hour (Appendix 9). If the total machine hours spent on digging the first 520 m of the roadline is 126.30 h (Table 17A), the total cost incurred on this section of the road is computed as:

Cost item		Total cost in shs
Machine costs:		
shs 309.85/h x 126.30 h	...	39 124.75
Overhead costs:		
shs 294.20/day x 26 days	...	7 649.20
Total cost	...	46 773.95

From Table 17B, the D4 spent 88.62 h digging the rest of the roadline (780 m). The estimated cost of owning and running a D4 as summarized in Appendix 11 is shs 332.65 per hour. Therefore, the total cost incurred on the rest of the roadline is:

Cost item		Total cost in shs
Machine costs:		
shs 332.65/h x 88.62 h	...	29 479.45
Overhead costs:		
shs 294.20/day x 11 days	...	3 236.20
Total cost	...	32 715.65

Given the above cost figures, the cost of forming the road body by the County tractor is shs 89.95/m and shs 41.95/m in the case of the D4. These costs are low compared to the quotations given by two road contractors approached in 1978 to construct similar access roads in the Training Forest. The cost figures given by these contractors were shs 130.00/m in case of Mwananchi Engineering and Construction Company

(MECCO) and shs 125.00 by the Zangie Construction Company. In the Meru Forest Project, the cost of forming the road body of a similar road standard is estimated to be shs 16.60/m (Chilembu 1984).

To be able to make comparisons on the excavation costs, the volume of soil cut and filled within a given road section has to be considered. This enables one to have a standard cost measure which is expressed as the cost of cutting and hauling 1 m^3 of soil for 1 km distance; i.e., shs/m^3km.

From Table 17A, the total volume of soil excavated (cut) and hauled to either net fill areas or sidecast by the County tractor is 3544.8 m^3. Since the length of road treated by the County is 520 m, the excavation cost is estimated as:

$$\text{shs } 46773.95/\{(3544.8 \text{ m}^3)(0.52 \text{ km})\} = \underline{\text{shs } 25.35/\text{m}^3\text{km}}$$

For the rest of the roadline, the volume excavated and hauled by the D4 is 3291.7 m^3 (Table 17B). Excavation cost is therefore:

$$\text{shs } 32715.65/\{(3291.7 \text{ m}^3)(0.78 \text{ km})\} = \underline{\text{shs } 12.75/\text{m}^3\text{km}}$$

To be able to estimate excavation costs, the length of the road and the volume of soil to be excavated and hauled within this road length have to be known. For example, where County 1164 tractor and D4 machines are to be used for forming the road body in conditions similar to those encountered in this study, the excavation costs can be estimated from the following equations:

County 1164 tractor: $TC = 25.35(a)(b)$ (24)

D4 tractor: $TC = 12.75(a)(b)$ (25)

where: TC = total cost of forming the road body, in shs

25.35 and 12.75 = constants; i.e., the cost of excavating 1 m^3 of soil and hauling it 1 km distance using the County or D4 tractor, respectively

a = total volume of soil to be excavated and hauled, in m^3

b = the length of the road to be excavated and filled, in km.

As stated in Section 3.7.2, in order to be able to compute total earthwork costs (CE), the cost of excavating and hauling 1 m^3 of soil over a 1 m distance (C) has to be known. From the findings of this study, the cost of excavating and hauling 1 m^3 of soil per 1 m distance (C) is shs 0.025 and shs 0.013 for the County 1164 tractor and D4 respectively. Since the excavation cost for the County tractor is higher than for the D4, to be on the conservative side its cost has been used here to estimate total earthwork costs.

For equations (20), (21) and (22) the following variables are so far known:

- the volume of soil to be excavated and hauled to different sections of the road (Vii, Vjj, Vij and Vis) plus the product of volume x distance per road section (Table 13)
- excavation and hauling cost per m^3 per 1 m distance (C = shs $0.025/m^3m$)
- the average hauling distance per road segment (L) (Section 3.7.2)

- the total volume of logs to go over this road (T = 32 700 m^3) (Section 3.7.2).

Table 13 gives the values required by the above three equations. Knowing these values, total earthwork cost (CE) for the designed 8 % road grade divided into 30 m working segments is computed as follows:

$$CE = \frac{(0.025 shs/m^3m)(56220.00\ m^3m)}{(32700 m^3)(0.03 km)}$$

$$CE = shs\ 0.57/m^3km$$

Total earthwork costs for 60 m, "terrain" interval lengths plus the other two road grade designs (6 % and 10 %) were computed in the similar manner. Total earthwork costs for the three grade designs plus their hauling costs (as computed in Section 3.7.1.8) are summarized in Table 19.

Table 19. Total earthwork and hauling costs for the three grade designs.

Road grade design (%)	Interval length	Earthwork costs (shs/m^3km)	Hauling cost (shs/m^3km)	Total earthwork and hauling cost (shs/m^3km)
6	30 m	3.40	0.87	4.27
	60 m	1.64	0.87	2.51
	Terrain	1.62	0.87	2.49
8	30 m	0.57	0.88	1.45
	60 m	0.39	0.88	1.27
	Terrain	0.33	0.88	1.21
10	30 m	5.87	0.90	6.77
	60 m	3.77	0.90	4.67
	Terrain	3.27	0.90	4.17

From Table 19, the design grade with minimum total of earthwork and hauling costs is the designed 8 % grade. This is the same road grade design recommended for construction when earthwork volume x distance was used as a criteria for determining the optimal road grade (Section 3.4.3 and Table 16). Results from Table 19 also indicate that 60 m and terrain breaks apparently appear to be too wide to permit accurate estimates of earthwork costs. Terrain breaks in particular provide erratic estimates (the estimates for all the three designs using terrain breaks are less than those using 60 m and 30 m segments). When 30 m and 60 m interval lengths are compared, the latter (because of wide length and less accurate earthwork volume estimation) results to less accurate earthwork cost. This suggests that the common practice of using 100 ft (30 m) segment lengths in U.S.A. (Lecklinder and Lund 1971) is based on the observation that longer segment lengths provide poor design estimates.

4.2.2 Road grading

A motor grader (Caterpillar 120G) with a net power of 93 kW was used for grading the road (Plate 3). By tilting the blade, the grader was able to make side ditches, side slopes and back slopes also. The side ditches made were 30 cm deep and "V" shaped while backslope angles were 10:1. On tangent sections of the road, the road was crowned while on curves, it was superelevated. The slope angles for both camber and super-elevation were about 8 % which is higher than what is normally recommended for forest roads (Dykstra 1983b, Lundquist 1979, FAO 1977, Skaar 1972a). The relatively steep camber was judged

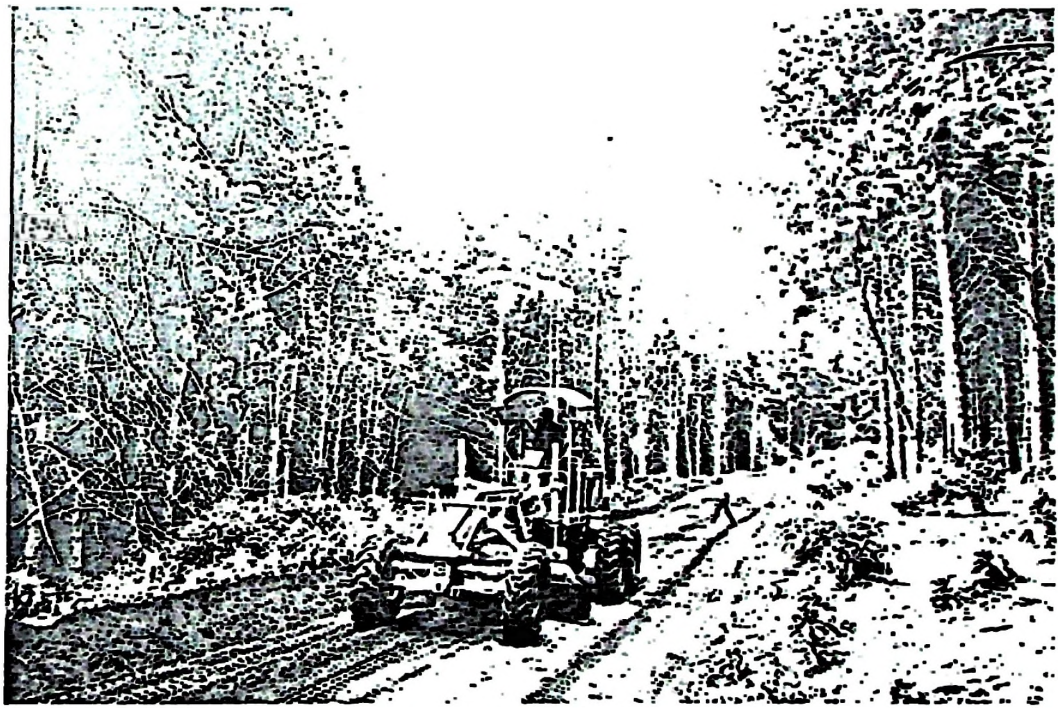


Plate 3. Motor grader grading the road.

necessary in order to ensure adequate drainage during the rainy season.

Table 20 summarizes production rates measured during the grading operation.

Table 20. Road grading production.

Day	Grading time (h)/day	Graded distance (m)/day	Volume cut and hauled (m^3)/day	Volume cut and hauled (m^3)/h
1	7.15	240	299.8	40
2	7.50	520	362.0	48
3	8.33	360	361.6	43
4	4.18	180	112.7	27
Total	27.16h	1300m	1136.1m^3	40 (Average)

Although in Table 14 it is indicated that the road was graded twice, only one of the two undertaken gradings is considered here when computing grading costs. The second grading was necessary in this case because of lengthy delays between the first grading and the completion of the road. Normally, however, only one grading would be required.

From Appendix 11, the estimated cost of owning and running a motor grader is shs 530.55 per hour. Total machine hours spent on grading the road were 27.16 h (Table 20). Therefore, the total estimated cost is :

Cost item		Total cost in shhs
Grader machine:		
shs 530.55/h x 27.16 h	...	14 409.75
Overhead:		
shs 294.20/day x 4 days	...	1 776.80
Total grading cost	...	16 186.55

The estimated cost of grading the road was shs 12.45 per running metre while according to Chilembu (1984), the cost of grading access roads in the Meru Forest plantation is shs 2.75 per metre (overhead costs inclusive). The low grading cost in the Meru Forest is probably due to under-estimation of the machine and overhead costs. For example, although Vestman (1982) estimated that it costs shs 450.00 to own and run a motor grader for one hour (as compared to shs 530.55 found in this study) when computing the grading costs (using the same motor grader), instead of using this cost figure (shs 450/h), Chilembu (1984) used only a part of it; i.e., the direct operating cost, which is shs 250/h. Estimated overhead costs by Chilembu (1984) account for about 10 % of the machine running costs.

4.3 Road gravelling or murraming

To provide fine gravel (murram) as surfacing material for the road, three murram pits located along the river banks were identified within a radius of 3.5 km from the road (Figure 1). Samples analysed from these pits show that the percentage of clay and silt in two of the pits is less than 10 %, an indication that the murram would be suitable as road surface material (Skaar 1972a). Table 21 shows the particle size distribution of the samples collected from the three murram pits.

Table 21. Particle size distribution as analysed from the samples collected from the three murram pits.

Murram pit No.	Particle size distribution (%)		
	Gravel (≥ 2.00 mm)	Sand (0.106-2.00 mm)	Clay + Silt (≤ 0.106 mm)
1	35.3	50.4	14.3
2	57.0	34.0	9.0
3	57.0	38.4	4.6

Murram pit number 3 (Figure 1) was selected as the murram source for this road, for the following reasons:

- To reach the new road to be murramed from pit No. 1 and No. 2, the loaded trucks would have to travel on an adverse grade. In case of pit No. 3, the loaded trucks would be travelling on a favourable grade except near the bridge (Figure 1).

- Pit No. 1 and 2 have once been dug and the remaining murram is rather deep and consequently difficult and expensive to extract.

- Pit No. 1 and 2 are close to the villagers who have once complained about water pollution when one of the pits was being used.

- Improving the tracks leading to pits No. 1 and 2 would not benefit the Training Forest much since the tracks and the pits are outside the Training Forest.
- The high percentage of silt and clay in pit No. 1 makes its murram only marginally suitable as a road surface material.
- The average round trip distance from pit No. 1 is 10 km and 8 km from pit No. 2 as compared to only 4.9 km for pit No. 3.

4.3.1 Improvement of the track

To reach pit No. 3, which is on the upper end of the Training Forest, the track leading to the pit had to be improved and re-aligned, and a wooden bridge had to be constructed to cross one of the streams. Since the improved track and the bridge are within the Training Forest and in addition make the upper end of the forest accessible, the costs incurred on improving the track and the bridge are considered to be well justified.

The County 1164 tractor and a Caterpillar D6 tractor were used in improving the 2 km track leading to the murram pit. Manual labour was employed to clear the road and construct a 50 m corduroy on both ends of the bridge. On this section (whose water table was high) several loads of murram were dumped by use of Isuzu trucks.

