Western Australian School of Mines

Selection Criteria
For
Loading and Hauling Equipment -
Open Pit Mining Applications

Volume 1

Raymond J Hardy

This Thesis is presented for the Degree
Of
Doctor of Philosophy
Of
Curtin University of Technology

July 2007

DECLARATION

This thesis does not contain material previously accepted for the award of any other degree or diploma in any university. To the best of my knowledge and belief, where this thesis contains material previously published by any other person, due acknowledgement has been made.

………………………..

R J Hardy
9 July 2007

THESIS PRESENTATION

Scope of the research, breadth of topics and resulting extensive documentation required presentation of the thesis in two volumes.

Volume 1 **Selection Criteria Issues, Productivity and Analysis –** Chapters 1 to 3 included in Volume 1; and

Volume 2 **Selection Process, Costs, Conclusions and Recommendations –** Chapters 4 to 7 - plus References, Appendices and Supplementary Information – included in Volume 2.

ABSTRACT

Methods for estimating productivity and costs, and dependent equipment selection process, have needed to be increasingly reliable. Estimated productivity and costs must be as accurate as possible in reflecting actual productivity and costs experienced by mining operations to accommodate the long-term trend for diminishing commodity prices,

For loading and hauling equipment operating in open pit mines, some of the interrelated estimating criteria have been investigated for better understanding; and, consequently, more reliable estimates of production and costs, also more effective equipment selection process.

Analysis recognizes many of the interrelated criteria as random variables that can most effectively be reviewed, analyzed and compared in terms of statistical mathematical parameters.

Emphasized throughout is the need for management of the cyclical loading and hauling system using conventional shovels/excavators/loaders and mining trucks to sustain an acceptable “rhythm” for best practice productivity and most-competitive unit-production costs.

Outcomes of the research include an understanding that variability of attributes needs to be contained within acceptable limits. Attributes investigated include truck payloads, bucket loads, loader cycle time, truck loading time and truck cycle time.

Selection of “ultra-class” mining trucks (≥ 290 -tonne payload) and suitable loading equipment is for specialist mining applications only. Where local operating environment and cost factors favourably supplement diminishing cost-benefits of truck scale, ultra-class trucks may be justified. Bigger is not always better – only where bigger can be shown to be better by reasons in addition to the modest cost benefits of ultra-class equipment.

Truck over-loading may, to a moderate degree, increase productivity, but only at increased unit cost. From a unit-cost perspective it is better to under-load than over-load mining trucks.

Where unit production cost is more important than absolute productivity, under-trucking is favoured compared with over-trucking loading equipment.

Bunching of mining trucks manifests as a queuing effect – a loss of effective truck hours. To offset the queuing effect, required productivity needs to be adjusted to anticipate "bunching inefficiency". The "basic number of trucks" delivered by deterministic estimating must provide for bunching inefficiency before application of simulation applications or stochastic analysis is used to determine the necessary number of trucks in the fleet.

In difficult digging conditions it is more important to retain truck operating rhythm than to focus on achieving target payload by indiscriminately adding loader passes. Where trucks are waiting to load, operational tempo should be restored by sacrificing one or more passes. Trucks should preferably be loaded by not more than the nominal (modal) number plus one pass.

The research has:

- Identified and investigated attributes that affect the dispersion of truck payloads, bucket loads, bucket-cycle time, loading time and truck-cycle time.
- The outcomes of the research indicate a need to correlate drilling and blasting quality control and truck payload dispersion. Further research can be expected to determine the interrelationship between accuracy of drilling and blasting attributes including accuracy of hole location and direction.
- Preliminary investigations indicate a relationship between drill-and-blast attributes through blasting quality control to bucket design, dimensions and shape; also discharge characteristics that affect bucket cycle time that needs further research.

ACKNOWLEDGEMENT

"No man is an Island, entire of itself." from *Devotions* by John Donne

Soon after setting out on the research journey recorded in this thesis the words of John Donne, above, were brought home with undeniable impact. In addition to the knowledge acquired during a working lifetime in the mining industry, civil construction and associated vocations the author still depended on many others to safely and satisfactorily reach journey's end. Generous support and contribution of many friends, associates and casual acquaintances in industry and academia that have assisted along the way are acknowledged with sincere thanks and great appreciation.

Every reasonable effort has been made to correctly acknowledge the owners of copyright material. The author would be pleased to hear from any copyright owner incorrectly, or not, acknowledged.

Support and assistance of the following are especially acknowledged:

Associate Professor Dr Emmanuel Chanda of the WA School of Mines (WASM) Mining Department, for his continued encouragement, guidance and supervision; also at WASM, Dr Peter Lilley (formerly Director of WASM, now with CSIRO, Perth), Mahinda Kuruppu and Clive Workman-Davies assisted from time to time.

Western Premier Coal Limited, Collie, WA – within the Wesfarmers Limited group of companies - provided funding for WASM research projects that partially supported the research work described by the thesis. That support is gratefully acknowledged.

At WesTrac, Caterpillar dealer located in South Guildford Western Australia (franchised some 80 years as the first Cat dealer outside USA, more recently expanding into New South Wales and China), personnel that have assisted with information and operating data are too numerous to individually acknowledge. To WesTrac staff in general my sincere thanks.

Individual acknowledgements must be afforded Jim Walker, CEO, for opening the door to the huge knowledge base acquired by WesTrac and possessed by its South Guildford staff; Fred Haynes for his interest, support and supply of weighing data in the early stages; Tony Power who kept the author in touch with technical advances and developments in the mining industry; Adrian Clement who gave his time generously, provided access to VIMS and Minestar data to support the research, also for demonstrating the exciting benefits and promise of these real-time data access systems and Peter Perich for showing the way to market equipment prices and comments on written down values of mining equipment. More recently Quentin Landre has supplied Caterpillar equipment specification material; also providing information to enable any WASM student to access specifications and performance data vital to their understanding of mobile mining equipment.

Particularly I am indebted to WesTrac for access to FPC simulation software and other mining equipment applications over the years; also for a lifetime supply of Performance Handbooks and related specification material.

To David Rea of Caterpillar, many thanks for the insight afforded the author on VIMS and related matters; also to Caterpillar Global Mining generally for access to technical material made available at recent Miners' Forum's in Western Australia.

Other OEM - dealership contacts also contributed including: from Minepro, P & H dealer, Noel Skilbeck and staff; from Komatsu, Alan Costa; from Liebherr, Henning Kloeverkorn, Merv. Laing and Duncan Hooper; from Terex Mining – O & K, Craig Watson and Andy McElroy; from Hitachi, Eric Green – to all many thanks and best wishes to those contacts who have moved on in their career paths.

Tony Cutler of OTRACO provided comprehensive information on large earthmover tyres, especially for ultra-class trucks; also Hans Weber of Michelin provided technical information on tyre application and management for large mining trucks.

To the many other personal and commercial contacts in and associated with the mining industry that have steadily increased the volume of knowledge and understanding available to the author, knowledge that was tapped to progress the research, the author extends heartfelt thanks and appreciation.

Last but not least the author acknowledges the contribution of his partner in life, Heide, who supported the research with an interesting mixture of tolerance and vigorous encouragement including many exhortations to "get on with it" – when the author was experiencing adequate strength of spirit but an inhibiting weakness of flesh.

Raymond J Hardy, July 2007.

VOLUME 1

TABLE OF CONTENTS

VOLUME 2

TABLE OF CONTENTS

APPENDICES

APPENDED TABLES

MATHEMATICAL PRINCIPLES AND NOTES 471

DISTRIBUTION TESTING 501

TABLES AND FILES ON CD (INSIDE BACK COVER)

SELECTED TABLESS

Listed below are "Selected Tables" that readers may find of practical use or may serve as a foundation for further research. For convenience they are provided on a CD inside the back cover of Volume 2.

Table Number	Table Title	Page Number For Hard Copy
3.4	Loader Efficiency - η_{LE}% - and Number of Truck-Loading Passes	Vol. 1, p 50
3.57	Productivity Criteria for Sacrificing Bucket Loads	Vol. 2, p 459
3.59	Cycle Time Calculator, 4 Parts	Vol. 2, pp 460 - 463
3.70 and 3.71	Probability Tables	Vol. 2, pp 464, 465
5.10	Cost Criteria for Sacrificing Bucket Loads	Vol. 2, p 467

SUPPLEMENTARY INFORMATION

Three papers published during the tenure of the research have been referred to in the text. Copies of files are on the CD inside the back cover of Volume 2.

Hardy#1, Raymond J, **(2003)** *Four-Pass Loading, Must Have Or Myth?* Fifth Large Open Pit Mining Conference, 2003, Kalgoorlie, Western Australia, The Australasian Institute of Mining and Metallurgy.

Hardy#2, Raymond J, **(2003)** *Outsource or Owner Operate,* Twelfth International Symposium on Mine Planning and Equipment Selection, Kalgoorlie, Western Australia, Australasian Institute of Mining and Metallurgy.

Hardy#3, Raymond J, **(2005)** *Outsource versus Owner Operate*, IIR Contract Mining Conference, 2005, Perth, Western Australia. (An update of a previous paper included in references as Hardy#2, 2003).

LIST OF TABLES

LIST OF FIGURES

CHAPTER 1

INTRODUCTION

1.1 RESEARCH CONCEPTION

Traditional mobile equipment selection methods, load-and-haul cost estimating techniques and productivity forecasting have more often been artistic than scientific based on observations and experience of the author. Historically, empirical performance data for loading and hauling has been analyzed deterministically using expected or typical values. More recently application of information technology systems for control and recording performance of mining operations in real time has produced a wealth of reliable data. Availability of reliable operational data facilitates immediate management response to inefficiencies and provides instant remedies for productivity misadventure. Deterministic calculations based on empirical data adopting expected ("mean") values as inputs, tempered with intuition and refined with experienced insight have historically yielded estimates of productivity and operating costs that largely provided satisfactory management guidance. But all too often such analyses have proved to be remote from actual achievable results. Too often outcomes have not met expectancy based on traditional estimating and forecasting methods. Occasionally, performance realisation is consistent with expectancy; but it later becomes obvious that errors in data and/or logic have been compensated by fortuitous conservative calculations – a product of experience.

Scale of open pit mining operations has increased realizing unit cost benefits to accommodate a competitive market driven by steadily decreasing real-terms prices of mineral commodities. Accordingly mobile equipment capacities have been increased to provide mining operators with the ability to respond. Increasing scale of mining equipment, especially loading equipment and mining trucks, has been accompanied by unit cost benefits, especially for haulage, that diminish with increasing equipment scale. Diminishing unit-cost outcomes demand increasingly precise analysis and estimating techniques. Models adopted for analysis need to accurately represent actual operating situations. Stochastic techniques are replacing deterministic methods. More recently, dynamic-simulation computer applications are being increasingly adopted resulting in improved reliability in forecasting, production strategy development, mine planning and cost estimation.

Although modeling, productivity forecasting and cost estimating have advanced many traditional methods and simplistic techniques still being applied need to be upgraded. Improved analysis alternatives need to be identified or developed and made available for practitioners. These conclusions have been material in justifying need for research of Selection Criteria for Load and Haul Equipment for Open Pit Mining.

1.2 RESEARCH OBJECTIVES

The following research objectives were identified:

1. Broadly review open pit mining economics, productivity and unit mining costs, to establish relative contributions of individual productivity/cost components.
2. Identify significant productivity and cost drivers in loading and hauling operations in open cut mining, reviewing existing standards; or in default, to suggest suitable performance criteria.
3. Determine issues that influence productivity and unit cost estimates, which traditionally have been provided for by some form of efficiency factor, contingency allowance or the like in lieu of realistic analysis.
4. Prioritize the individual issues, research previous work by others and, if warranted, analyze the various problems working towards solutions, or to at least suggest paths to understanding of each issue.
5. Develop new techniques for improved equipment selection practices, endorse existing, or recommend some basic rules for operating the largest open pit mining equipment as well as offering management protocols and techniques for improved productivity and cost benefit.
6. Examine mining truck and loading equipment inter-actives in order to review the current philosophic trend in the open pit mining industry that “biggest isn’t always best”. Further, to consider why industry practitioners often opt for a small number of larger loading machines when an increased number of smaller loading machines of equivalent total bucket displacement will likely provide more efficient loading, greater operating flexibility and hauling

operations with improved “rhythm” - given that mine planning can make available increased working positions.

7. Review modeling research on “bunching” (the cause of queuing effects) of mining trucks and resulting effect in terms of operating efficiency, productivity and unit load and haul costs.
8. Continue on from initial research for a paper Four Pass Loading – Must Have Or Myth presented at the 5th Large Open Pit Conference, Kalgoorlie Western Australia, November 2003 – copy provided in Supplementary Information on the CD pocketed inside back cover of Volume 2 - (Hardy#1, 2003); and to further research truck payload distributions, relate bucket load distributions to payload distributions, analyzing haulage cost imposts for widely disperse payload distributions; and to examine implications for large mining trucks.
9. Analyze productivity, unit costs and efficiency of large mining trucks, including the trend for diminishing returns with increasing scale; and discuss directions for future development including limitations to current generic configurations and problems with extrapolating current designs.
10. Interpret the results of all reviews and investigation to determine criteria for reliable equipment selection, productivity and cost estimation criteria to improve management of load and haul operations and realize planned project outcomes.
11. Summarize the research results and generally provide improved understanding of the fundamentals of productivity, unit costs and equipment selection criteria for open pit mining applications.

The first stage in equipment procurement is to adopt-and-modify, or develop, a reliable process that includes investigation, estimation of quantities and required productivity, identification of complying equipment, and matching loading to hauling capacity to satisfy a pre-determined production programme. These procurement activities are the focus and objective of the research leading to interpretations and conclusions in this thesis – to provide criteria and protocols to enable:

- Pre-emption of selection of inadequate equipment or provision of excessive, redundant capacity.

- Determination of principles for conduct of a reliable procurement process to avoid undesirable outcomes.
- Assignment of accountability for definitive due diligence and investigation.
- Realization of best-practice load-and-haul productivity and cost outcomes.

1.3 HISTORICAL BACKGROUND

1.3.1 Research Formulation

The historical evolution of surface mining practices using specially designed load and haul equipment is recent when compared with development of mining methods in general. Technological advances and market demands for goods have driven an ever-increasing need for mineral commodities. Late in the 19th and early in the 20th century the burgeoning demand for energy minerals and base metals by the maturing industrial revolution provided the stimulus for mining operations of increasing scale. Mineral resources identified to meet demand for bulk commodities have trended away from the rare small-rich native-metal occurrences and narrow seams of high-quality solid fuels to lower grade/quality, more extensive deposits favouring surface access rather than traditional underground vein-mining methods.

In the context of this thesis open pit mining is a separate evolutionary development of surface mining, distinct from strip mining for coal and quarrying for basic construction materials. It has evolved rapidly from mid-twentieth century with more dramatic advances over the most recent 30 years. Compared with the low evolution of underground mining, the historically pre-eminent mining method, technological advance of open pit mining methods has been spectacular. Some of the methods using open pit load-and-haul equipment have been retrospectively adapted to underground mining operations in the thrust for increased productivity and lower mining costs.

In the light of modern technological advances, review of some time-honored open pit mining practices and procedures and equipment applications has posed unanswered questions. A desire to provide answers to these questions has been the focus of research recorded in this thesis.

Obvious issues, subjects of debate and divided opinion throughout a significant period of recent industrial experience, stand out as ready-made research topics. These include:

- Optimum number of passes of loading equipment to load mining trucks for each open pit mining operation.
- Management practices to realize the expected cost benefits from increased scale of open pit operations and equipment.
- Determination of how the distribution of payloads in trucks affects open pit mining economics.
- Understanding of loading performance drivers for optimization of loading efficiency and lowest-possible loading costs.
- How to implement control of truck payloads to an acceptable distribution range with confidence to deliver optimum mining economics and safe operations.
- Economic comparison of under -trucking or over-trucking: Is under-trucking of loading equipment by rounding down theoretical truck numbers to a digital number, more economic than rounding up to over-truck or even perfectly match intrinsic performance of trucks with loading equipment?
- Economic comparison of over-shoveling i.e., under-trucking: Is it more economic to over-shovel, i.e., under-truck, with trucks at maximum productivity; or to over-truck with shovel at maximum productivity? and the relationship of these operating states to bunching and queuing in general.
- Efficiency of shovel and truck matching: Is it more effective in terms of productivity and load-and-haul costs to have, say;
  - ➢ Two large shovels loading trucks in four passes;
  - ➢ Or three smaller shovels to load the same trucks in six passes?

Given that shovels are adequately serviced by trucks at all times and provided that mine planning can accommodate the additional working face(s):

- ➢ Is it more economical to fill trucks to target payload capacity even if more bucket passes are required?

- Or should a pass or passes, and payload, be sacrificed to maintain operational “rhythm”; and
- Circumstances that determine the adoption of either option?

These obvious questions and other production and cost issues not so obvious in planning, estimating and operating open pit mines, were prioritized to practically limit the scope of the research.

It was realized that thorough investigation of load-and-haul production and costs would determine a scope for the current research that includes all previously identified unanswered questions.

From literature research, retrospective review of case studies and consulting projects accumulated over approximately ten years, a number of what are believed to be the more significant key productivity and cost drivers have been identified, examined, analysed and interpreted. For these research topics, economic implications have been determined, decision-making criteria identified and practices recommended to achieve optimum economics of load-and-haul operations over a range of open pit-mining situations.

To limit the research to a practical scope load-and-haul operations in deep open pits have been the main focus of the research. Scope and directions taken by the research have been significantly influenced by involvement in and experience gained from the evolution of open pit mining in Western Australia.

Although influenced by Western Australian open pit mining history and practices, it is reasonable to expect the generic nature of analysis throughout, together with interpretations and conclusions of the research, will apply equally for open pit operations of similar nature elsewhere.

1.3.2 Mining Commodity Prices – The Ultimate Cost Driver

Real-terms prices of mining commodities have been steadily diminishing. Figure 1.1 demonstrates this trend for 150 years to 1995. This trend, driven by marketing competition, is believed to be continuing, albeit at reduced differential rate.