Given the cost of owning and running the County tractor per hour to be shs 309.85/h (Appendix 9), a D6 as shs 673.30/h (Appendix 11) and an Isuzu truck as shs 8.60/km (Appendix 10), the total cost incurred in improving this track is estimated as follows:

Cost item		Total cost in shhs
County tractor:		
shs 309.85/h x 29.0 h	...	8 985.65
D6 tractor:		
shs 673.30/h x 23.0 h	...	15 485.90
Isuzu trucks:		
shs 8.60/km x 34 km	...	292.40
Manual labour:		
shs 23.05/day x 49 mandays	...	1 129.45
Watchman:		
shs 23.05/day x 13 mandays	...	299.65
Overhead:		
shs 294.20/day x 11 days	...	3 236.20
Total	...	29 429.25

4.3.2 Construction of the wooden bridge

As stated above, because of the stream crossing, a wooden bridge had to be built. Activities, materials and costs involved during the construction of this bridge are summarized below:

Cost item		Total cost in shs
Felling and bucking *Eucalyptus* trees:		
shs 23.05/day x 24 mandays	...	553.20
Log skidding by Ford 6600 tractor:		
shs 147.30/h x 6.0 h	...	883.80
Log transport to the bridge site by tractor trailer:		
shs 147.30/h x 9.0 h	...	1 325.70
Digging and decking using County tractor:		
shs 309.85/h x 12.0 h	...	3 718.20
Construction - manual labour:		
shs 23.05/day x 121 mandays	...	2 789.05
Watchmen:		
shs 23.05/day x 32 mandays	...	737.60
Nails: shs 35.00/kg x 24 kg	...	840.00
SIDO clips: shs 30.00/piece x 20 clips	...	600.00
Log value: shs 50.00/m^3 x 21 m^3	...	1 050.00
Overhead: shs 294.20/day x 24 days	...	7 060.80
Total	...	19 558.35

Although the track and bridge were improved and constructed for the purpose of hauling murram, only part of these costs should be allocated to the new road since the track and bridge will be used for other purposes after the murraming operation. For the purpose of cost allocation, it was assumed in this study that 20 % of the bridge- and track-improvement costs were undertaken solely to benefit the new road. Therefore, the new road is allocated 20 % of the total track and bridge costs; i.e.,

$$\text{shs } 29429.25 \times 20\ \% + \text{shs } 19558.35 \times 20\ \% = \underline{\text{shs } 9797.50}$$

4.3.3 Clearing and digging of murram

A total area of about 1200 m^2 around the murram pit was manually cleared of all trees, shrubs and other vegetation. The D6 was used to open the pit by pushing aside the top soil before employing a Caterpillar 215 excavator for digging the murram. An estimated 935 m^3 of soil was pushed aside to permit the excavation of about 700 m^3 of murram.

Using the estimated costs of owning and running the two Caterpillar machines used in opening the pit and digging the murram as summarized in Appendix 11, the total cost of clearing, opening and digging murram is summarized as:

Cost item			Total cost in shs
Manual labour:	shs 23.05/day x 24 mandays	...	553.00
D6 bulldozer:	shs 673.30/h x 9.0 h	...	6 059.70
215 excavator:	shs 692.60/h x 48.0 h	...	33 244.80
Watchmen:	shs 23.05/day x 38 mandays	...	875.90
Overhead:	shs 294.20/day x 14 days	...	4 118.80
Total		...	44 852.20

4.3.4 Hauling of murram

Murram was hauled from the murram pit to the road site by use of 7 t Isuzu tipper trucks. Loading of murram onto these trucks was done by a County tractor with a back hoe and the 215 excavator. Plate 4 shows one of the hauling trucks being loaded by the 215 excavator.

The loading capacities of the County tractor and the 215 excavator were 0.20 m^3 and 0.87 m^3 per bucket, respectively. From the time study carried out, it was found that to load one truck load (2.6 m^3), it took the County tractor about 7 minutes while the excavator took only 4 minutes.

Although the trucks could carry about 3.5 m^3 of murram per trip, only 2.6-3.0 m^3 of murram was loaded and hauled per trip due to an adverse grade (+ 13 %) encountered near the bridge and the low bearing capacity of the dirt track, both at the murram pit and at the bridge site.

Based on detailed time studies carried out during the hauling operation, daily summaries on the total loading time, total distance travelled, total volume hauled and the length of the road murramed were recorded as shown in Table 22.

Unlike the existing access roads in the SUA Training Forest where the whole roadway (4.5 m) has been murramed, in this study road only the carriage way (3.0 m) was murramed. In order to minimize the loss of murram due to traffic and to regulate the rate of water flow from the road, both sides of the road were sown with creeping grass known as Kikuyu grass.

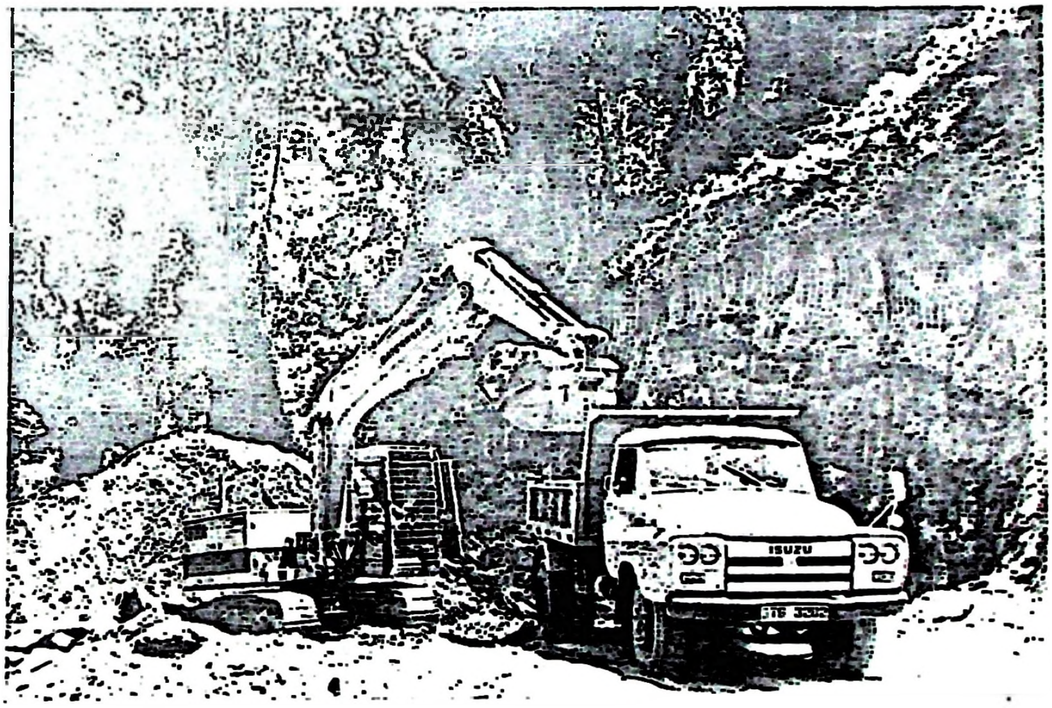


Plate 4. A Caterpillar 215 excavator loading murram into a 7 t Isuzu tipper.

Table 22. Daily summaries during murram hauling operation.

Day No.	Type of loader	No. of trucks per day	Loading time (h)	Distance from or to station (km)	Murram hauling distance (km)	No. of trips/ day	Volume hauled/ day (m^3)	Length of road murramed (m)	Thickness of murram layer (cm)
1	County	2	4	34	1.0	8	27.5	50	13.5
2	County	1	4	18	21.0	11	30.4	75	10.1
3	County	2	3	18	48.0	12	25.0	70	9.0
4	County	2	2	-(b)	25.0	4	8.0	15	13.3
5(a)	-	2	-	34	-	-	-	-	-
6	Excavator	2	3	34	70.0	16	43.6	100	14.5
7	Excavator	2	4	-(b)	111.0	25	67.0	200	11.2
8	Excavator	2	2	-(b)	67.4	14	37.6	150	8.4
9	Excavator	2	4	-(b)	92.0	18	47.6	150	10.6
10	Excavator	2	3	-(b)	202.4	32	96.0	290	11.0
11	Excavator	2	2	-(b)	121.3	19	57.0	150	12.7
12	Excavator	2	3	34	38.4	6	18.0	50	12.0
Totals	-	-	34	172	797.5	165	457.7	1300	11.5 (Average)

Notes: (a) Operations had to be suspended at the beginning of day 5 because of heavy rains.

(b) Trucks were kept overnight in the forest-so travel to the murram pit was not required.

During the hauling operation, despite two trucks hauling murram to the road site, only one truck was studied per day. One round trip journey formed one work cycle i.e., travelling loaded, unloading, travelling empty and loading. Data collected from 86 round trips studied (Appendix 16) were subjected to statistical and regression analysis using HP 85 micro-computer statistics and regression programs. Variables measured and used in the analysis were as follows:

DISTKM = distance from the murram pit to the road site, in km

LOADGt = loading time, in min

TRAVLD = travelling loaded time, in min

TRAVEM = travelling empty time, in min

UNLODt = unloading time, in min

LOADcM = volume of murram loaded per trip, in m^3

TOTALt = total round trip time, in min.

Table 23 summarizes the statistics for the murram hauling operation.

Table 23. Summary statistics for the murram hauling operation.

Statistic / Variable	Mean	95 % C.I on mean	Standard deviation	Minimum	Maximum
DISTKM	2.47	2.39- 2.54	0.34	2.0	3.2
LOADGt	3.96	3.75- 4.17	0.98	2.5	10.7
TRAVLD	8.50	8.20- 8.82	1.44	6.5	11.5
TRAVEM	7.82	7.53- 8.11	1.34	6.0	10.8
UNLODt	3.12	2.56- 3.69	2.62	1.0	16.0
LOADcM	2.67	2.64- 2.71	0.16	2.6	3.0
TOTALt	23.42	22.58-24.26	3.92	18.0	39.9

To estimate total round trip time (TOTALt), regression equations based on one or two independent variables were developed. For each equation, a null hypothesis relating TOTALt to the independent variable(s) expected to influence it was stated.

Using DISTKM and LOADcM as independent variables to estimate TOTALt, two regression equations were developed as shown below.

Regression equation 1

HO : TOTALt = f(DISTKM)

Appendix 17 summarizes the computer output for this problem. From the computer print-out, a regression equation which estimates round trip time is developed as:

$$\text{TOTALt} = 6.64 + 6.80\ \text{DISTKM} \qquad (26)$$

$$R^2 = 0.34,\ n = 86$$

The low value of R^2 suggests that only a small fraction of the variation in TOTALt is explained by variations in DISTKM. However, the co-efficient of DISTKM is found to be statistically significant with an error probability less than 1 per cent (Appendix 17). When DISTKM and TOTALt are plotted on a graph, a linear relationship is observed (Fig. 8).

Regression equation 2

HO : TOTALt = f(DISTKM, LOADcM)

From the computer print-out (Appendix 17), the regression equation

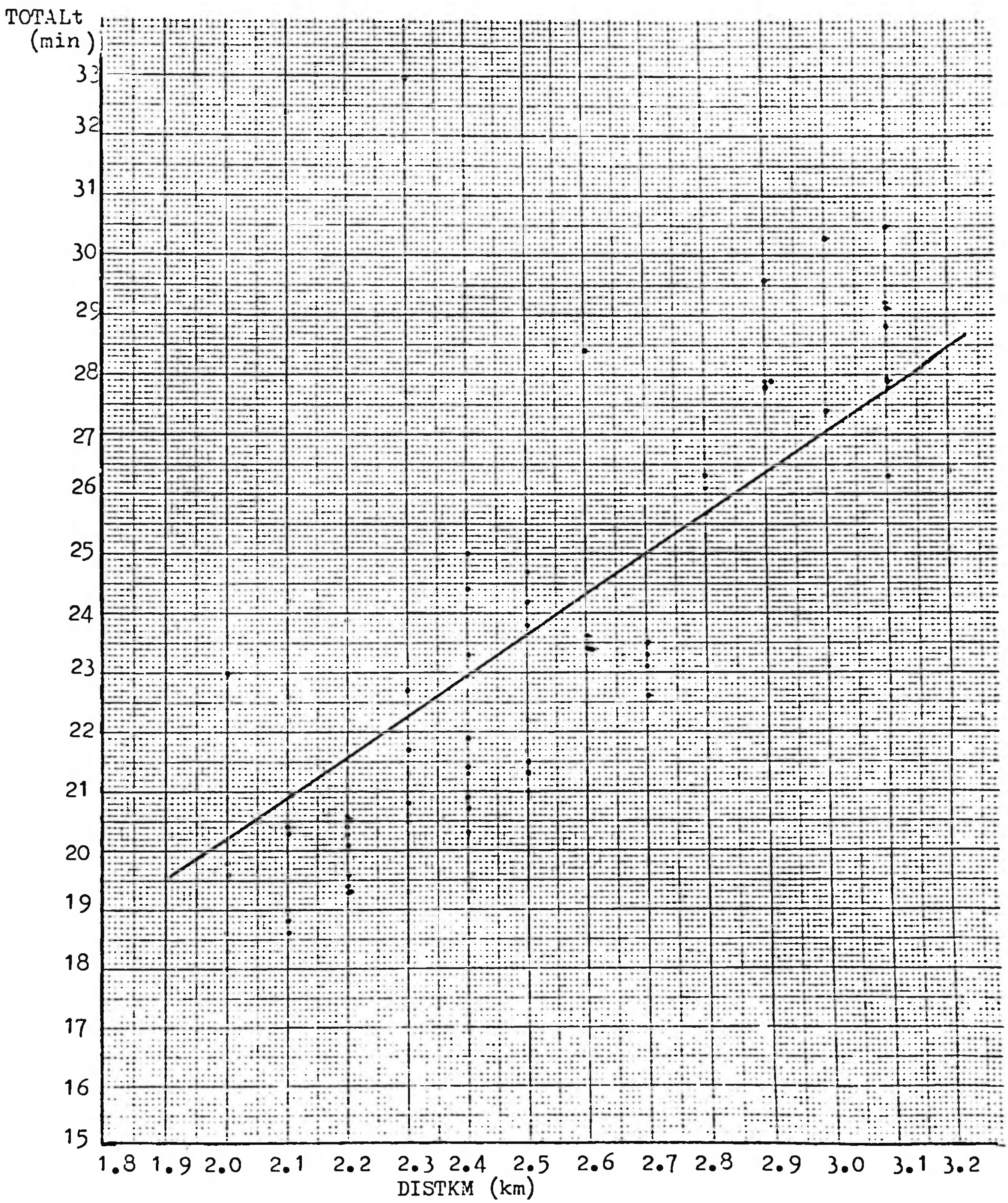


Figure 8. The relationship between DISTKM and TOTALt.

estimating TOTALt is developed as:

$$TOTALt = -8.61 + 4.06\ DISTKM + 8.23\ LOADcM \qquad (27)$$

$$R^2 = 0.39,\ n = 86$$

The R^2 value is slightly higher than R^2 in equation (26) showing that variations in TOTALt are explained more by variations in DISTKM and LOADcM rather than DISTKM alone. Since the value of R^2 is still low, it is clear that there are other variables or factors whose variation influence the variations in TOTALt also. Appendix 17 shows that the coefficients of DISTKM and LOADcM are both significantly different from zero at 1 per cent level of statistical probability.