Competitive reduction of mining costs has driven the downward trend whilst sustaining profit margins, albeit also diminishing, at levels that justify investment in mining projects. The mining industry has generally accommodated revenue and cost pressures by a combination of increasing scale of projects and advancing technology. Consequently increased unit productivity demands have been met by increased scale, and enhanced performance of mining equipment.

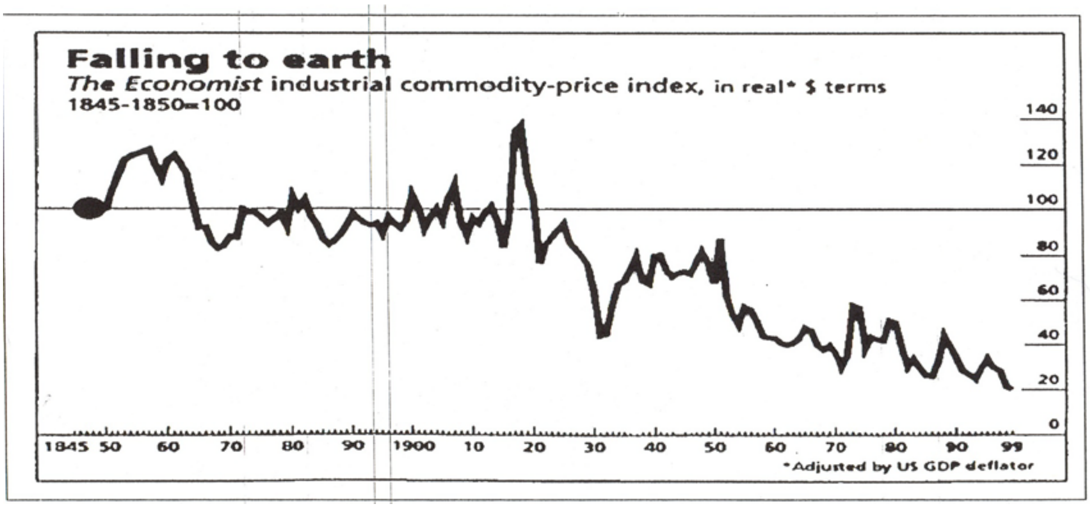


Figure 1.1 – Commodity Prices: Long Term Trend

1.3.3 Recent Surface Mining History

1.3.3.1 Heritage

Mining of commodities by mankind, necessary for survival and technological development, is an endeavour almost as old as hunting for and gathering of food and the evolving agrarian pursuits of primitive man.

Mining for metals and solid fuels developed from simple gathering and gouging for native metals and materials in weathered surface stratigraphy to following the productive stratigraphical elements laterally and in depth. Configuration of the host structures inspired the development of underground hand mining methods, techniques to persist, albeit with modest mechanization, into the early 20th century. Historical records confirm the comparative infancy of surface mining as a recognized separate mining technique.

Georgius Agricola (Agricola, 1556) does not identify surface mining, particularly open pit mining, as a separate mining method for commodity extraction. Only underground operational methods, essentially vein mining, following high grade ore values, are recorded in three stratigraphical settings:

Venae profundae – veins extending to great depth.

Venae diletatae – veins thickened or lensed out.

Venae cumulatae – combined smaller veins relatively close together – a series of conformable stringers.

James McClelland (McClelland, 1941) describes a number of iron ore and base metal surface mining operations utilizing power shovels on rails loading into rail cars. Introduction of road transport with 20-ton petrol-engined trucks and the resulting increased flexibility is noted. Steeper access ramps at 8% - 10% are identified as advantageous. By 1938, off-highway mining trucks had been introduced to the Mesabi Range iron ore mines, Minnesota, USA, both in the typical 2 – 4 wheel configuration of modern day trucks and tractor-trailer ore carriers. Track-mounted power shovels have been developed contemporaneously with evolution of the modern mining truck.

1.3.3.2 Western Australia

In early 20th century Western Australia, open pit mining (then termed "open cut" mining) typically utilized glory-hole recovery of ore mined by underhand benching of surface crown pillars as oxidized (weathered) ore sources to supplement underground gold operations. In the early 1970's gold miners benefited from a rising gold price, due mainly to IMF deregulation of the gold price of US$35 per ounce Troy and allowing the gold price to float on an open market. Open pit gold mining commenced at a greenfields deposit at Telfer in the Little Sandy Desert of the East Pilbara. From the mid 1970's through the 1980's open pit mining on the Golden Mile at Kalgoorlie, extracted surface pillars in the weathered zone above underground workings that previously could not economically or safely be mined by underground methods. In the past twenty-five years, resurgence of gold mining based on open pit mining methods was facilitated by lower-cost carbon-in-leach and carbon-in-pulp treatment of weathered (oxidized) ores. Exploitation of lower grade weathered ore remaining after underground operations, plus some greenfields low-grade developments, burgeoned and then waned. More recently the gold mining industry has consolidated to a small number of larger open pit gold mines still operating. Current open pit gold operations mine both weathered and primary (sulphide) ores requiring concentrating followed by pyrometallurgical or biological calcining prior to leaching and gold recovery.

Early coal production at Collie, Western Australia for power generation and locomotive fuel was mainly from underground mines using room-and-pillar mining methods, supplemented by small-scale open pits using conventional rope shovels and highway trucks or, at least in one instance, small track-mounted dragline where seams outcropped. When the Hebe underground mine flooded in the 1950’s (fortunately without loss of life), the Muja open pit was commenced using conventional rope shovels and trucks on the same coal measures containing eight seams with the Hebe as the basement seam. After initial box-cutting with motor scrapers to remove tertiary sediments above the coal measures, open pit mining commenced on the basement Hebe seam with the bonus of picking up seven thinner seams higher in the coal measures that were uneconomic to mine by underground methods. Open pit mining of the Muja coal measures and other seam groups within the Collie Basin, which larger mining equipment can extract economically, is continuing.

Lifting of the embargo on exporting of iron ore from Australia in the early 1960’s initiated the world-class open pit iron ore mines of the Pilbara region of Western Australia with large infrastructure requirements of ports, railways, towns and services that currently retain a significant market share of world iron ore production. Open pit mining techniques developed on the Mesabi Ranges, USA, were imported to Western Australia. Electric rope shovels and largest available mining trucks were introduced with the iron-ore industry keeping abreast of technology by upgrading as larger trucks and loading units became available.

1.3.3.3 Evolution of Open Pit Mining Equipment

Since the 1940’s, there has been constant development of rope shovels with introduction of hydraulic excavators, both backhoes and shovels, initially developed in Europe in the 1960’s. The earliest large mining hydraulic shovels and backhoes were introduced to Australia at Weipa, North Queensland about 1974, soon followed at Collie in 1976, when a third generation of 85-ton trucks replaced 40-ton mining trucks that had replaced smaller mining trucks introduced soon after the Muja open pit commenced operations. Currently Collie coal mines use 240-ton mining trucks and electric-powered rope shovels to load them. Equipment upscaling at Collie in the 1970’s was triggered by a practical fourfold increase in the price of oil. Return to coal-fired electric power generation became an economic necessity in Western

Australia. “Mothballed” coal-fired thermal power stations were restored as base load generation with oil-powered generation plant relegated to peak load service.

Similar parallel history of open pit mining development to that described above for Western Australia has been experienced both nationally and internationally. Development of strip coal mines in Queensland and NSW involving dragline stripping and significant load-and-haul pre-strip operations have been spectacular. Emergence of Japan as booming heavy-industry economy post WW II created unprecedented demand for coal and iron ore that was met by Pilbara iron ore and Bowen Basin coking coal.

1.3.4 Demand for Increasing Scale

A generation ago open pit mines using truck haulage were developed using equipment sourced from the construction industry. The worldwide upsurge in large infrastructure projects following World War II provided incentive for mobile equipment manufacturers to develop off-highway trucks and equipment to load them. There was also a demand for other earthmoving equipment including motor scrapers, graders, track and wheel dozers and support equipment for the major civil earthworks projects undertaken at that time. Typical developments included the Tennessee Valley Authority and Interstate Highway initiatives in USA, Snowy Mountain Hydroelectric Authority in Australia and the rebuilding of European war-affected infrastructure.

Currently original equipment manufacturers (OEM’S’s) market mining trucks and loading shovels specifically designed for open pit mining applications. Open pit mining operators have a choice of mining trucks with either mechanical drive or electric drive, both DC and AC, to 360 tons (330 tonnes) or more. These trucks can be matched to rope shovels to 1400 tonnes operating mass, bucket capacity 56 cubic metres, i.e., some 100 tonnes per bucket pass; or hydraulic excavators/shovels to 1,000 tonnes operating mass, bucket capacity 47 cubic metres, i.e., some 85 tonnes. Extrapolation of designs to even larger equipment, particularly trucks, is currently an active discussion topic throughout the mining and equipment supply industries. There will be need in the future for innovative designs that depart from the currently available mining truck configuration and physical form to achieve significant measurable improvements in performance and operating incremental cost benefits.

Scaling up mining trucks and matching loading equipment for economic and strategic reasons, has been a pervasive feature of open pit mining operations internationally and for all commodities. Market demand has been met by response from OEM’s with “ultra”-class - +300tonne - mining trucks and matching loading equipment, including ubiquitous rope shovels and hydraulic excavators, of ever-increasing scale and capacity. But an unfortunate side effect of increasing equipment scale has been diminishing incremental cost-reduction benefits.

1.3.5 Evolutionary Trends

Recent history has been a story of ever-increasing scale of open pit mining production equipment with rapid obsolescence. Upgrades by OEM’S have been frequent with major re-works of mechanical mining trucks, more fuel-efficient engines, low profile tyres with longer life potential, mine-specific design bodies that provide for lower tare and higher payload. A recent major development has been provision of AC electric wheel motors with improved torque-speed curve characteristics approaching the power transmission efficiency of mechanical drives.

The improved efficiency of AC wheel motors compared with dated DC technology can be expected to increase interest in trolley assist systems. As the current cost of diesel fuel is at levels exceeding historical costs in countries that embraced trolley assist because they were penalized by unnaturally high oil prices, e.g., South Africa when sanctioned for their apartheid policy.

Mining trucks have been increasing in scale to realize the benefits of reduced unit cost of production. Accordingly, there has been demand for loading equipment that can three or four-pass load larger trucks to yield cost savings through both expected lower unit loading costs from larger equipment and haulage cost benefit from reduced truck-loading times.

OEM’s generally offer a nominal 240 ton payload mining truck. They are now offering “ultra-class” trucks up to 360 - 400 ton payload with mechanical and AC electrical drivelines. Early “ultra-class” mining trucks used DC wheel motors. When first available AC wheel motors were supplied with “ultra-class” trucks with DC motors as an option. Currently DC motors tend to phase out at 240 ton trucks with OEM’S retroactively providing AC alternatives to DC systems down through the range of large mining trucks.

Loading equipment for four-pass loading of “ultra”-class trucks, must have effective bucket capacity of 90 to 100 tons. Allowing for practical bucket fill factors, bucket capacities must be in the range of 95 to 105 tonnes metric.

An increase in the number of passes to load a truck, say to 5, 6 or 7 will obviously tend to increase loading and total truck cycle times, reduce productivity and increase unit cost due to extra loading time and truck haulage time. However, a number of latent and indirect productivity/cost offsets tend to significantly compensate cost of increased number of passes. These latent and indirect cost offsets are identified and quantified in relative terms in Chapter 5.

The technology for autonomous truck haulage systems exists. Remote load-and-haul units are currently working in underground mines. Autonomous haulage systems are being tested for feasibility and operationally tested in open pit mines. Such systems can be expected to become commercial and available in the near future.

CHAPTER 2

RESEARCH METHODOLOGY

2.1 RESEARCH BACKGROUND

2.1.1 Uncertain and Imprecise Estimating Practices

Experience with production planning, equipment selection and cost estimation for feasibility studies, mining contracts and in-house budgets has revealed accuracy limitations of estimates and selections based on determinative processes. To reduce the risk of significant production shortfalls, and to cover the gap between determinative estimation and reality, contingency allowances, or “fudge” factors, are applied.

Traditionally contingency allowances have covered for omissions of Pareto-style practices for compiling production and economic profiles of prospects to decide whether to embrace for development or reject a possible project. Contingency allowances are not only a treatment for risk of omission in estimating. These corrective adjustments also cover for latent conditions, and possibly for lazy or inadequate due diligence in the early investigative stages of feasibility studies through to the procurement process as a prospect progresses to development.

In essence, contingency allowances in traditional estimating are expected to cover for any lack of fit or inadequacy of the conceptual model in representing actual operations and suitability of that model as a basis for productivity determination and cost estimation.

Increasing cost pressures have demanded ever-increasing precision in estimating techniques to accommodate falling commodity prices in real terms - Section 1.3.2. Traditional estimating techniques, based on simple-system models using deterministic processes manipulating discrete data are being supplemented by more sophisticated analysis of complex models applying stochastic processes and simulation techniques.

The more modern approaches to estimating forecasting and budgeting using system simulations have introduced new terminology (Banks, 1996). Such as:

- *System* – defined by Banks below – for example open pit mine production operations;
- *Entities* – mobile equipment: loaders, trucks and support equipment;
- *Attributes* – speed, capacity, breakdown rate;
- *Activities* – loading, hauling, turn and spot, turn and dump, waiting (queuing);
- *Events* – complete loading, breakdown; and
- *State Variables* – available, operating, standby, down.

> *"A system is defined as a group of objects that are joined together in some regular interaction or interdependence towards accomplishment of some purpose."* (Banks, 1996)

Systems can be discrete or continuous. Generally open pit mining operations can be considered as discrete systems. Some activities within a discrete system can be continuous, e.g., conveyor transport of materials mined with continuous-mining methods or conversion of a discrete loading activity to a continuous sub-system through a surge hopper and conveyor feeder. This research reviews open pit mining systems using shovels loading mining trucks that are taken to be discrete, and analyses and interprets activities that affect mining productivity and unit costs.

Traditional production scheduling and cost estimations are based on conceptual operations models that are simplified to a degree that is comfortable for the estimator. Variables are considered as discrete; variability is dealt with by using mean, or occasionally, modal values. Traditional calculation processes are deterministic, using expected (mean) values of empirical data. Most component inputs for analysis of the nature and significance of attributes are randomly variable distributions best analyzed stochastically.

This research examines some of the more important productivity and cost drivers by analyzing operational data recorded by real-time monitoring facilities that track activities of mining entities, i.e., shovels, trucks and supporting equipment. The benefits of data collection and presentation by dynamically programmed dispatch facilities are manifest.

Research described herein is a natural progression from limited accuracy outcomes by applying traditional simple models and deterministic processes for mining equipment selection both in terms of type and numbers. Estimates produced, forecasts created and budgets compiled for productivity and mining costs by deterministic process occasionally deliver inadequate results, even when modified by practical factors to make the analysis fit experience-based anticipated outcomes, i.e., empirical evidence. Research has enabled investigation of reasons for actual outcomes falling short of predictions in open pit mining operations.

Production and cost estimating, and operating standards, based on estimated average performance may not deliver expectancies. For example:

- Mining-truck payload dispersion over a wide range has become more important as mining trucks have increased in size. Compared with under-loaded trucks, life of driveline components, including tyres, tends to reduce, and braking distance increases, for over-loaded trucks. Increased payload dispersion, and consequent truck cycle-time variability, increases bunching effects.
- Selection of trucks and loading equipment on the basis of estimated average performance without probabilistic consideration of combined, binomially distributed availability and utilization of truck and loading equipment fleets tends to underestimate fleet numbers.
- Tendency to adhere to time-honoured standards, such as, three or four pass loading without investigation of all a-productive flow-on effects to interrelated operational functions.
- Truck-bunching effects near the "match" point, where loading and hauling capacities are practically equal, can cause loading equipment to be idle for short periods. Simply over trucking to ensure that loading production is maximized also determines collective productivity; but trucking efficiency is reduced, hauling ucosts increase and overall mining ucosts increase. Over trucking treatments include the tendency to always round up parts of trucks or loading equipment when analyzing production and costs – generally without considering the potential cost-reduction benefit of under trucking, or "over-shoveling", to maximize truck efficiency for any required productivity.
- Lack of appreciation of the diminishing cost benefits with increasing scale of loading and hauling equipment, a result of ignoring details of necessary

changes in support equipment, facilities, and management systems – especially the need for increasingly-precise management of time.

2.1.2 Questionable Standards and Measures

Adequately productive mining operations must also be efficient, and must sustain competitive commodity costs. It is important to differentiate between absolute production, i.e., the total tonnes or bank cubic metres (BCM) loaded and hauled, and productivity, i.e., the rate of production, usually, per unit of time, per unit of capacity, per unit of expense, per machine or per man-hour, and the like. Focus on absolute production alone rather than a balanced consideration of efficiency, productivity and cost is often an impediment to best management practices and equipment selection for open pit mining.

Key Performance Indicators (KPI) adopted by mining operations for management control are almost invariably productivity oriented. There appears to be confidence that optimum economic costs are a natural consequence of maximum productivity. This not necessarily so and some circumspection is justified. It is often necessary, to consider productivity and operating costs independently. Certainly increased productivity may significantly reduce estimated or budget costs. But depending on the circumstances and how productivity is increased, "abcosts" (= absolute costs, i.e., empirical costs – actually experienced and realised after the event; not relative to any unit basis such as time, production units and the like.) may stay the same or increase, albeit moderately; and "ucosts" (= unit costs, i.e., abcosts per unit of time, production, and the like.) may vary from significant decrease to significant increase. In these potentially confusing situations it is understandable that fallacious conclusions can result. It is easy to understand how technical investigators and observers can fall for the *"post hoc ergo propter hoc"* fallacy – *"after this therefore because of this"*. Paul Samuelson (Samuelson, 1958) explains this fallacy with the examples: *"The difficulty of analyzing causes when controlled experimentation is impossible is well illustrated by the confusion of the savage medicine man who thinks that a both witchcraft and a little arsenic are necessary to kill his enemy, or that only after he has put on his green robe in spring will the trees do the same."* It is important to extend any individual analysis of either productivity or costs to consideration of interrelated factors and how bottom-line economics are likely to be affected.