Since the total machine hours spent loading murram onto trucks and the total distance travelled during the murraming operation are summarized in Table 22, the total cost incurred hauling murram to the road site is computed as below:

Cost item			Total cost in shs
County tractor:	shs 309.15/h x 13.0 h	...	4 018.95
215 excavator:	shs 692.60/h x 21.0 h	...	14 544.60
Isuzu trucks:	shs 8.60/km x 969.5 km	...	8 337.70
Overhead:	shs 294.20/day x 12 days	...	3 530.40
Total		...	30 431.65

4.3.5 <u>Spreading and compaction of murram</u>

Spreading and compaction of murram was done simultaneously with the hauling operation. By tipping the truck body while moving at low speed it was possible to spread the load of murram over a length of 8-10 m. Two people were employed to remove big stones from the road surface and smooth out the murram (Plate 5).

Spreading of the murram was started at the upper end of the road and proceeded down slope to the Rest House. This arrangement not only improved compaction of the murram by the trucks but also protected the subgrade (not yet murramed) from being destroyed by the hauling trucks.

The total cost of spreading murram and removing large stones from the road surface manually was:

shs 23.05/day x 36 mandays = shs 829.00

4.3.6 Cost summary

The total cost of providing murram for the road is summarized below:

Activity		Total cost in shs
Improving the track to the murram pit	...	5 885.85
Construction of the wooden bridge	...	3 911.65
Clearing and digging the murram	...	44 852.20
Hauling murram to the road site	...	30 431.65
Spreading and removing stones manually	...	829.00
Total	...	85 910.35

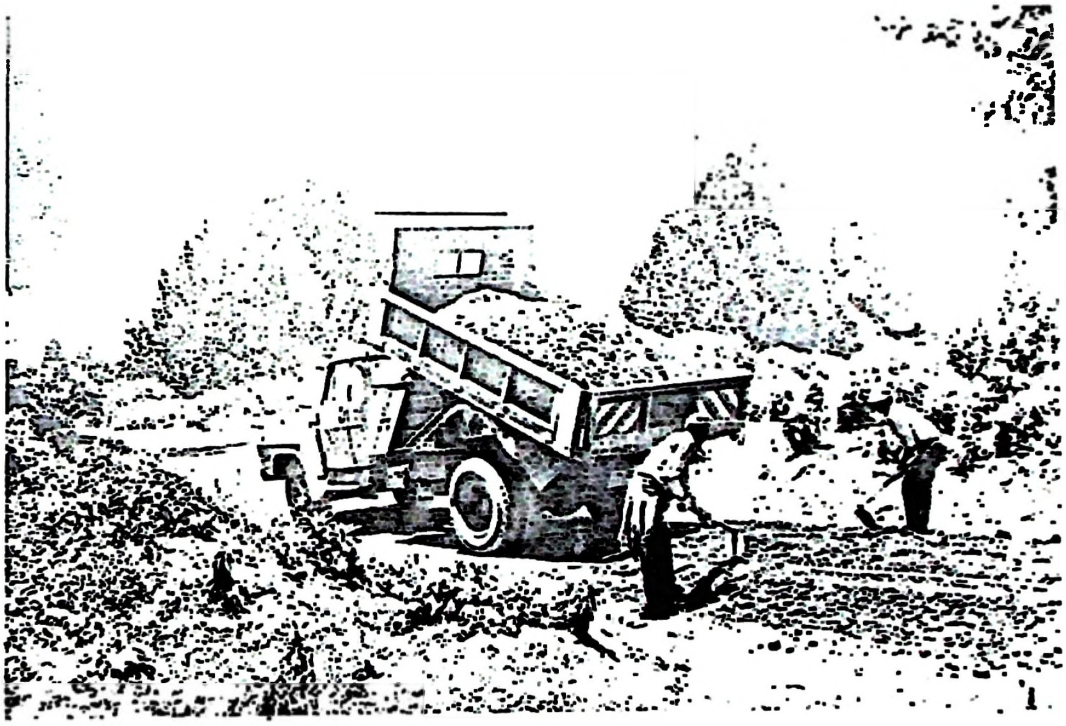


Plate 5. A 7 t Isuzu truck unloading murram while a two-man crew spreads it evenly on the carriage way.

From the above summary, the cost of murraming 1 km is shs 66 084.90. This is almost the same as the shs 66 550/km estimated at the Meru Forest Project (Chilembu 1984) to murram a road of the same standard.

From the above summary, it is important to note that the clearing and digging operation took over 50 % of the total cost. Such high costs are explained by the extensive use of heavy machines (D6 bulldozer and 215 excavator) and earth moving required to open the murram pit.

Since a total volume of 458 m^3 of murram was hauled to the road site (Table 22) at a total cost of shs 85 910.35, the average cost of excavating and hauling 1 m^3 of murram was shs 187.80/m^3. This is within the range of cost estimates given by the Zangie Contractor in 1979/80 when constructing the existing access roads in the SUA Training Forest. The cost of excavating and hauling 1 m^3 of murram per trip for the different road sections (Figure 1) were given by Zangie Contractor as shs 165/m^3 (I), shs 180/m^3 (II), shs 210/m^3 (III) and shs 265/m^3 (IV). The average hauled volume per trip was 3 m^3 of murram while round trip distances for these road sections were 5 km (I), 9 km (II), 14 km (III) and 8 km (IV).

4.4 Culvert installation

Prior to the actual culvert installation operation, the following activities were carried out: location of the culvert points, determination of culvert sizes required, moulding of culverts and transport of stones, gravel and culvert sections to the culvert points. Figure 3 shows the 10 selected culvert points along the new road.

A concrete mixture formed of murram and cement in the ratio of 3:1 was used to fabricate 1 m culvert sections. A 45 cm culvert section required a 50 kg bag of cement while a 60 cm culvert section used up 1 1/3 bags of cement. A total of 60 culvert sections were moulded; out of these, thirty-five were 45 cm in diameter and the rest were 60 cm in diameter.

Installation of culverts was done manually by a crew of 7 people. Altogether 10 culverts were installed but before the last two culverts were installed, the average element times in each of the first eight culverts were computed and analysed for improvement. Activities in order of their performance and their average durations for the first eight culverts are shown in Table 24.

Digging of culvert channels was done manually using hoes, spades and mattocks (Plate 6). In order to minimize the rate of water flow within the culverts, the channels were inclined at a slope angle of 3-4 % although a fall of 5-6 % is recommended by Skaar (1972c).

Culvert sections were laid down in the channels manually (Plate 7) and filled with well-compacted soil on both sides. On top, a 10 cm concrete layer of murram and cement of the ratio 3:1 was poured and left to dry until the next day. Each culvert point required 3 bags of cement to make this concrete layer.

On both ends of the culvert, end walls of the dimension 1.50 x 0.53 x 0.20 m were erected. Each end wall required murram and stones plus 1 bag of cement to make a concrete mixture in the ratio 4:1. Plate 8

Table 24. Activities and their estimated average durations during the installation of the first 8 culverts and the proposed order of work performance when installing the remaining two culverts.

Activity or work element	Pre-requisite activity	Total duration (h)
A. Digging the culvert channel	-	3.08
B. Preparing the concrete mixture	A	1.36
C. Laying, assembling and filling around the culvert sections with soil	A	1.67
D. Pouring and spreading concrete mixture on top of culvert	B,C	0.39
E. Preparing the concrete mixture for the end walls	D	1.17
F. Fixing end wall moulds on both ends of the culvert	D	1.27
G. Filling endwall moulds with concrete mixture and stones	E,F	0.58
H. Covering the culvert top with murram layer	G	0.56
I. Preparing apron and collar concrete bases	H	0.93
J. Preparing concrete mixture for the culvert apron and collar	I	0.42
K. Fixing side wall moulds to culvert aprons and collars	I	0.73
L. Building apron and collar with concrete mixture and stones	J,K	0.89
M. Plastering the end walls, aprons and concrete collar walls	L	1.11
	Total	14.16

Plate 6. A 4-man crew digging the culvert channel manually.

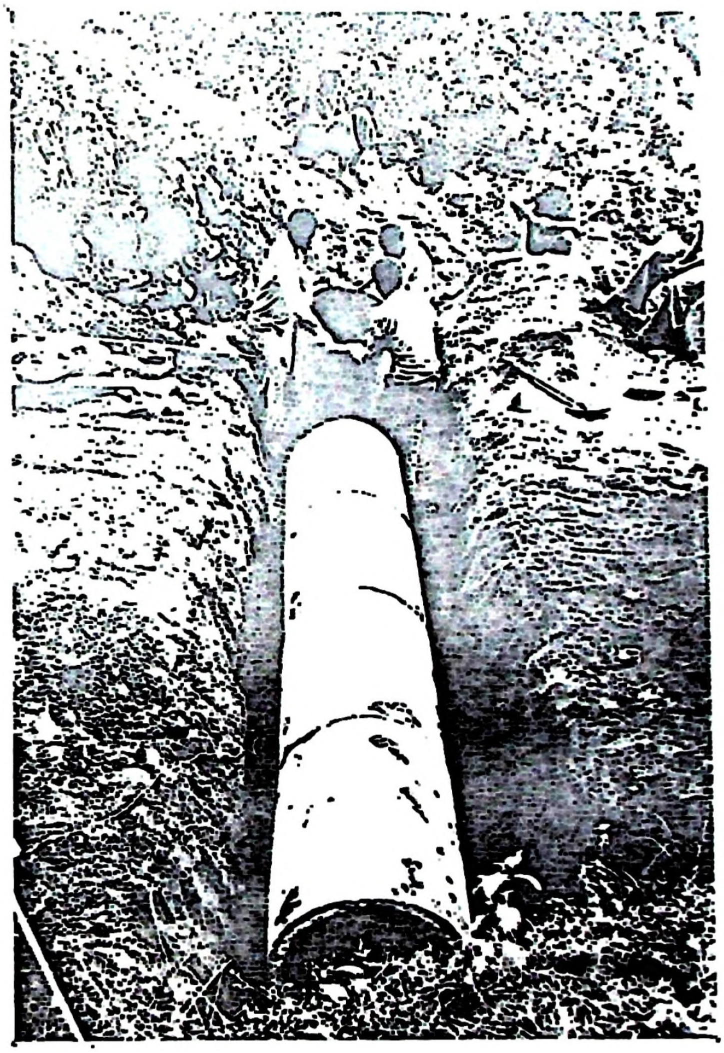


Plate 7. Laying the culvert sections manually.

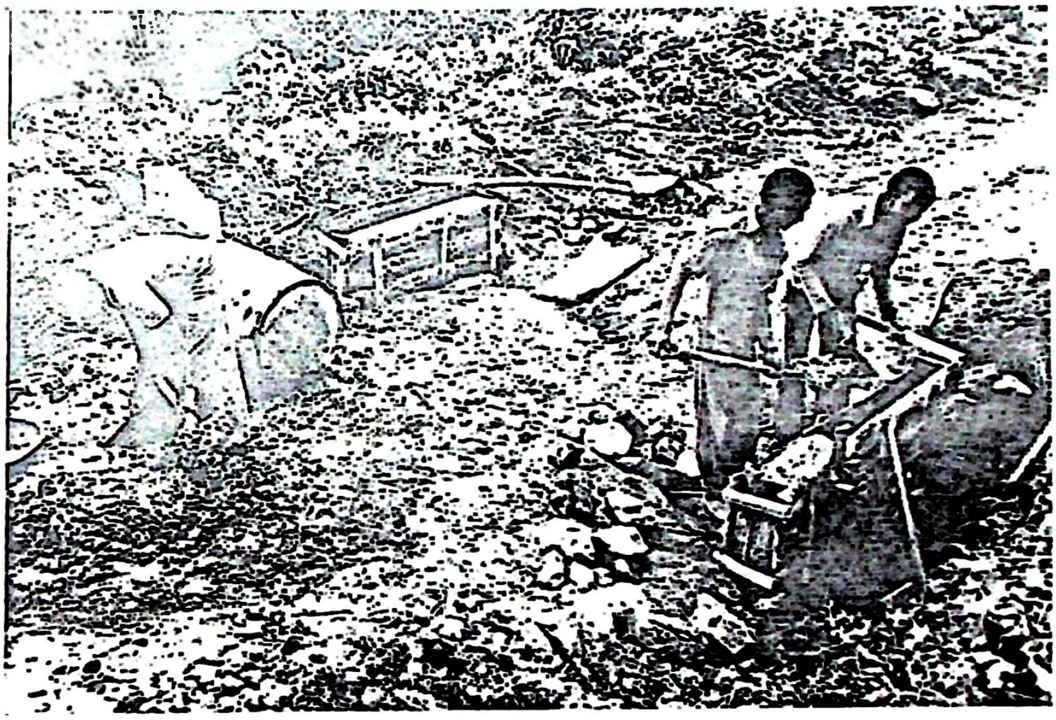


Plate 8. Erection of end walls by filling the end wall moulds with concrete mixture and stones.

shows a mason plus two helpers building one of the end walls. At the inlet and outlet of each culvert, concrete collars and aprons were constructed to prevent undercutting of culverts by running water. The concrete layer on top of the culvert was covered or filled with a 20-25 cm murram layer before employing one person to dig mitre drains at the ends of each culvert apron. These mitre drains were aimed at facilitating smooth drainage of water from the culverts.