In support of this perception of over-focus on productivity, typical KPI (from a recent mining contract) are listed in Table 2.1 appended in Volume 2. The emphasis on production and productivity of the KPI's in Table 2.1 is obvious with only one exception in this typical list where a ucost is set as a performance standard. Certainly productivity influences costs, but not exclusively nor necessarily predictably. Certainly planned production will be achieved by realizing budgeted productivity, but there is no certainty that budgeted or lowest costs will result.

The value of daily production tallies as a measure of efficiency has long been recognized as questionable. Fivaz, Cutland and Balchin (Fivaz, 1973) in discussing open pit fleet control, noted that: *"The initial problem to be tackled was to seek an answer to the question: How good are the operations on shift?"* The measure traditionally used was production tally. This figure however has great shortcomings as a yardstick for operating efficiency…." Although absolute production has been recognized as a questionable standard for measuring operating efficiency, managements often persist with focusing primarily on production, with only secondary attention to operating efficiency, when addressing equipment selection or monitoring open pit operations.

In more recent times, application of sophisticated dispatch systems utilizing real-time analysis of loading and hauling operations by computers has provided a programmed automatic response in allocating hauling units to optimize performance of loading resources. These systems, based on global positioning (GPS) of mobile equipment, are adopted by open pit mine operators to realise the promise of significantly improved truck utilisation in open pit operations with complex multi-loading position options. However, is the efficiency improvement as significant as the market promotional material would have us believe? Perhaps some credit should be afforded the one or more human minds that monitor dispatch systems and over-ride them when practical commonsense so indicates? Effect of degree of complexity of operations must be considered. Digital dispatch systems are obviously more beneficial for large mining operations with several mining groups, say three or more, with more than 20 mining trucks operating. Only moderate or even small benefit can be expected from application of dynamically-programmed dispatch systems to smaller mining operations with 20 or less trucks. In such cases manual dispatch systems alone can provide a substantial improvement and dynamically-programmed digital-computer facilities may only provide small additional improvement? One important advantage of digital dispatch systems is the detailed and comprehensive data collected that can provide useful reports for operations management. Operational data can be delivered in real time, for rapid remedial management response to unacceptable load-and-haul equipment dispositions.

The *de facto* standard of three or four pass loading of mining trucks also warrants investigation. It should be noted that shovel/excavator-truck match to theoretically yield four passes can, in practical truck loading, yield a distribution with values from two to seven passes or more depending on the characteristics of the material being loaded and truck loading practices. This issue is expanded and reviewed in some detail in Chapter 3.

A recent broad industry review of load and haul operations indicated that the *"average benchmark for mining truck loading is five passes"* (Gregory, 2003). In his paper "Excavator Selection", Bruce Gregory indicates that only small cost penalty results from varying from the indicated optimum modal value over a range of four to six passes as illustrated by Table 2.2 (Gregory, 2003).

Other issues in relation to truck payload distribution dealt with in Chapter 5 indicate that, in appropriate best load-and-haul costs may be realized by loading with modal values of five or six passes in preference to the quasi-industry standard of four passes.

Natural learning curves for introduction of new mining equipment to a site require time. Estimated performance and cost estimates using deterministic processes based on theoretical performance criteria are rarely achieved early in the life of new equipment, or substantially upgraded models. This trend seems to be more pronounced as scale of mining trucks increase; also for loading equipment to load them.

Acquiring recently developed large open pit mining equipment with little application history is a risk to be considered. This is especially true if retaining current operating status of the mining operation is imperative. In the early stages of detailed planning activity and at mining commencement, operations may suffer substantial inefficiency from the additional management and technical workload of shepherding largely unfamiliar untried equipment through to optimum performance. Such learning curves can be steep and costly.

Table 2.2 Analysis of Effects of Excavator/Truck Matching On Load and Haul Costs

(Gregory, 2003)

	Excavators								
Operating Weight (tonnes)	**100**			**400**			**650**		
Bucket Capacity (cu metres)	**6.5**			**20**			**34**		
Truck Payload (tonnes)	**Number of passes**	**Load Time (Mins)**	**Cost Index %**	**Number of passes**	**Load Time (Mins)**	**Cost Index %**	**Number of passes**	**Load Time (Mins)**	**Cost Index %**
49	4	2.13	200						
91	7	3.48	144	2	1.25	125	2	1.25	135
146	12	5.73	130	3	1.72	114	3	1.72	115
187	15	7.08	126	4	2.18	104	3	1.72	102
230	18	8.43	134	5	2.65	101	4	2.18	100
353				8	4.05	110	6	3.12	106

Cogent demands on management and technical resources early in a project may make selection of unfamiliar, untried equipment unwise, even unacceptable. Significant benefit-opportunity integral with the higher degree of risk of new, advanced, equipment can be expected. Stable mining operations are likely better able to accept the risk of more technologically-enhanced equipment that offers significant promise of efficiency, productivity and cost benefits.

2.1.3 Cost Benefits from Up-scaling and "The Law of Diminishing Returns"

Increasing scale of mining trucks has realized diminishing cost benefits as demonstrated by Figure 2.1, (Gregory, 2002).

Diminishing cost benefits with scale of mine haulage trucks has resulted in higher sensitivity to marginal and, increasingly significant, latent and interrelated costs that need to be considered in analysis of haulage costs and equipment selection.

OEM's normally offer generically-designed mining truck models, with a range of payload capacities typified by Figure 2.1. Payload to gross machine weight (GMW) ratio for all models is within an overall range of 0.52 to 0.64 – excluding prototype models (Table 4.2, Chapter 4). Variation of payload/GMW ratio between trucks of similar payload from competing OEM is generally within a more moderate range.

Marketing representations by dealers and OEM attempt to emphasize such small differences between equipment they offer and equipment offered by competitors, interpreted as significant productivity and ucost benefits. However, when all relevant factors, speed/torque curves, fuel burn, tyre usage are included in comparisons, differences in mining truck productivity and "ucosts" tend to be subtle rather than significant. Estimated productivity and "ucost" differences between trucks of approximately equal capacity tend to be so small that comparisons are often inconclusive.

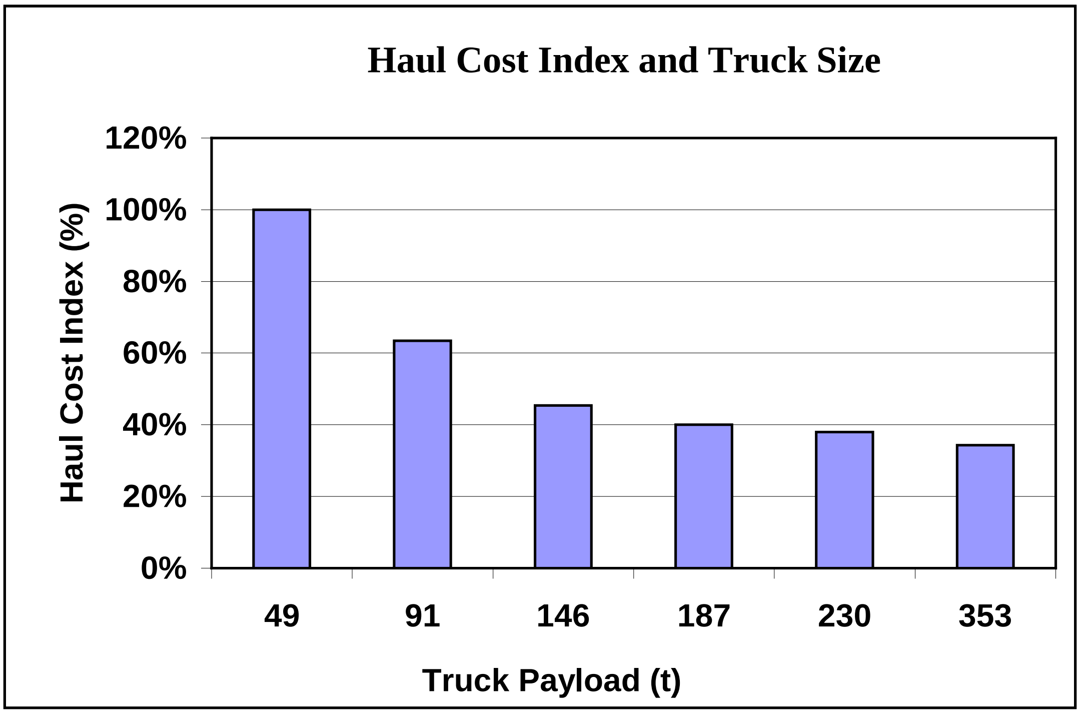


Figure 2.1 Haul Cost Index vs. Truck Size (Gregory, 2002)

Of recent times there have been attempts to change generic designs for improved increased payload/GMW ratio. Other wheel arrangements and radical configurations are available in large special purpose trucks that have recently received renewed industry interest. These developments have been more fully considered during the research and are described in more detail in Section 4.2.4. For the purposes of the following discussion, and generally through the research, only the traditional 2 - 4 wheel configuration, generic designs are considered.

Experience indicates that, over the range of mining-truck models, cost per operating hour of other components of owning-and-operating (O&O) costs are practically proportional to truck capacity. For the purposes of discussion, operating labour is considered fixed. Other owning and operating (O&O) costs for a range of models that are extrapolations of the same generic design result in similar unit-cost components, Cost components including unit cost of fuel, lubrication and related consumables, life cycle maintenance costs, preventative maintenance costs (both including labour) and, to a significant degree, tyres, all tend to be directly related to fuel burn of mining trucks. In turn, fuel burn is directly related to work done, i.e., energy-demand of each operational cycle. Even when truck scale increases, the unit costs for each component input, i.e., cost per unit produced, tends to be constant. But as truck scale rises and productivity increases, cost benefit from the "fixed operator" cost diminishes. It seems that the well-established "Law of Diminishing Returns" – a law recognized by economists and technologists - that "refers to the amount of *extra output* that we get when we successively add equal extra units of a *varying input to a fixed* amount of some *other input*" – (Samuelson, 1958) – has application here.

Samuelson summarizes "The Law of Diminishing Returns" as follows:

"An increase in some inputs relative to other comparatively fixed inputs will cause total output to increase. *But after a point the extra output resulting from the same additions of extra inputs will become less and less*. This falling off of extra returns is a consequence of the fact that the new "doses" of the varying resources have less and less of the fixed resources to work with." (Samuelson, 1958)

Empirical evidence shows that a Law of Diminishing Returns applies to a range of models of mining trucks – (Figure 2.1). This trend has been analyzed in some detail to add to understanding of the limited economic enhancements that are realised by upscaling mining trucks. This has some implications of economic limitations for further upscaling beyond the "ultra"-class mining trucks now available. There are also implications for demand for up-scaled loading equipment that is driven by up-scaled mining trucks. The flow-on implications for support equipment for open pit mining are manifest.

Truck haulage costs are a substantial proportion of open pit mining costs – typically some 38% compared with 15% for drill & blast and 12% for truck loading – Figure

2.2 (Gregory, 2002). The cost breakdown shown in Figure 2.2 is a typical expectation for an accumulation of cost estimates for feasibility studies.

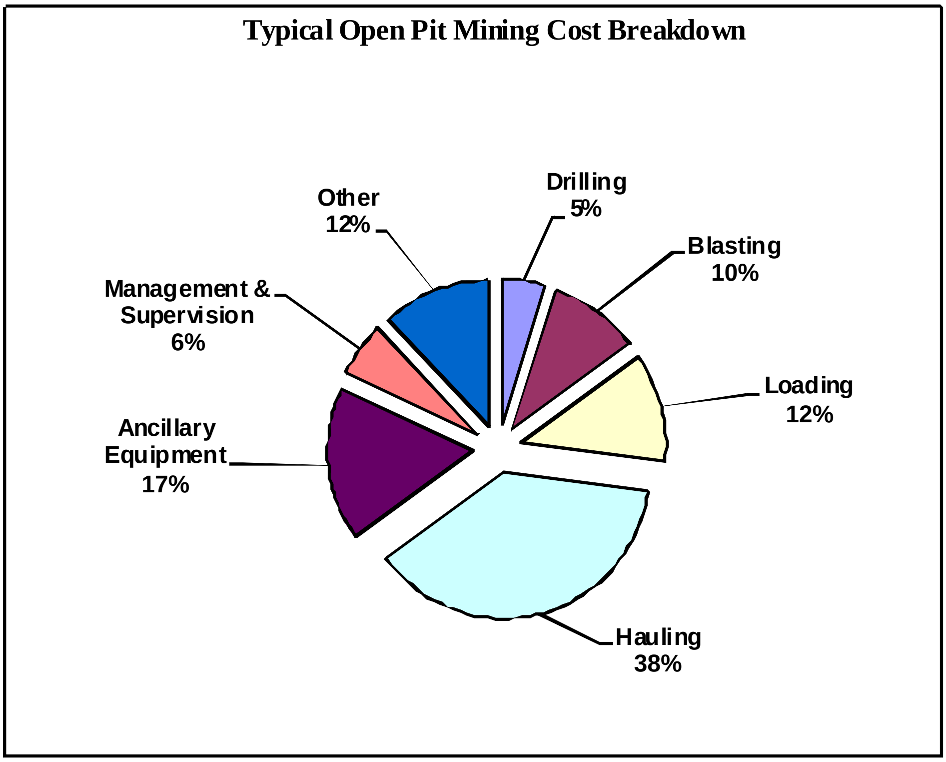


Figure 2.2 – Typical Open Pit Mining Costs by Activity (Gregory, 2002)

As pits deepen, haul distances extend and cycle times increase. Truck-haulage proportions of total open pit mining costs increase. It can be assumed from Figure 2.2 that mean values of combined load and haul costs typically contribute about half of total open pit mining costs. Further loading costs contribute about a quarter of total load-and-haul costs. When load-and-haul cost tends to increase as a proportion of total cost, haulage costs will contribute the majority (more than 75%) of the increase. Understanding the productivity and cost interrelationships between loading and truck haulage is critical for valid analysis of best economics for truck haulage or combined load-and-haul.

The above discussion and illustration demonstrates the relative importance of truck haulage costs compared with total mining costs. Although loading costs do not rank significantly higher than other costs, such as drilling and blasting or ancillary equipment, the interrelationship between loading and hauling demands that, in practice, analysis of either of these elemental activities requires consideration of the operational contribution, constraints, limitations, and costs of the interrelated function.

2.1.4 Improving Technology and Analytical Practices

Equipment selection for a “greenfields” mining operation, or for replacement equipment to sustain or improve project economics as conditions and planning change, requires that the larger the equipment the more precise must be the assessment, selection and estimating processes with commensurately increasing technical and management effort.

It is counter-beneficial for mining equipment to provide improved technology and capability that may be hindered by traditional, over-simplified selection process and practices often overly focused on production, do not include adequate due diligence, and, consequently, disappoint and fail to deliver the economic promise that initially justified the investment.

Research described in this thesis addresses current techniques and practices, considers what is available with current technology, and looks forward to what might be possible in the future.

2.2 RESEARCH PHILOSOPHY

2.2.1 Research Techniques and Reality

The broad process for the research described herein, adopted the policy that individual issues, productivity and cost drivers would be considered in terms of singular attributes whilst, conceptually at least, keeping all interrelated attributes constant. The successive activities of a mining operation are interdependent. The best unit cost per tonne of ore treated or per unit of commodity produced is not necessarily realised by minimizing the operating cost of each individual activity (Atkinson, 1992).

The artificiality of considering processes and mechanics of singular attributes whilst keeping interfacing attributes constant was considered. Parallels were identified in research strategies adopted within scientific fields. The writings of David Suzuki, a Professor of Genetics at the University of British Columbia, are relevant. In his book (Suzuki, 1990) Suzuki describes the scientific process for acquiring knowledge to control the environmental world in which we all live as follows:

"Science's Jigsaw Puzzle

Science provides a unique way of describing the world around us. Its basic methodology is to focus on one aspect of nature, isolating it from all else, controlling everything impinging on it and measuring everything that happens. Science has enabled us to look at nature and gain insights, but ones that are fragmented into bits and pieces.

The apparent success of science in acquiring knowledge necessary to control the world is based on the assumption that if nature is reduced to its most basic parts, a comprehensive explanation will eventually be obtained by simply fitting the components back together like pieces in a giant jigsaw puzzle.

Twentieth century scientists discovered that this hope simply does not hold up in the real world. They found that we cannot know anything with absolute certainty even at the most elementary level of subatomic particles. This is because the behaviour of those particles can be described only statistically, not with certainty. Physicists have also learned that the very act of observing nature changes it, because in order to see it we have to control it; thus we can never know nature as it really is.

Perhaps the most shocking discovery was that when we reduce nature to its most basic components, the description we obtain is of little value in predicting the properties of those elements when they are combined."

There are obvious parallels to the above observations in mining engineering research. Many of the disciplined management processes used, including quality assurance, continuous improvement programmes and formal risk management processes, involve reducing operating systems and cost compilations to the most basic elemental attributes. Analysis techniques consider basic attributes in isolation and manipulate values assigned to each attribute for improved control including efficiency, productivity, cost benefit, reduction to acceptable risk residuals and the like. The points made by David Suzuki are interpreted in mining-engineering-research equivalents as follows:

- Generally mining-engineering research adopts a basic methodology of considering relevant individual attributes of systems in isolation.

- Insights can be gained from analysis of behaviour facilitated by fragmentation into suitable individual attributes conceptually amenable to consideration in isolation.
- The aim is for a comprehensive understanding of a system by collectively considering all attributes together by means of the individual insights gained in isolation.
- Any comprehensive understanding so obtained is subject to a degree of uncertainty; and so must be adopted and applied with a degree of caution.
- Degree of uncertainty is a normal consequence of the nature of the individual attributes where variability characteristics can only be described stochastically.
- Any conclusions drawn from consideration of individual attributes must be applied with caution in forecasting behaviour of the combined system.