Based on the experience gained during the installation of the first 8 culverts, a work sequence to be followed when installing the remaining 2 culverts was formulated. This work sequence was aimed at reducing the total installation time and improving work or methodological performance. Thus, in addition to Table 24 showing the average durations of the different work elements, the order of their performance is also shown.

In order to be able to isolate the critical activities, i.e., the activities which must be completed on time if the minimum installation time is to be achieved, the situation was analysed using the Critical Path Method (CPM). As a first step, based on Table 24, a network diagram for this operation was drawn as shown in Figure 9.

Because activity A has no preceeding activity (Table 24), its origin must be node 1, the beginning of the project. Referring to Table 24, the pre-requisite indicates that both activities B and C must be completed before activity D can begin. A "dummy" activity with a duration time of zero is therefore introduced to join nodes 3 and 4.

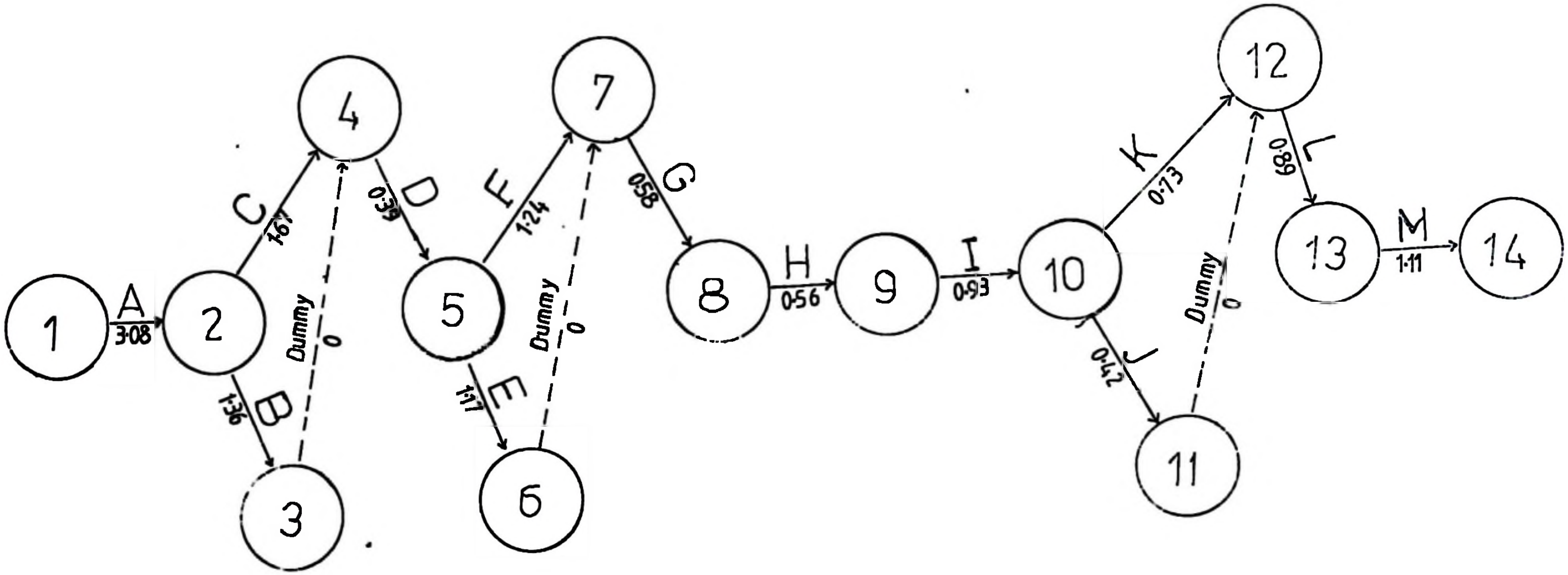
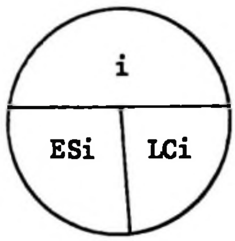


Figure 9. A network diagram for the culvert installation problem.

This dummy activity is distinguished from "real" activities by drawing it with a dashed line. Other dummy activities encountered on the network are indicated in Figure 9, i.e., between nodes 6 and 7 and between nodes 11 and 12. It is important also to note that, under each activity, activity duration is shown and the numbering of nodes is in ascending order from left to right.

To solve the critical path problem, an algorithm which lends itself to hand calculation was used. This algorithm is described in detail by Dykstra (1984) but, before examining it, the following notations which appear in every node are defined.



i = node number

ESi = Earliest Start time of all activities beginning at node i

LCi = Latest Completion time of all activities ending at node i

Assuming that the installation of a culvert starts at time zero, a value of zero is assigned to node 1, i.e., ES1 = 0. This initiates the algorithm. Since activity A requires 3.08 h and ESi is a cummulative time from the start of the project until the activities at node i can begin, then ES2 = ES1 + D12, where D12 is the duration of activity A (Figure 9). Therefore, ES2 = 0 + 3.08 h = 3.08 h and this value is entered in the space for ES2 in node 2 as indicated in Figure 10. By similar reasoning, ES3 = 3.08 + 1.36 = 4.44 h and ES4 = 3.08 + 1.67 = 4.75 h (Figure 10).

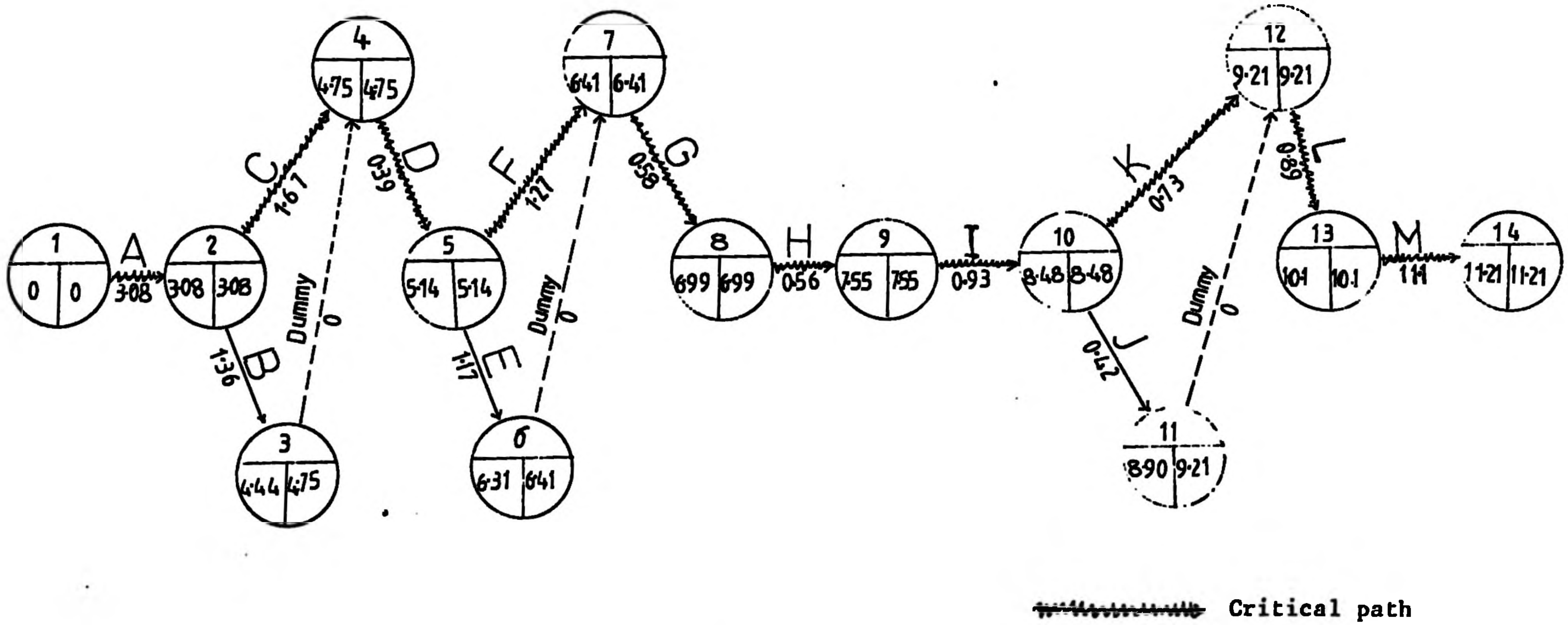


Figure 10. A completed CPM network for the culvert installation operation with all ESi, LCi values and the critical path indicated.

The values of ESi for all the nodes in the network are determined using the procedure described by Dykstra (1984). This procedure is summarized below.

Step 1: Set ES1 = 0

Step 2: Proceed to the next higher numbered node, node j. The earliest start time of all activities starting at node j is computed as: ESj = max(ESi + Dij)

For example, to determine the value of ES4, both ES2 and ES3 are considered because activity C and "dummy" activity end up in node 4. The value of ES4 is equal to the longer of:

ES2 + D24 = 3.08 + 1.67 = 4.75 h

ES3 + D34 = 4.44 + 0 = 4.44 h

The more limiting time is 4.75 h and this becomes the value of ES4.

Step 3: If j = N, stop. ESN is equal to the total project duration. If j < N, then go to Step 2.

From Figure 10, the earliest start time in the last node, ESN = ES14, is 11.21 h. This indicates that the culvert can be installed within 11.21 hours if the critical activities are started and completed on time. This would mean saving 2.95 h, or 20 % of the total installation time per culvert; as compared to the average total time of 14.16 h in Table 24.

As stated earlier, activities to be improved must lie on the critical path. To determine the activities which lie on the critical path, a backward pass is made from node N to node 1. During the backward pass, the latest completion time for each node i (LCi) is computed. Backward pass as described by Dykstra (1984) involves the following steps.

<u>Step 4</u>: Set LCN = ESN

<u>Step 5</u>: Proceed to the next lower numbered node, node i. Compute the latest completion time for all activities ending at node i as follows: $LCi = \min_{j}(LCj - Dij)$

<u>Step 6</u>: If i = 1, stop; the backward pass has been completed. If i > 1, go to Step 5.

At the end of the backward pass, LC1 must equal ES1. Figure 10 shows values of ESi and LCi for all nodes.

An activity is considered to lie on the critical path if it satisfies the following three conditions simultaneously (Dykstra 1984):

(i) ESi = LCi

(ii) ESj = LCj

(iii) ESj - ESi = LCj - LCi = Dij

Using the above conditions, activities which lie on the critical path are: A, C, D, F, G, H, I, K, L and M, as indicated in Figure 10. These were activities where emphasis had to be put when installing the remaining two culverts in order to reduce total installation time.

From Figure 10, it is observed that the non-critical activities are associated with slack or float time. This is the case in nodes 3, 6 and 11 for activities B, E and J respectively. These activities can be delayed for a period indicated by the float time without causing any delay on the total project duration or if they are started on time, these are free or float times experienced when waiting for the corresponding critical activities to be completed.

One objective of the CPM is to identify critical and non-critical (slack) activities so that work can be organized in such a way that both critical and slack activities are done simultaneously. From the preceding paragraphs, it is evident that through CPM analysis, it is possible to reduce total culvert installation time (experienced during the first eight culverts) from 14.16 h (Table 24) to 11.21 h (Figure 10) by following the schedule indicated by CPM solution, i.e., doing both critical and non-critical activities simultaneously.

CPM also helps to identify critical activities so that emphasis is put on the improvement of these activity times. During the installation of the last two culverts, CPM analysis was incorporated in the work methodology in order to improve installation time. Results from the time studies carried out show that besides the 2.95 h saved through CPM analysis (by doing both critical and non-critical activities simultaneously), another 1.05 h were saved due to improvement in activity times. Table 25 compares the activity times experienced during the installation of the first eight culverts and the last two culverts.

Table 25. Activity times and improvement of activity times during the installation of the first eight and the last two culverts.

Activity	Installation time (h)		Improvement in activity times (h)
	First 8 culverts	Last 2 culverts	
A	3.08	2.96	0.12
B	1.36	1.17	0.19
C	1.67	1.36	0.31
(B + C)[1]	3.03	1.36	(1.67)[2]
D	0.39	0.38	0.01
E	1.17	1.55	-0.38
F	1.27	1.38	-0.11
(E + F)[1]	2.44	1.55	(0.89)[2]
G	0.58	0.50	0.08
H	0.56	0.48	0.08
I	0.93	0.67	0.26
J	0.42	0.40	0.02
K	0.73	0.58	0.15
(J + K)[1]	1.15	0.58	(0.57)[2]
L	0.89	0.83	0.06
M	1.11	0.85	0.26
Totals	14.16	10.16	1.05

Notes: 1. Total elapsed time when the two activities in brackets are done sequentially (in the case of the first eight culverts) and simultaneously (in the case of the last two culverts).

2. Total improvement in project time due to adoption of the schedule indicated by the CPM solution and improvement or reduction in activity times.

The Table shows that unlike in the first eight culverts, only one time for each pair of the following activities (which were done simultaneously) was used in determining total installation time during the last two culverts: B and C, E and F, and J and K. In this case, the activity which took longer duration was the one which determined project duration i.e., C, E and K.

Improvements in the activity times plus the scheduling of activities so that both critical and non-critical activities could be done simultaneously (during the last two culverts) resulted in the reduction of average installation time by 4.00 h or 28 % of the average installation time for the first eight culverts. Although some of the reduction in individual activity times is probably due to a learning effect that would have occured in any case, it is certain that incorporation of CPM analysis significally contributed to the reduction of total culvert installation time during the last culverts.

Costs incurred on installation of culverts are divided into two categories: preparatory costs and installation costs. These are summarized as follows:

Cost item			Total cost in shs
A: Preparatory costs			
A1. Location and determination of culvert sizes:			
Field work :	shs 23.05/manday x 2 mandays	...	46.10
Office work:	shs 70.55/manday x 3 mandays	...	211.65
Overhead :	shs 294.20/day x 2 days	...	588.40

Cost item			Total cost in shs
A2. Moulding of culvert sections:			
Mason	: shs 35.00/day x 25 days	...	875.00
Helper	: shs 23.05/day x 25 days	...	576.25
Cement	: shs 120.00/bag x 69 bags	...	8 280.00
Murram			
- digging	: shs 309.85/h x 9.3 h	...	2 881.60
- hauling	: shs 147.30/h x 12.0 h	...	1 767.60
Rented culvert forms:	shs 100.00/day x 25 days	...	2 500.00
A3. Transport of murram to culvert points:			
Hauling	: shs 8.60/km x 50.4 km	...	433.45
A4. Transport of culvert sections to culvert sites:			
Labour	: shs 23.05/manday x 14 mandays	...	322.70
Truck	: shs 8.60/km x 98 km	...	842.80
Sub-total		...	19 325.55

B. Installation costs

B1. Channel digging, culvert assembly and erection of end walls:			
Mason	: shs 60.00/day x 10 days	...	600.00
Labour (6)	: shs 23.05/manday x 60 mandays	...	1 383.00
Hand tools (4):	hoes, spades and mattocks	...	110.25
Overhead	: shs 294.20/day x 10 days	...	2 942.00
B2. Filling the road way with murram:			
Labour	: shs 23.05/manday x 4 mandays	...	92.20
Overhead	: shs 294.20/day x 1 day	...	294.20
B3. Construction of aprons and concrete collars:			
Mason	: shs 60.00/day x 5 days	...	300.00
Labour	: shs 23.05/manday x 10 mandays	...	230.50
Cement	: shs 120.00/bag x 7 bags	...	840.00
Overhead	: shs 294.20/day x 5 days	...	1 471.00
B4. Plastering the end walls, aprons and collars:			
Mason	: shs 60.00/day x 3 days	...	180.00
Labour	: shs 23.05/manday x 6 mandays	...	138.30

Cost item				Total cost in shs
B4.	Cement	: shs 120.00/bag x 3 bags	...	360.00
	Overhead	: shs 294.20/day x 3 days	...	882.60
B5.	Digging outlets and mitre drains:			
	Labour	: shs 23.05/manday x 15 mandays	...	345.75
Sub-total			...	16 169.80

The overall total cost incurred on installing the 10 culverts is

shs 19 325.55 + shs 16 169.80 = shs 35 495.35

or shs 3549.55 per culvert. The actual installation cost took only 45 % of the total cost while the rest was spent on moulding of culverts and transport of materials and culvert sections to the sites. From the above computations, moulding of culvert sections seems to be the most expensive cost item, with about 48 % of the total cost being spent on this activity alone. This raises a question for future investigation as to whether it would be cheaper to construct box culverts instead of using concrete pipe culverts on this road.

When the total installation costs per culvert are compared with the cost figures quoted by the two road building contractors, the costs in this study are found to be comparatively low despite the contractors' cost figures being based on 1978 and 1981 market prices. Costs of moulding two sizes of culverts and installing them are summarized in Table 26 for the three different road contractors.

Table 26. Estimated costs of moulding and installing culverts in the SUA Training Forest.

Contractor	Road section	Moulding cost/m		Installation cost/culvert	Total cost/ culvert*1	
		45 cm (shs/m)	60 cm (shs/m)	(shs)	45 cm (shs)	60 cm (shs)
MECCO	I-III	732	1000	11 073	15 465	17 073
	V	732	1000	10 780	15 172	16 780
Zangie Co.	I-III	700	1000	5 000	9 200	11 000
	IV	1500	1500	6 250	15 250	15 250
University	(This study road)	263	302	1 858	3 436	3 670

Note: *1 6 culvert sections/culvert point.

When culvert installation costs in section IV (installed by Zangie Contractor) are compared with the installation costs found in this research road, the former is found to be four times more costly. On the other hand, in the Logging and Road Building Unit it is estimated that it costs shs 200.00 to mould 1 m culvert section and about shs 2955.00 to install one culvert point (Chilembu 1984).

4.5 Overall cost summary

Cost item		Total shs	%
Routing and clearing the roadline	...	69 681.50	24
Earth work	...	95 676.15	33
Gravelling	...	85 910.35	30
Culvert installation	...	35 495.35	13
Overall total cost	...	286 763.35	100

From the above summary, the cost of constructing 1 km of road is shs 220 587, or about shs 220.60 per m. This is about 42 % of what the Zangie Contractor estimated would be needed to construct this section of road in 1981 and 34 % of MECCO's 1979 bid.

According to Chilembu (1984), the average cost for the Logging and Road Building Unit to construct a similar road in the Meru Forest Plantation is shs 120.60 per m. The low construction figure given by Chilembu is attributed to under-estimation of the true machine costs. For instance, Chilembu (1984) estimates the costs of owning and running earth moving machinery at only about 55 % as much as the detailed cost data collected for this study suggest.

Earthwork and road gravelling operations, which are generally considered to be the most expensive operations when constructing roads (Allal and Edmonds 1977), required about 63 % of the total construction costs.

The cost distribution among labour, machines and materials shows that labour cost accounts for about 21 % of the total construction costs, machine costs about 73 % and materials, only 6 %.

While in some operations labour cost took a significant portion of the total cost, in most operations, machine and material costs took a bigger share of the total cost. Cost distribution on labour, materials and machines on different work operations shows the following pattern.

Cost item	Cost distribution (%)		
	Labour	Materials	Machines
Routing and clearing the road line ...	37	-	63
Earth work ...	14	-	86
Gravelling ...	13	1	86
Culvert installation ...	32	44	24
Weighted average ...	21	6	73

Where forest roads are to be constructed manually, overhead costs (transportation of workers to and from the forest) could be minimized by recruiting the labour force within a radius of 3 km from the work place (Deepak 1978). Long walking distances, fluctuation of labour force, lack of relevant expertise and equipment sometimes result in labour-intensive roads being more expensive. For instance, in Kenya where 1200 km of rural access roads were constructed manually using a labour force of about 10 000 people, a 4 m wide gravel road cost about shs 780/m (Strehlke 1982), more than three times as much as the semi-mechanized road construction in this study. This means therefore, before a decision is made on whether to use manual labour or machines for constructing forest roads, the above factors have to be taken into consideration since manually constructed roads are not necessarily minimum cost roads.

CONCLUSIONS

This study has been undertaken with the objective of developing a comprehensive methodology to improve planning, control and cost-estimation procedures for the construction of forest roads in Tanzania. The methodology thus developed incorporates the following:

(a) Conventional engineering procedures for road design and layout, including culvert-size determination.

(b) Mathematical programming for planning of earthwork allocation among cut and fill areas.

(c) Estimation of hauling costs as a function of road gradient and alignment, utilizing the principles of engineering mechanics.

(d) Determination of the optimal road design from among several alternatives using as a criterion the total of earthwork cost and hauling cost.

(e) Utilization of the critical path method to reduce the cost of culvert installation.

The mathematical programming techniques utilized in this study are not new but they have not been widely used in forest road planning and construction. Furthermore, these techniques have not previously been integrated into a comprehensive planning and control methodology for the purpose of improving forest road construction procedures.

For the purpose of this study, the optimum road grade was taken to be the one which minimized the total of earthwork and hauling costs. To determine an optimum road grade to construct, three road grade alternatives; i.e., 6 %, 8 % and 10 %, divided into 30 m, 60 m and "terrain" working segments (interval lengths), were tested for optimality using two solution procedures; microcomputer-based linear programming (LP) and a transportation tableau method. Although both resulted in identical optimal solutions, the transportation tableau method seems at present to be more appropriate for use in Tanzania due to the fact that it requires only hand calculations (LP, by contrast, requires access to a microcomputer).

In utilizing the transportation method, the most natural criterion for allocating earthwork among cut and fill areas would be the minimization of earthwork costs. In selecting the road design to be used in this study, distance between net cut and net fill areas was used as a surrogate for the cost of transporting earthwork between these areas. After the road was built so that costs could actually be determined, an analysis showed that the same design alternative would have been selected on the basis of cost as was selected on the basis of earth moving distance. Although this cannot be considered a general result, it verifies that for the road involved in this study, earthwork cost was proportional to earth moving distance.

To determine earthwork volumes during road design, engineers typically determine cross-sectional cut and fill areas at intervals of about 30 m along a route (Moffit and Bouchard 1975). As a test of the sensitivity of this parameter, three interval lengths were used in this study: 30 m, 60 m and "terrain" distance (lengths determined by terrain breaks). Analysis of earth moving costs for the three interval lengths showed that only the 30 m intervals provided satisfactory accuracy. Earthwork costs estimated using the other interval lengths were only about 70 % of the actual costs as estimated by the shorter 30 m interval length.

In the 1950's, a team of engineers at the U.S. Forest Service developed a theory for calculating log hauling times based on road gradient and alignment (Byrne *et al.* 1960). However, this theory has not previously been utilized as part of a comprehensive methodology to determine optimum road design on the basis of total earthwork and hauling cost. The application in this thesis demonstrates that such a methodology can be satisfactorily developed and utilized in forest road engineering.

The critical path method (CPM) has been used in many construction projects to streamline the work sequence and assist decision makers in assigning priorities to critical activities so that both construction costs and time are minimized (Haarlaa 1974). The application of CPM in this study resulted in a reduction of total culvert installation time by about 28 %. Although other factors, particularly the learning effect, undoubtedly contributed to this reduction of total installation time, it is evident that improved work organization through CPM was a major factor in reducing total culvert installation time.

The total construction cost of the research road considered in this study was shs 286 763 or shs 220 587/km. Earthwork and road gravelling, both done by machines, accounted for about 33 % and 30 % of the total cost, respectively, while the other two operations, routing/road clearing and culvert installation (done manually), took about 24 % and 13 % respectively. Compared to bids submitted by private and public road contractors, the cost of constructing this research road was only 1/3 to about 1/2 as much as the contractors' estimates.

5.1 Suggestions for additional research

During this study, several possibilities for additional research aimed at further improvements in methods for planning and control of forest road construction became evident. These include the following:

(a) An optimum road grade might be viewed as the one which minimizes the sum of earthwork, hauling and maintenance costs. In order to broaden the methodology developed in this study so that maintenance cost can be considered, research would be required on the influence of road gradient and possibly other design parameters on the cost of maintaining forest roads in Tanzania.

(b) To determine total hauling cost over a given road grade, several variables which influence vehicle hauling costs must be measured. Since in real situation it will be very difficult to measure all the variables involved, simulation models could be used to predict the required vehicle hauling power over various road grades and alignment and show how hauling speed is influenced by vehicle power, road grade, load size and road alignment.

(c) Earthwork and road surfacing were both carried out in this study as machine-intensive operations. Although this is the most common way of doing such operations in Tanzania, to some extent it is counter to the national policy of maximizing labour-intensive operations. In order to use labour effectively for earthwork and road surfacing, research on ways to motivate and organize labourers will be required. Besides reducing unemployment, the successful development of labour-intensive road construction methods would also reduce the fraction of road construction costs allocated to foreign exchange through the purchase of machinery, spare parts, and fuel.

(d) The use of round concrete culverts on forest access roads has more or less become a tradition in Tanzania. Considering the time and cost involved in moulding the culvert sections, transporting them to the site and assembling them, there is a need to study the methodology, costs and time involved in constructing box culverts for comparison purposes.

5.2 Concluding remarks

The analysis and methodological procedures presented in this study provide guidelines to forest managers and other personnel engaged in the planning and construction of forest roads. As with any quantitative methodology, it should be emphasized that the procedures used in this study only provide information for decisions; they must be used together with the forest engineer's judgement in order to improve forest-road decision making. Invariably, it is impossible to fully describe a complex planning problem in terms of a mathematical model. All of the models used in this study (the earthwork cross-section model, the transportation model, the engineering-mechanics model for truck travel time, and the critical path model) are abstractions from reality. If these models can help improve forest-road planning and construction, then they can provide a valuable service to forest managers by suggesting actions that can be taken and the probable consequences of those actions. No model can relieve a forest manager of the responsibility of making the difficult and costly decisions that have to be made in providing access to all parts of a forest. The procedures applied in this study can, however, provide a stronger foundation on which to make those decisions.

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WSA:ys
31/3 1985

Appendix 1. Data sheet used during road alignment.

Date: .... Tools used: ....

No. of people/crew: ....