2.2.2 Personal Rules and Understandings

The following personal rules have been applied during the research:

- All standards and time-honoured rules may be subjected to scrutiny in the light of advancing technology, availability of empirical data, availability of updated analysis methods and consequent improved understanding.
- Results from analysis of an individual attribute or component, when combined with interrelated attributes within a system, must be re-considered for combinatorial effects.
- The benefit of hindsight in the form of outcomes from research performed by predecessors is a great legacy and advantage to be used with appropriate humility, due acknowledgement and great respect.
- In undertaking research, there must be realization that others may have previously or recently traversed the same investigative paths. Somewhere in the world others may be currently pursuing similar lines of thought and be advancing knowledge in a similar area of research. In such circumstances it is assumed that duplication of research effort is not wasted; it may *we*ll add to the work of others, lead to even better understanding and to discovery of original concepts.

The final results described in this thesis are made available to all others currently working, or who may in the future work in a related field or extension of the research described herein. Future researchers finding these results of value, carry best wishes for success in achieving their objectives.

2.2.3 Units

Metric units have been adopted throughout the research. Other units have been introduced when required but are defined at the first usage.

Hours are solar hours based on 24 hours per day for 365 (366 days per year for leap years is ignored as being of small effect) as the calendar dictates. “Calendar time” is a generic term for the solar-time system.

2.3 METHODOLOGY AND LIMITATIONS

2.3.1 Research Procedures

The general approach to research of selection criteria for load-and-haul equipment for open pit mining was essentially analytical, drawing on:

- Initial conception based on personal experience and field observations to investigate and achieve a better understanding of the systems, drivers, operational problems and outcomes; and
- Empirical evidence collected during the research to test the hypotheses and theories developed by analysis.

Methodology adopted for the research and consequent limitations to scope of the research includes:

- Discussion of initial outline with general objectives with supervisors.
- Literature research.
- Review of personal work history for relevant experiences.
- Identification of issues needing resolution based on experience, peer discussions and literature research and developing strategies to realize solutions.
- Collection of empirical data related to efficiency, productivity and cost performance of load and haul activities.

- Development of hypotheses based on actual observations, analysis and interpretation of results; and interpretations of references or data acquired during the research.
- Work on hypothetical case studies using empirical data from operations and productivity and cost analysis to test hypotheses.
- Interpret results and record outcomes.
- Write a thesis in compliance with guidelines of Curtin University of Technology for submission to supervisors that fulfills the objectives of the approved research outline with a view to ultimate acceptance.
- Research has been limited to examination of significant issues, events, activities and attributes identified during the initial broad review of open pit mining economics.
- The research set out to develop and present each hypothesis, generally prove or disprove validity, to interpret the results in terms of recommended practices and procedures for realizing benefit in efficiency, productivity and operating costs.

2.3.2 Statistics

Collection and analysis of actual data from operating open pit mines has been required. Analysis of that data has yielded descriptive statistics that provide a reliable means of modelling open pit mining production activities for accurate prediction and forecasting of productivity and cost performance.

All empirical data for bucket loads and bucket cycle times have been accepted as continuous random variables. Data for bucket passes is, of course, discrete. Statistical mathematical purists may consider that accepting such data analyzed herein as "random" to be, at least, questionable. That the subsequent analysis assuming variables to be continuous and random; and that any subsequent modelling of distributions appears to yield reasonable consistent and expected results is considered sufficient justification for any assumptions made. Due consideration has been given to the "After this therefore because of this." fallacy when making assumptions.

Analytical procedures generally follow a series of simple activities:

- Data was collected from operations included truck payloads, bucket loads, number of bucket passes per payload, loading equipment cycle times, truck loading times, incremental components of hauling operations time and total hauling cycle time.
- Empirical data was generally assumed to be random and continuous; and could be so modelled, with the exception of bucket passes per truck payload. Bucket passes can be modelled by a Poisson, specifically an Erlang, discrete distribution.
- Comparative dispersion of data distributions for load-and-haul attributes and the probability of occurrence (or non-occurrence)of events, was a principal analytical focus.
- Some confidence interval limits for selected variables were found to be set by design or safety protocols, such as, Caterpillar's "10:10:20 Policy" described and considered in some detail in Chapters 3 and 5.
- Analytical process involved examining data for obvious spurious records and applying appropriate filtering. Any filtering applied to eliminate spurious data has generally been small; and is considered to have not materially influenced the conclusions drawn from developed statistics. But any substantial filtering to deliberately select subsets of data for analytical purposes that are considered material in effect is identified and discussed in detail.
- Development of descriptive statistics for measures of central tendency, measures of dispersion to adequately describe each individual distribution studied, plotting histograms and superimposing a suitable distribution model, also testing for skewness.
- Initial model fitting was qualitative with selected verifications using the Kolmogorov-Smirnov (K-S) test for examples of each type of data including truck payloads, bucket loads, bucket cycle times, truck loading times and truck cycle times. Summary information and sample outputs of Kolmogorov-Smirnov tests from the research and supplementary Probability and Quantile plots are included in a separate section entitled "Distribution Testing" in Volume 2, Appendices.

- Interpretations, implications and inferences that can logically be drawn from the statistical results are described and summarized at appropriate locations throughout the text, generally summarized in Chapter 6 and Chapter 7.

Mathematical background required to implement research analysis is documented in a separate section entitled, Mathematical Principles – Notes, included in Volume 2, Appendices, pp 457 – 486.

2.4 SIGNIFICANCE OF RESEARCH

Research topics within the total body of work where significant advances have been made include:

- Mining-truck payload analysis and interpretations, based on several data sets downloaded from several unidentified open pit mining operations made available by equipment dealers and reported in this thesis with permission of mining property owners that:
    - ➢ Provide improved understanding of payload distribution statistics.
    - ➢ Provide an insight into the relationship between bucket passes and payload dispersion.
- Bucket load analysis using raw data downloaded by Caterpillar’s Minestar communications system from the vital information management system (VIMS) for each mining truck incorporating on-board load sensing and a reporting system.
- Testing and verifying hypotheses based on truck payload and average bucket load relationships - possibly a first published analysis of such data and systems to test such theories.
- Research outcomes have delivered an initial understanding of the difficulties in reducing dispersion of truck payloads. The most effective control methods appear to result from ensuring adequate fragmentation and consistency of condition of material to be loaded and hauled, and/or loading with more passes of reduced quantity.

- Development of greater understanding of limitations to cost benefits from upscaling mining trucks that will assist with equipment selection and operating practices in open pit mines.
- Expanding on, and advancing previous research, on the comparative cost benefits over a range of bucket passes to load mining trucks. *De facto* industry standards of three/four pass loading have been questioned (Hardy #1, 2003) further investigated and, subsequently, comparative cost benefits quantified identifying and allowing for a number of latent costs that are most often ignored in traditional load-and-haul cost estimations.
- Potential questionable reliability of predictions and forecasts due to general adoption of traditional deterministic analysis has been investigated. The necessity for a more realistic stochastic analysis for determining behaviour of many variable attributes in loading and hauling systems is a significant research outcome.

2.5 FACILITIES AND RESOURCES

All research work has been undertaken solely by the author. Experience gained during more than thirty-five years of mining technical support, mining contracting, mine management and mining consulting practice has provided a valuable foundation for the research. A significant resource supporting the research is the author's relevant experiences together with a substantial personal reference library including technical reports, notebooks and diaries. The many case studies accumulated have provided a substantial basis for the research. A well-equipped home office, supplemented by occasional visits to the WA School of Mines, Kalgoorlie has provided facilities in which the research was implemented.

The Robinson Library within Curtin University of Technology, industry and journal websites have provided relevant reference material.

Research supervisors, colleagues, industry peers and contacts, OEM and dealer representatives have been most supportive and generous with information, data and pertinent references. Where used in the research process, such material assistance has been duly acknowledged.

2.6 ETHICAL ISSUES

The nature of the research, industrial orientation and sources of information do not necessitate any significant ethical considerations.

No person, persons or organisation have been identified or quoted without permission, especially personal communications, or the material has been published and is available for public information. All such material used in the research has been duly acknowledged.

To the best of the author’s knowledge, there have not been any ethical transgressions either in process of the research or recorded in this thesis.

CHAPTER 3

LOAD AND HAUL PERFORMANCE

3.1 PROJECT PRODUCTION SCALE

3.1.1 Introduction

A number of open-pit project scale parameters have to be identified and/or determined prior to selecting loading and hauling equipment to deliver required productivity most economically. Equipment selection should preferably provide some flexibility to cover risks such as latent production timing demands. The degree of flexibility provided as a risk treatment to cover short term demands to upscale; or downscale, operations is a matter for assessment depending on the risk profile of each individual project.

An early and essential task is to determine the initial scale of operations. Production programmes and mining rates used for initial planning need to be confirmed by proving feasibility including project economics, marketing, environmental and socio-economic acceptability and general practicality and community acceptance prior to committing to development. Many factors influence the production scale of a project including:

- Mineral resources and economically determined ore reserves available for mining.
- The commodity to be produced – whether it is nationally and/or globally, strategic, whether it is high unit value such as precious metals or gemstones or low unit value such as bulk commodities including salt, basic construction materials, iron ore, oil sands, steaming and coking coal.
- Marketing issues including local, national or international demand, commodity production the market will bear and long term marketing strategy for project production.
- Availability of the necessary service resources such as power and water and the necessary generating, capturing and reticulation infrastructure, human and equipment resources that can be tapped for the benefit of the project; and, if

not existing or available, a practical situation exists for establishment of the project resource requirements.