Station No.	Longitudinal profile		Cross-sectional profile						Remarks
			Height difference from the centre of the road						
	Compass azimuth	Gradient (%)	6 m L	4 m L	2 m L	2 m R	4 m R	6 m R	

Appendix 2. Drainage table based on Talbot's formula for 100 mm per hour rainfall.

No. of hectares	Area in square meters required for waterway							
	Impervious 100 % Runoff	Steep slopes, Heavy soils Moderate cover		Moderate slopes, Heavy to light soils Dense cover		Gentle slopes, Agricultural soil and cover		Flat land Pervious soils
	C = 1.0	C = 0.8	C = 0.7	C = 0.6	C = 0.5	C = 0.4	C = 0.3	C = 0.20
1	.183	.146	.128	.110	.092	.078	.055	.037
2	.300	.240	.216	.185	.154	.123	.092	.062
3	.418	.334	.292	.250	.209	.167	.125	.083
4	.518	.414	.363	.311	.259	.207	.155	.104
5	.613	.490	.429	.367	.306	.245	.184	.122
6	.702	.562	.491	.421	.351	.281	.211	.140
7	.788	.630	.552	.473	.394	.315	.236	.158
8	.871	.697	.610	.523	.436	.348	.261	.174
9	.951	.761	.666	.571	.478	.381	.285	.190
10	1.030	.824	.721	.618	.515	.412	.309	.206
20	1.731	1.305	1.212	1.039	.866	.693	.520	.346
30	2.348	1.878	1.643	1.408	1.174	.939	.704	.469
40	2.913	2.330	2.038	1.747	1.456	1.165	.874	.582
50	3.442	2.754	2.410	2.066	1.722	1.377	1.033	.689
60	3.947	3.158	2.763	2.368	1.974	1.579	1.184	.789
70	4.431	3.545	3.102	2.659	2.216	1.773	1.329	.886
80	4.898	3.918	3.429	2.939	2.449	1.959	1.469	.980
90	5.350	4.280	3.745	3.210	2.675	2.140	1.605	1.070
100	5.790	4.632	4.053	3.474	2.895	2.316	1.737	1.158

Appendix 3. LP output for the designed 8 % road grade alternative.

Variable No.	Variable name	At	Activity	Objective coefficient	Reduced cost
1	X0101	bs	18.6	60	0
2	X0201	ll	0.0	90	30
3	X0301	ll	0.0	120	60
4	X0102	bs	26.9	30	0
5	X0202	ll	0.0	60	30
6	X0302	ll	0.0	90	60
7	X0402	ll	0.0	120	90
8	X0503	ll	0.0	120	90
9	X0603	ll	0.0	90	60
10	X0703	ll	0.0	60	30
11	X0803	bs	74.5	30	0
12	X0804	bs	57.5	60	0
13	X0904	bs	54.0	60	0
14	X0905	bs	9.9	30	0
15	X1005	ll	0.0	60	30
16	X1105	ll	0.0	90	60
17	X1206	ll	0.0	60	30
18	X1306	bs	3.7	30	0
19	X1406	ll	0.0	60	60
20	X1507	bs	6.3	60	0
21	X1407	bs	64.6	30	0
22	X1507	ll	0.0	60	0
23	X1508	ll	0.0	60	30
24	X1608	bs	13.0	30	0
25	X1708	ll	0.0	30	0
26	X1709	bs	13.9	30	0
27	X1809	ll	0.0	30	0
28	X1810	bs	9.2	30	0
29	X1910	bs	12.4	30	0
30	X2010	ll	0.0	60	30
31	X1911	ll	0.0	60	30
32	X2011	ll	0.0	30	0
33	X2111	bs	44.5	30	0
34	X2211	ll	0.0	60	30
35	X2212	ll	0.0	60	30
36	X2312	bs	40.4	30	0
37	X2313	bs	17.5	60	0
38	X2413	bs	181.3	60	0
39	X2414	bs	41.0	30	0
40	X2514	ll	0.0	60	30
41	X2515	ll	0.0	60	30
42	X2615	bs	5.4	30	0
43	X2715	ll	0.0	60	30
44	X2616	ll	0.0	60	30
45	X2716	bs	20.7	30	0
46	X2816	ll	0.0	50	20

Z value = 31 515 (MIN)

Appendix 4. Data sheet for sub-operations which were timed on shift-level basis.

Date	Type of activity or sub-operation	No. of workers	Tools or equipment	Machine hours	Work output	Section of the road (peg nos.)	Starting time	Break time	Ending time

Remarks:

Appendix 5. Data sheet for earth work and grading sub-operation.

Date: .... Ending time: ....

Machine type: .... Starting time: ....

Mandays: .... Difference: ....

Section of the road: ....

Recorded time / Activity	Effective time (min)	Delay time (min)		Total activity time (min)
		Necessary delay	Unnecessary delay	
Travelling to the road site				
Actual earth work or grading sub-operation				
Travelling back from the road site				
Break				
Total				

Remarks:

Appendix 6. Data sheet used in recording the cross-sectional height differences of each station during the earth work sub-operation.

Station No.	Measurements	Gradient (%)	Cross-sectional height difference							
			4 m L	3 m L	2 m L	1 m L	1 m R	2 m R	3 m R	4 m R
	Before After									
	Before After									
	Before After									
	Before After									
	Before After									
	Before After									
	Before After									
	Before After									
	Before After									

Remarks:

Appendix 7. Data sheet used during gravelling sub-operation.

Date: .... Break time: Starting .... Hauling time: Starting ....

No. of trucks/day: .... Ending .... Ending ....

Difference .... Difference .....

Trip No.	Distance (km)	Volume/ trip (m^3)	Road section (peg nos.)	Loading time (min)	Travelling loaded (min)	Unloading time (min)	Travelling empty (min)	Remarks

Appendix 8. Culvert installation data sheet.

Date:	Culvert No.:	<u>Break time</u>	<u>Installation time</u>
No. of workers: ...	Culvert size:	Starting:	Starting: ....
Tools used:	Laying slope:	Ending:	Ending: ....
	No. of culvert sections:	Difference:	Difference: .....

Activities	Starting time	Ending time	Delays time	Mixture ratio & other remarks
1. Digging the culvert channel				
2. Preparing the concrete mixture				
3. Laying, assembly and filling of culvert sections with soil				
4. Pouring concrete mixture on top of assembled culvert				
5. Preparing the concrete mixture for the end walls				
6. Fixing end wall moulds on both ends of the culvert				
7. Filling the end walls with concrete				
8. Covering the concrete layer on the road way with murram				
9. Preparing concrete bases for aprons and culvert collars				
10. Preparing concrete mixture for aprons and culvert collars				
11. Fixing side wall moulds for the aprons and culvert collars				
12. Constructing aprons and collars				
13. Plastering the end walls, aprons and culvert collar sides				

Appendix 9. The annual cost of owning and running a County 1164 tractor and a Ford 6600 tractor.

Type of tractor:	County 1164	Ford 6600
Purchase price:	shs 450 000	shs 202 000
Estimated life span:	6 years	6 years
Average machine hours/year:	750 h	860 h
Estimated salvage value:	10 %	10 %

Fixed costs		
Depreciation:		
County : (450000 - 45000)/6	67 500.00	-
Ford 6600: (202000 - 20200)/6	-	30 300.00
Interest:		
County : {(450000 + 45000 + 67500)/2}8 %	22 500.00	-
Ford 6600: {(202000 + 20200 + 30300)/2}8 %	-	10 100.00
Insurance: (From Training Forest records):	7 047.00	1 239.30
Taxes: (From Training Forest records):	2 500.00	500.00
Sub-total	99 547.00	42 139.30
Variable costs		
Maintenance and repair: (100 % of depreciation)	67 500.00	30 300.00
Fuel:		
County : 8.3 l/h x 750 h x shs 5.00/l	31 125.00	-
Ford 6600: 5.5 l/h x 860 h x shs 5.00/l	-	23 650.00
Oil and lubricants:		
County : 0.23 l/h x 750 h x shs 30.00	5 175.00	-
Ford 6600: 0.061 l/h x 860 h x shs 30.00	-	1 573.80
Labour:		
Driver: 940 x 12 x 1.68	18 950.40	18 950.40
Helper: 600 x 12 x 1.40	10 080.00	10 080.00
Sub-total	132 830.40	84 554.20
Grand total	232 377.40	126 693.50

N.B. The cost of owning and running/hour	shs 309.85	shs 147.30

Appendix 10. The annual cost of owning and running an Isuzu 7 ton truck (excluding fuel cost).

Model	:	TXD 45
Purchase price	:	shs 270 000
Estimate life span	:	6 years; 9000 machine hours; 150 000 km
Estimate salvage value	:	10 % of the orginal price

Fixed costs	shs/year
Depreciation: {270000 - (270000 x 10 %)}/6	40 500
Interest: {(270000 + 27000 + 40500)/2}8 %	13 500
Insurance: (From Training Forest records):	7 047
Taxes: (From Training Forest records):	2 500
Sub-total	63 547
Variable costs	
Fuel: see Section 3.7.1.6	-
Oil and lubricants: 0.145 l/h x 1500 h x shs 30.00	6 525
Maintenance and repair: 100 % of depreciation	40 500
Tyres: 1 set/year @ shs 3000 x 6	18 000
Labour: Driver -- shs 900 x 12 x 1.40	15 120
Turnboy - shs 600 x 12 x 1.40	10 080
Sub-total	90 225
Grand total	153 772

N.B. Owning and running cost per hour (excluding fuel cost) is shs 153772/1500 = shs 102.50/h. Fuel cost depends on road grades.

Appendix 11. The cost of owning and running different Caterpillar machines per hour.

Model of machine:	D4E bulldozer	D6D bulldozer	120G grader	215 excavator
Flywheel power (kW):	56	104	93	67
Purchase price (shs):	821 145	1 625 569	1 347 490	1 850 885
Estimated life span (Years):	8	8	8	8
Estimated machine hours/year:	1 000	1 000	1 000	1 000
Salvage value	10 %	10 %	10 %	10 %
Fixed costs/h				
Depreciation:	92.40	182.90	151.60	208.20
Interest (8 % of AAI):	39.80	78.85	65.35	89.75
Insurance (3 % of purchase price):	24.65	48.75	40.40	55.50
Taxes (0.7 % of purchase price):	5.75	11.40	9.45	12.95
Sub-total	162.60	321.90	266.80	366.40
Variable costs/h				
Maintenance and repair:	92.40	182.90	151.60	208.20
Fuel:	45.00	130.00	75.00	85.00
Oil and lubricants:	6.75	12.60	11.25	8.10
Driver:	15.80	15.80	15.80	15.80
Helper:	10.10	10.10	10.10	10.10
Sub-total	170.05	351.40	263.75	327.20
Grand total (shs/h)	332.65	673.30	530.55	693.60

Appendix 12. A summarized procedure for computing hauling costs over the designed 6 % road grade based on equation (19).

Section No.	Name of section	Lk (m)	Gradient (%)	Ct (shs/h)	S = (SLk + ELk)/2 (km/h)	(V)(S) (m^3)(km/h)	(Ct)(Lk)/(V)(S) (shs m/m^3km)
01	T1	60	6	167.00	38.8	271.6	36.90
02	C1	20	6	151.00	30.0	210.0	14.40
03	T2	30	6	155.50	36.7	256.9	18.15
04	C2	30	6	157.00	33.0	231.0	20.40
05	T3	70	6	165.50	37.6	263.2	43.90
06	C3	30	6	136.75	21.0	147.0	27.90
07	T4	120	6	162.50	36.3	254.1	76.75
08	C4	30	6	118.75	10.0	70.0	50.90
09	T5	20	6	121.25	11.5	80.5	30.10
10	C5	30	6	123.75	13.0	91.0	40.80
11	T6	30	6	130.25	17.0	119.0	32.85
12	C6	20	6	137.50	21.0	147.0	18.70
13	T7	20	6	141.00	23.5	164.5	17.20
14	C7	20	6	143.50	25.0	175.0	16.40
15	T8	20	6	141.00	23.5	164.5	17.20
16	C8	40	6	136.75	21.0	147.0	37.20
17	T9	20	6	136.00	20.0	140.0	19.45
18	C9	20	6	133.75	19.0	133.0	20.10
19	T10	210	6	162.25	38.7	270.9	125.80
20	C10	20	6	127.00	16.0	112.0	26.70
21	T11	20	6	126.00	14.5	101.5	24.80
22	C11	30	6	123.50	13.0	91.6	40.70
23	T12	30	6	121.25	11.5	80.5	45.20
24	C12	30	6	118.75	10.0	70.0	50.90
25	T13	100	6	160.50	35.0	245.0	65.50
26	C13	60	6	118.75	10.0	70.0	101.80
27	T14	20	6	126.75	15.5	108.5	[illegible]
28	C14	20	6	136.75	21.0	147.0	18.60
29	T15	130	5	158.40	38.4	268.8	76.85
Total							1139.50

Appendix 13. A summarized procedure: for computing hauling costs over the designed 10 % grade based on equation (19).

Section No.	Name of section	Lk (m)	Gradient (%)	Ct (shs/h)	S = (SLk + ELk)/2 (km/h)	(V)(S) (m^3)(km/h)	(Ct)(Lk)/(V)(S) (shs m/m^3km)
01	T1	60	6	167.00	38.8	271.6	36.90
02	C1	20	8	162.80	30.0	210.0	15.50
03	T2	30	7	165.45	33.6	235.2	21.10
04	C2	30	7	165.00	33.0	231.0	21.40
05	T3	70	9	182.25	35.3	247.1	51.60
06	C3	30	6	133.00	21.0	147.0	27.15
07	T4	120	7	170.00	36.3	254.1	80.30
08	C4	30	6	118.75	10.0	70.0	50.90
09	T5	20	6	121.25	11.5	80.5	30.10
10	C5	30	6	123.75	13.0	91.0	40.80
11	T6	30	9	140.50	17.0	119.0	35.40
12	C6	20	7	141.00	21.0	147.0	19.20
13	T7	20	8	149.50	23.5	164.5	18.20
14	C7	20	8	153.50	25.0	175.0	17.55
15	T8	20	5	136.25	23.5	164.5	16.55
16	C8	40	4	128.50	21.0	147.0	34.95
17	T9	20	6	135.20	20.0	140.0	19.30
18	C9	20	5	130.00	19.0	133.0	19.55
19	T10	210	5	188.50	38.0	266.0	148.80
20	C10	20	6	127.25	16.0	112.0	22.70
21	T11	20	7	129.00	14.5	101.5	25.40
22	C11	30	6	123.75	13.0	91.0	40.80
23	T12	30	5	118.00	11.5	80.5	44.00
24	C12	30	5	116.70	10.0	70.0	50.00
25	T13	100	4	146.25	34.9	244.3	59.85
26	C13	60	5	116.70	10.0	70.0	100.00
27	T14	20	5	124.50	15.5	108.5	22.95
28	C14	20	6	133.00	21.0	147.0	18.10
29	T15	130	5	158.40	38.4	268.8	76.85
Total							1165.90

Appendix 14. Uprooting stumps by use of a back hoe of the County tractor.

1. Raw data

Observation No.	Stump diameter (cm)	Stumping time (min)	Observation No.	Stump diameter (cm)	Stumping time (min)
1	32	12.1	22	46	8.0
2	35	27.5	23	32	4.0
3	22	10.1	24	37	2.0
4	29	20.5	25	30	8.3
5	30	20.2	26	37	6.5
6	30	23.0	27	38	10.7
7	30	11.6	28	32	20.1
8	22	6.0	29	28	2.1
9	35	9.5	30	15	2.0
10	17	1.2	31	12	1.5
11	22	5.0	32	33	9.4
12	30	9.0	33	27	6.6
13	26	9.5	34	25	1.6
14	15	2.5	35	22	9.0
15	41	12.0	36	28	6.3
16	23	14.5	37	26	4.7
17	25	10.1	38	28	4.9
18	25	8.5	39	28	9.3
19	30	11.5	40	20	5.2
20	15	2.0	41	20	5.2
21	44	10.0			

2. Basic statistics

Variable	Mean	Standard deviation	Coefficient of variation	Standard error
Stump D (cm)	27.8536	7.7832	27.9432	1.2155
Stumping time (min)	8.7268	6.2510	71.6297	0.9762

3. Analysis of variance table (AOV Table)

Source	Df	Mean square	F value
TOTAL	40		
REGR	1	277.4668	8.42
Stump D	1	277.4668	8.42
Residual	39	32.9624	

Appendix 14. (Continued)

Null hypothesis: $b1 = 0$

Tabulated $F(1,39)$ value at 1 % level = 7.33 < 8.42. Thus, the null hypothesis is rejected.

4. Regression co-efficients

Variable	Standard format	Standard error	R squared
CONSTANT	-.6986	3.370	
Stump D	.3384	0.117	0.177

$\therefore$ y = -0.699 + 0.338 Stump D

Appendix 15(a). An optimal solution for the 6 % grade design with 30 m interval lengths.

Supply areas	Demand areas									Slack vol. (m^3)	Total supply (m^3)	u_i
	D01	D02	D03	D04	D05	D06	D07	D08	D09			
S01	60 ([illegible])	30 ([illegible])								0 (119.7)	202.0	+0
S02	90 +30	60 +30								0 ([illegible])	591.7	+0
S03										0 (475.4)	475.4	+0
S04										0 ([illegible])	551.8	+0
S05										0 ([illegible])	665.8	+0
S06										0 (846.9)	846.9	+0
S07										0 (1010.6)	1010.6	+0
S08										0 (951.1)	951.1	+0
S09										0 (736.4)	736.4	+0
S10										0 (702.7)	702.7	+0
S11										0 (842.4)	842.4	+0
S12										0 (986.5)	956.5	+0
S13										0 (1027.7)	1027.7	+0
S14										0 (1106.4)	1106.4	+0
S15										0 (1305.9)	1305.9	+0
S16										0 (1395.8)	1395.8	+0
S17										0 (1124.9)	1124.9	+0
S18										0 (878.7)	878.7	+0
S19										0 (830.5)	830.5	+0
S20										0 (745.0)	745.0	+0
S21										0 (998.3)	998.3	+0
S22										0 (831.7)	831.7	+0
S23										0 (757.0)	757.0	+0
S24										0 (672.5)	672.5	+0
S25										0 (612.0)	612.0	+0
S26										0 (595.0)	595.0	+0
S27										0 ([illegible])	694.5	+0
S28										0 (749.9)	749.9	+0
S29										0 (510.4)	510.4	+0
S30			90 +0	120 +0	240 +0	270 +0	300 (126.4)	330 (53.7)	350 (12.0)	0 (278.9)	471.0	+0
S31			60 +0	90 (149.3)	210 +0	240 (115.2)	270 (55.9)	300 +0	330 +0	0	450.0	-30
S32			30 (107.6)	60 (132.7)	180 +0	210 +0	240 +0	270 +0		0	240.3	-60
S33			30 +90	30 +60	90 (10.0)	120 +0	150 +0	180 +0		0	10.0	-150
S34			90 +180	60 +120	60 (130.1)	90 (231.0)	120 +0	150 +0		0	361.0	-180
S35				20 +210	30 (58.9)	60 +0	90 +0	120 +0		0	58.9	-210
Total Demand (m^3)	46.3	36.0	107.6	202.0	199.0	346.2	311.9	53.7	12.0	22811.8	24796.6	
V_j	+60	+30	+90	+120	+240	+270	+300	+330	+350	+0		

Appendix 15(b). An optimal solution for the 6 % grade design with 60 m interval lengths.

Supply areas	Demand areas D01	D02	D03	D04	D05	Slack vol. (m^3)	Total supply (m^3)	u_i
S01	60 86.5	960 +840				0 507.2	593.7	+60
S02						0 1027.1	1027.1	+0
S03						0 1518.7	1518.7	+0
S04						0 1961.7	1961.7	+0
S05						0 1439.1	1439.1	+0
S06						0 1828.8	1828.8	+0
S07						0 2134.1	2134.1	+0
S08						0 2701.8	2701.8	+0
S09						0 2003.6	2003.6	+0
S10						0 1575.5	1575.5	+0
S11						0 1830.0	1830.0	+0
S12						0 1429.5	1429.5	+0
S13						0 1207.0	1207.0	+0
S14						0 1444.3	1444.3	+0
S15		20 192	240 +0	300 +0	305 +0	0 789.4	981.4	+60
S16		60 197.5	180 140.1	240 287.0	290 65.7	0	690.3	+0
S17		60 +120	60 +0	120 370.9	170 +0	0	370.9	-120
Total Demand (m^3)	*86.5*	*389.5*	*140.1*	*657.9*	*65.7*	*23397.8*	*24737.5*	
v_j	+0	+60	+180	+240	+290	+0		

Appendix 15(c). An optimal solution for the 6 % grade design divided into terrain interval lengths.

Supply areas	Demand areas D01	D02	D03	D04	D05	D06	Slack vol. (m^3)	Total supply (m^3)	u_i
S01	60 62.4	970 +990					0 227.7	290.1	+60
S02							0 930.2	930.2	+0
S03							0 380.6	380.6	+0
S04							0 1518.7	1518.7	+0
S05							0 1961.7	1961.7	+0
S06							0 1439.1	1439.1	+0
S07							0 1828.8	1828.8	+0
S08							0 2134.1	2134.1	+0
S09							0 2701.8	2701.8	+0
S10							0 2003.6	2003.6	+0
S11							0 1575.5	1575.5	+0
S12							0 1830.0	1830.0	+0
S13							0 1429.5	1429.5	+0
S14							0 1207.0	1207.0	+0
S15		180 +0	220 +0	320 +0	380 +0	420 +0	0 1444.3	1444.3	+0
S16		120 +0	160 87.1	260 +0	320 80.9	360 24.0	0 789.4	981.4	+200
S17		60 389.5	100 +0	200 +0	260 300.8	300 +0	0	690.3	+140
S18		80 +160	40 +80	60 453.4	120 95.9	160 +0	0	549.3	+0
Total Demand (m^3)	62.4	389.5	87.1	453.4	477.6	24.0	23402	24896	
v_j	+0	-80	-40	+60	+120	+160	+0		

Appendix 15(d). An optimal solution for the designed 8 % grade with 60 m interval lengths.

Supply areas	Demand Areas D01	D02	D03	D04	D05	D06	Slack vol. (m^3)	Total supply (m^3)	u_i
S01	60 45.5	240 +180					0 316.9	362.4	+0
S02	120 +60	180 +120					0 441.1	441.1	+0
S03		120 +60					0 327.3	327.3	+0
S04		60 186.0	300 +240				0 154.5	340.5	+0
S05		60 +0	180 +120				0 96.0	96.0	+0
S06		120 +60	120 +60				0 203.0	203.0	+0
S07			60 74.6	300 +240			0 137.5	212.1	+0
S08			60 +0	180 +120			0 155.4	155.4	+0
S09			120 +60	120 +60			0 77.5	77.5	+0
S10			180 +120	60 +0	300 +240		0 21.6	21.6	+0
S11				60 12.3	180 +120		0 73.2	85.5	+0
S12				120 +60	120 +60		0 0.9	0.9	+0
S13				180 +120	60 128.7	240 +190	0	128.7	+0
S14					60 110.5	120 +70	0 275.2	385.7	+0
S15					120 +60	60 +10	0 166.2	166.2	+0
S16					230 +170	50 26.1	0 45.0	71.1	+0
Total Demand (m^3)	45.5	186.0	74.6	12.3	239.2	26.1	2497.6	3081.3	
V_j	+60	+60	+60	+60	+60	+50	+0		

Appendix 15(e). An optimal solution for the designed 8 % grade divided into terrain interval lengths.

Supply areas	Demand areas												Slack vol. (m^3)	Total supply (m^3)	u_i
	D01	D02	D03	D04	D05	D06	D07	D08	D09	D10	D11	D12			
S01	50 30.6	270 +190											0 160.9	191.5	+0
S02	110 +60	210 +130											0 451.4	451.4	+0
S03	170 +120	150 +70											0 347.1	347.1	+0
S04	230 +180	90 +10											0 394.4	394.4	+0
S05		45 55.5	285 +235										0	55.5	-35
S06		45 34.2	195 +145										0	34.2	-35
S07		80 93.0	160 +110										0 67.1	160.1	+0
S08		130 +50	110 +60										0 174.6	174.6	+0
S09			50 87.6	175 +145									0 81.4	169.0	+0
S10			50 +0	75 +45									0 155.4	155.4	+0
S11				30 13.0	35 +20								0 77.5	90.5	+0
S12				30 +0	15 14.3	50 +25							0 2.2	16.5	+0
S13				50 +20	40 +25	30 +5	90 +50						0 0.4	0.4	+0
S14						25 8.7	35 13.2	115 +85					0	21.9	+0
S15						100 +70	40 19.0	40 +5	150 +110				0 120.9	139.9	+5
S16							10 +75	30 89.4	80 +40				0 26.1	115.5	+0
S17								70 +40	40 40.4	75 +25			0 54.1	94.5	+0
S18								110 +80	80 +40	45 317.3	105 +155		0 106.9	504.2	+0
S19									135 +95	100 +55	50 3.5	80 +55	0 162.8	166.3	+0
S20											55 +5	25 5.6	0 58.9	64.5	+0
S21												45 +20	0 12.9	12.9	+0
Total Demand (m^3)	30.6	182.7	87.6	13.0	14.3	8.7	32.2	89.4	40.4	317.3	3.5	5.6	2434.9	3260.2	
V_j	+50	+80	+50	+30	+15	+25	+35	+30	+40	+40	+50	+25	+0		

Appendix 15(f). An optimal solution for the 10 % grade with 30 m interval lengths.

Supply areas	Demand areas																		Slack vol (m^3)	Total supply (m^3)	u_i
	D01	D02	D03	D04	D05	D06	D07	D08	D09	D10	D11	D12	D13	D14	D15	D16	D17	D18			
S01	60 (63.5)	30 +0																	0 (20.9)	84.4	-30
S02	90 +30	60 (61.5)	210 (73.0)	240 +0	270 +0	450 +0													0 (49.7)	204.2	+0
S03	120 +0	90 +0	180 +0	210 (87.8)	240 +0	420 +0					870 +0	900 +0	930 (134.3)	960 +60					0	221.9	-30
S04	150 +0	120 +0	150 +0	180 +0	210 (79.5)	390 +0					840 +0	870 (78.1)	900 +0						0	157.6	-60
S05	180 +0	150 +0	120 +0	150 (85.8)	180 +0	360 +0													0	85.8	-90
S06	210 +0	180 +0	90 +0	120 (149.0)	150 +0	330 +0	360 +0												0	149.0	-120
S07	240 +0	210 +0	60 (88.6)	90 (103.7)	120 (2.7)	300 +0	330 +0												0	185.0	-150
S08	270 +0	240 +0	30 (91.2)	60 +0	90 +0	270 +0	300 +0												0	91.2	-180
S09				60 180	30 (24.2)	150 (59.6)	180 +0												0	83.8	-300
S10					60 +120	120 +0	150 (131.8)	180 +0											0	131.8	-330
S11						90 +0	120 (20.8)	150 (102.3)	450 +0										0	123.1	-360
S12						60 +0	90 (188.4)	120 +0	420 +0										0	188.4	-390
S13						30 +0	60 (126.6)	90 +0	390 +0	420 +0	480 +0	510 (26.2)	540 +0						0	152.8	-420
S14							60 +0	30 +0	270 +0	300 (33.2)	360 (9.7)	390 (27.6)	450 +0						0	70.5	-540
S15								60 +30	240 +30	270 +30	330 +30	360 +30	390 (75.6)						0	75.6	-570
S16											300 +0	330 (16.7)	360 +0						0	16.7	-600
S17											270 +0	300 +0	330 (48.2)						0	48.2	-630
S18											240 +0	270 (21.2)	300 +0						0	21.2	-660
S19											210 +0	240 (15.0)	270 +0						0	15.0	-690
S20											180 +0	210 (20.8)	240 +0						0	20.8	-720
S21											150 (56.9)	180 +0	210 +0						0	56.9	-750
S22											120 +0	150 (94.6)	180 +0						0	94.6	-780
S23											30 +30	60 (37.5)	90 +0						0	37.5	-870
S24												90 +60	60 (177.6)	30 (92.3)	60 (78.4)	90 (8.2)	120 (69.9)		0	424.8	-900
S25													90 +60	60 +60	30 +0	60 (105.1)	90 +0	140 +60	0	105.1	-930
S26															90 +120	60 +60	30 (17.0)	20 (12.8)	0	29.8	-990
Total Demand (m^3)	63.5	81.5	252.8	426.3	106.4	59.6	330.8	102.3	117.2	33.2	66.6	337.7	435.7	92.3	78.4	113.3	86.9	12.8	70.6	2865.7	
v_j	+90	+60	+210	+240	+270	+450	+480	+510	+810	+840	+900	+930	+960	+930	+960	+990	+1020	+1010			

Appendix 15(g). An optimal solution for the 10 % grade with 60 m interval lengths.

Supply areas	Demand areas D01	D02	D03	D04	D05	D06	D07	D08	D09	D10	D11	Slack vol. (m^3)	Total supply (m^3)	u_i
S01	60 145.0	240 32.9							1080 +0	1140 76.6	1190 +0	0 34.1	288.6	+1020
S02	120 +240	180 329.0	240 5.5	420 +0	480 5.4			900 39.6	1020 +0	1080 +0		0	379.5	+960
S03		120 186.6	180 +0	360 +0	420 +0							0	186.6	+900
S04		60 +120	120 +120	180 286.2	360 +120							0	286.2	+720
S05		120 +240	60 +120	120 +0	180 +0	480 +0	540 +0	600 254.3	720 +60			0	254.3	+660
S06			120 +240	60 103.0	120 32.2	420 166.9	480 +0	540 +0	660 +0			0	302.1	+600
S07				120 +240	60 +120	240 +0	300 +0	360 152.9	480 +0			0	152.9	+420
S08					120 +240	180 +0	240 +0	300 66.4	420 +0			0	66.4	+360
S09						120 +0	180 +0	240 35.8	360 +0			0	35.8	+300
S10						60 +0	120 59.3	180 242.7	300 +0	360 +60	410 +60	0	302.0	+240
S11						180 +360	120 +240	60 +120	60 10.3	120 275.0	170 18.0	0	303.5	+0
Total Demand (m^3)	145.0	548.5	5.5	389.2	37.6	166.9	59.3	791.7	10.3	351.6	18.0	34.1	2557.7	
V_j	-960	-780	-720	-540	-480	-180	-120	-60	+60	+120	+170	+0		

Appendix 15(h). An optimal solution for the 10 % grade divided into terrain interval lengths.

Supply areas	Demand areas											Slack vol. (m^3)	Total supply (m^3)	u_i
	D01	D02	D03	D04	D05	D06	D07	D08	D09	D10	D11			
S01	50 93.9	160 +10										0	93.9	-55
S02	105 1.7	205 11.2							1065 56.5	1125 122	1165 12.8	0 163.0	387.2	+0
S03		155 174.8	195 +0						1015 +0	1075 +0	1115 +0	0	174.8	-50
S04		100 308.1	140 +0	345 +0								0	308.1	-105
S05		50 68.9	90 78.6	295 +0								0	147.5	-155
S06			40 +80	165 205.3	210 +0			660 +0				0	205.3	-285
S07			100 +120	105 91.3	150 +0	440 11.6	540 103.3	600 76.1	720 +0			0	282.2	-345
S08			155 +230	50 128.1	95 106.8	385 +0	485 +0	545 +0	665 +0			0	234.9	-400
S09				95 +90	50 +100	240 +0	340 +0	400 37.3	520 137.7			0	175.0	-545
S10						180 61.2	280 +0	340 +0	460 +0			0	61.2	-605
S11						120 +0	220 27.0	280 +0				0	27.0	-665
S12							160 61.3	220 +0				0	61.3	-785
S13							50 61.3	110 +0				0	61.3	-835
S14							120 +120	60 607.8				0	607.8	-885
Total Demand (m^3)	95.6	563.0	78.6	424.7	106.8	72.8	272.8	721.2	194.2	122.0	12.8	163.0	2827.5	
V_j	+105	+205	+245	+450	+495	+785	+885	+945	+1065	+1125	+1165	+0		

Appendix 16. Hauling murram from the murram pit to the road site by use of Isuzu 7 ton trucks.

Observation No.	DISTKM (km)	TRAVLD (min)	UNLODt (min)	TRAVEM (min)	LOADGt (min)	LOADcM (m^3)	TOTALt (min)
1	2.0	7.2	2.0	6.5	3.9	2.6	19.6
2	2.0	8.6	2.0	7.0	5.4	2.6	23.0
3	2.0	7.0	1.5	6.3	3.5	2.6	18.3
4	2.0	7.0	1.6	6.5	4.7	2.6	19.8
5	2.1	8.5	2.5	6.5	3.7	2.6	21.2
6	2.0	6.5	2.0	6.6	3.5	2.6	18.6
7	2.0	7.0	2.2	6.3	2.5	2.6	18.0
8	2.1	7.2	2.0	6.6	3.0	2.6	18.8
9	2.2	7.2	1.8	6.6	3.7	2.6	19.3
10	2.1	7.6	1.0	6.9	3.5	2.6	19.0
11	2.2	7.8	1.4	7.0	6.0	2.6	22.2
12	2.1	7.2	5.3	6.1	4.0	2.6	22.6
13	2.1	7.4	4.0	6.5	2.5	2.6	20.4
14	2.1	7.5	2.2	6.5	4.1	2.6	20.3
15	2.1	7.3	2.1	6.6	6.2	2.6	22.2
16	2.1	7.0	1.9	6.5	3.2	2.6	18.6
17	2.2	7.1	16.0	6.4	3.5	2.6	33.0
18	2.1	6.8	6.2	6.5	4.7	2.6	24.2
19	2.1	7.0	14.0	6.8	4.6	2.6	32.4
20	2.2	7.0	15.7	6.5	10.7	2.6	39.9
21	2.2	7.3	2.4	6.8	3.9	2.6	20.4
22	2.2	6.5	3.2	6.6	4.2	2.6	20.5
23	2.2	7.2	1.9	6.6	4.8	2.6	20.5
24	2.2	7.3	5.0	6.7	3.8	2.6	22.8
25	2.2	8.1	1.5	6.5	3.5	2.6	19.6
26	2.2	7.0	1.7	6.7	4.0	2.6	19.4
27	2.2	7.0	2.5	7.0	3.6	2.6	20.1
28	2.2	7.5	2.7	6.5	3.8	2.6	20.5
29	2.3	6.8	1.9	7.1	3.5	2.6	19.3
30	2.3	7.8	2.0	7.2	4.7	2.6	21.7
31	2.3	7.5	1.9	6.8	4.4	2.6	20.6
32	2.3	7.8	2.0	7.0	4.0	2.6	20.8
33	2.4	8.2	2.0	7.2	4.0	2.6	21.4
34	2.3	7.7	4.1	7.2	3.7	2.6	22.7
35	2.4	8.0	1.7	7.2	4.4	2.6	21.3
36	2.3	7.5	1.9	7.3	3.6	2.6	20.3
37	2.3	8.3	2.1	7.4	3.5	2.6	21.3
38	2.4	8.0	1.4	8.1	3.2	2.6	20.7
39	2.3	8.1	1.5	7.4	3.5	2.6	20.5
40	2.4	8.0	1.8	7.5	3.0	2.6	20.3
41	2.4	8.2	2.5	7.9	3.3	2.6	21.9
42	2.5	7.6	2.9	7.0	4.0	2.6	21.5
43	2.4	7.5	4.2	7.4	4.2	2.6	23.3
44	2.5	7.7	2.3	7.7	4.5	2.6	21.7
45	2.4	7.8	2.3	7.5	3.3	2.6	20.9
46	2.4	8.0	4.0	7.5	4.9	2.6	24.4
47	2.5	8.1	2.3	7.5	3.4	2.6	21.3
48	2.5	8.0	3.0	7.5	3.7	2.6	22.2
49	2.5	8.2	1.8	7.6	3.4	2.6	21.0
50	2.5	8.0	1.5	7.4	3.3	2.6	20.2
51	2.5	8.0	1.5	7.7	3.7	2.6	20.9
52	2.4	7.0	7.5	5.8	3.7	2.6	25.0
53	2.5	9.3	3.7	8.0	3.7	2.6	24.7
54	2.5	9.0	3.4	7.8	4.0	2.6	24.2
55	2.5	9.0	3.0	8.4	3.4	2.6	23.8
56	2.5	9.3	2.2	8.4	3.4	2.6	23.3
57	2.6	9.0	2.5	8.3	3.6	2.6	23.4
58	2.6	9.5	1.5	8.0	4.5	2.6	23.5
59	2.6	7.9	1.6	8.2	4.0	2.6	21.7
60	2.7	9.6	2.0	8.2	2.8	2.6	22.6
61	2.6	8.7	4.0	7.9	3.0	2.6	23.6
62	2.6	8.2	3.0	7.8	4.4	2.6	23.4
63	2.7	8.5	2.1	7.8	4.7	2.6	23.1
64	2.7	9.7	1.5	7.8	4.5	2.6	23.5
65	2.7	9.3	2.0	8.6	3.8	2.6	23.7
66	2.7	9.7	1.6	8.5	3.5	2.6	23.3
67	2.8	9.5	2.0	10.5	4.3	2.6	26.3
68	2.8	9.8	2.9	8.5	4.1	2.6	25.3
69	2.9	10.6	3.8	9.7	3.8	2.6	27.9
70	2.8	10.3	2.5	9.3	4.0	2.6	26.1
71	3.0	10.7	6.2	9.4	4.0	3.0	30.3
72	2.9	11.0	4.0	9.6	5.0	3.0	29.6
73	2.9	11.2	2.9	9.7	4.1	3.0	27.9
74	2.9	11.2	3.0	10.7	3.8	3.0	28.7
75	3.1	11.3	2.4	10.8	4.3	3.0	28.8
76	3.0	11.2	2.5	10.2	3.5	3.0	27.4
77	3.1	11.2	2.0	10.7	3.6	3.0	27.9
78	3.1	11.0	2.8	9.8	4.2	3.0	27.8
79	3.1	10.8	5.5	10.2	2.6	3.0	29.1
80	3.1	11.5	3.7	10.0	4.0	3.0	29.2
81	3.2	10.6	2.5	10.3	3.0	3.0	26.4
82	3.1	11.3	2.0	10.8	3.8	3.0	27.9
83	3.1	11.0	5.0	10.5	4.0	3.0	30.5
84	3.1	9.2	3.0	10.1	4.0	3.0	26.3
85	2.6	11.2	3.5	9.7	4.0	3.0	28.4
86	2.2	8.8	4.0	7.3	3.8	3.0	23.9

Appendix 17. Statistical and regression analysis on road murraming data.

1. Considering only one independent variable

Dependent variable : TOTALt
Independent variable: DISTKM

1.1 Basic statistics

Variable	N	Mean	Standard deviation	Coefficient of variation
DISTKM	86	2.467	0.336	13.641
TOTALt	86	23.417	3.924	16.756

1.2 AOV Table

Source	Df	Mean square	F value
TOTAL	85		
REGR	1	445.139	43.30
DISTKM	1	445.139	43.30
RESID	84	10.281	

R squared = 0.340

Tabulated F(1,84) value at 1 % = 6.960. Since this value is less than 43.30, the null hypothesis H0 = b1 is rejected.

1.3 Regression coefficients

Variable	Standard format
CONSTANT	6.641
DISTKM	6.799

y = 6.641 + 6.799 DISTKM

2. Considering two independent variables

Dependent variable : TOTALt
Independent variables: DISTKM, LOADcM

2.1 Basic statistics

Variable	N	Mean	Standard deviation	Coefficient of variation
TOTALt	86	23.417	3.924	16.756
DISTKM	86	2.467	0.337	13.601
LOADcM	86	2.674	0.156	5.854

Appendix 17. (Continued)

2.2 AOV Table

Source	Df	Mean square	F value
TOTAL	85		
REGR	2	256.958	26.83
DISTKM	1	445.139	46.49
LOADcM	1	68.777	7.18
RESIDUAL	83	9.576	

R squared = 0.393

Tabulated F(2,83) = 4.88 at 1 % level. Since it is less than 26.83, the null hypothesis HO:b1 = b2 = 0 is rejected.

2.3 Regression coefficients

Variable	Standard format
CONSTANT	-8.609
DISTKM	4.056
LOADcM	8.233

y = -8.609 + 4.056 DISTKM + 8.233 LOADcM