- Environmental and sociological constraints that may dictate capping productivity of a development project even to the extent of rendering it uneconomic.
- Intrinsic capacity of some infrastructure items such as railways, pipelines, conveyors and ports that have a minimum capacity for viability; and that predetermine a minimum production level to be tested against other potential constraining factors.

Factors identified above are some of the preliminary considerations that may well yield a determination, at least for planning purposes, of a suitable production scale for the project. In the absence of such project-specific criteria, it is necessary to consider previous industry experience either personal, advice from peers, or recorded by authors that have proposed standard formulae for the purpose. Lawrence Smith provides some "rules of thumb" divided broadly into two groups – (1) related to physical characteristics and (2) related to economic characteristics.

For current purposes the discussion will be confined to "Taylor's Law - for underground and open pit mines except for a few structurally controlled situations":

$$t/d = 0.014(\text{Reserves})^{0.75}$$

t/d = tonnes per day of ore produced. For open pit operations t/d must be increased by the waste/ore ratio and the total production rate so derived. Taylor's Law has been recorded here "for reference only" and for when all else fails. It "should be used with caution" (Smith L D, 1990).

3.1.2 Capacity Hierarchy

Experience indicates that each facility in a mining and ore treatment process tends, as the project develops and settles down to a mature production operation, to have a redundancy that progressively reduces in degree through the process. Managements seem to provide a degree of over-capacity relative to the next-in-line downstream activity in a project process hierarchy. Ernest Bohnet supports this with "For any analysis (of mining method or equipment selection) it is better to have the first unit in the chain of production with greater capacity than the downstream unit." (Bohnet, 1990). In-built redundancy appears to be particularly so when a process stage consists a single production unit. Immediately following discrete production facilities; and sometimes before, surge capacity is usually built into the system such as primary crushed ore stockpiles to facilitate continuous supply to downstream process activities.

In the mining operational context, drilling equipment generally has significant overcapacity as does hardware and facilities for preparation of and delivery of blasting agents. Mining capacity generally exceeds the ore production schedule. Capacity of loading equipment generally tends to exceed total mining production requirements; that is exceeding intrinsic capacity of the truck fleet. Evidence to support this observation is seen in bench marking studies as discussed in 3.2.4. There occasionally appears to be a tendency to over truck production schedule requirements; but in mature open pit operations truck redundancy seems less likely than excess capacity of loading equipment. Crusher capacity is often some 50% redundant compared with milling capacity, particularly where there is only a single primary crusher. In such cases there is always a substantial crushed ore stockpile to sustain mill feed for crusher downtime.

- It is most cost effective to deliver ore as mined and hauled directly to the crusher; but ore supply from the pit does not always match and may exceed crusher capacity from time to time. Many operations stockpile all or at least part of mined ore and reclaim it for transport to the crusher cross blending in the process. The cost of re-handling all ore to be crushed is often justified by, and offset by improved concentrator recovery. Cross blending metalliferous ores requiring concentrating delivers a more uniform head grade to the concentrator with resulting enhanced recovery. Ore stockpiling to facilitate cross-blending generally provides a bonus of flexibility where ore exposure in the pit is intermittent and vice versa. The ore re-handle equipment fleet usually has a significant redundancy compared with the operating profile of the crusher. Additional capacity in the re-handle fleet adds a further bonus of flexibility to mining operations.

The general tendency for provision of excess mining equipment capacity has at least two benefits:

1. There is inherent flexibility that can provide rapid response to production schedule increases, at least to an initial degree. If the loading equipment capacity is available it is more convenient and generally an easier task to add some hire trucks and auxiliary support for short term cover until the production crisis passes or until permanent haulage equipment and auxiliary support can be procured. Particularly, in the larger shovel range, hire or used equipment is not as readily available as trucks and support equipment.
2. Experience has shown that continuous improvement campaigns for cost benefits are most beneficially progressed by increasing utilisation of existing equipment capacity. Working operating equipment more hours in any given period is most beneficial. The next most beneficial improvement opportunity is to provide operating and maintenance labour to make operational any equipment fleet redundancy. This last practice manifestly reduces operational flexibility and increases operational risk. But, the immediate cost benefits and rewards of the resulting business opportunity may be attractive so making the increased risk acceptable. Depending on economic forecasts investing in equipment redundancy as a risk treatment may be justified by the benefits of future production flexibility.

In summary, mining operations have a tendency to be initially supplied with; or to acquire over time, equipment with substantial excess capacity. Justification for this over-capacity is believed to be a manifestation of intuitive or formal risk treatment by mine developers and mine managers. Formal risk analysis is a burgeoning technique that is a natural outcome of the needs of the mining industry to work smarter to accommodate the cost pressures resulting from the long-term real-terms prices for mined commodities as described in Section 2.3.2. Identifying, assessing and treating of mining operational risk is the formal process that likely justifies current mining equipment over capacity. But the industry has a long history of such overcapacity provisions that can only be explained as the intuitive response to the lessons of experience.

In the research described by this thesis the philosophy of carrying mining equipment overcapacity is not endorsed unless there are valid analytical reasons, technical and/or risk assessment, to over truck or over shovel.

3.2 LOADING EQUIPMENT

3.2.1 Selection Process

Surface mining operations (including open pit mining) require the application of many interdependent activities including ground preparation (ripping and pushing, drilling and blasting), excavation and loading, transport, waste disposal, ore sizing, attritioning and treatment and commodity recovery, packaging and transport (Atkinson, 1992).

This thesis is confined to open pit mines and to load and haul and closely related activities. Because of the interdependence between all activities in a mineral-commodity business endeavour, it is potentially misleading to focus on one individual activity or small partition of activities, refer to Section 2.5.1. Throughout the analysis that follows in this thesis, any results from isolated considerations have been viewed with caution. Such results have been considered, and, where possible have been tested, in combination with all inter-relatives; and, where necessary, modified to reflect these considerations and testing.

Elevated risk from equipment selection in isolation cannot be overstressed. Any process to select excavating and loading equipment must include due consideration of the mining trucks to be loaded. Similarly any selection process to select mining trucks must include due consideration of excavating and loading equipment to load them. In these circumstances it is fitting to consider which should be selected first - loaders or trucks? Traditionally there has been a tendency for selection of excavation and loading equipment to satisfy a production requirement and trucks are then selected to service the transport requirements. This is despite the well-recognized component cost criteria where transport is two to five times the cost of excavation and loading in open pit mines. Reliable processes for determining loading equipment and truck types, size and numbers consider loading equipment and trucks in isolation for convenience but in combination at the first opportunity in the process. This is revisited in Section 3.3.

3.2.2 Cyclic vs. Continuous Excavation

There are many options for materials handling systems and mining equipment to conduct open pit mining operations. The general materials handling system will be determined by the commodity, downstream treatment of the commodity such as direct use for power generation, feedstock for iron and steel, secondary and tertiary treatment to a high value additive for special manufacturing economies or extraction and recovery of precious metals or gem stones used in high-value industries or as a medium of financial exchange. Generally mining systems can be classified as *Continuous* or *Cyclic* (Atkinson, 1992).

- *Continuous excavation systems* used in open pit mining include bucket wheel and bucket chain excavators or dredges applied in brown coal mining for power generation typified by many European examples; also Yallourn and Morwell in Victoria – essentially large-volume open-pit mining operations.
- *Cyclic excavation systems* include shovels, draglines, wheel loaders and scrapers with ripping dozers that are specifically or severally applicable for a large range of operational scales, commodities and open pit mining configurations.

Continuous excavation systems are generally matched to continuous transport systems including belt conveyors, pipelines and, historically, aerial ropeways: but there could be applications where continuous excavation could be matched to cyclic transport, mining trucks, where the material mined suits continuous excavation but operational life or production rate cannot justify investment in continuous transport. Cyclic excavator systems can be matched to continuous transport systems particularly for large-scale long-life deep open pits where waste and ore transported on conveyors (generally after crushing to conveyable size) is the best economic solution. Most often, cyclic excavation systems are matched to cyclic transport systems, typically conventional loading equipment loading mining trucks. This cyclic excavation-transport combination is the focus of this thesis.

3.2.3 Loading Equipment Options

Rope shovels (sometimes termed cable shovels) have a long development history. Initially on rails, steam-powered rope shovels gave way to on-board diesel powered shovels on caterpillar tracks. As scale of shovels increased, larger electric DC motor drives were developed powered by AC/DC motor/generator sets controlled by Ward-Leonard systems with reticulated power delivered by trailing cable at standard frequencies (Hertz – Hz) and voltages now available to 13,800v AC.

Prior to the development of draglines for stripping overburden from coal seams, large stripping shovels were developed to side cast overburden. Recent developments in large mining shovels have focused on more efficient AC motor drives and electronic control systems.

Development of wheel loaders, from their introduction in the late 1930's, has continued for some fifty years with ever-larger machines currently offered from a number of original equipment manufacturers. Typically cycle times of large wheel loaders are some 150% or more compared with cycle times for an equivalent shovel or excavator. Mechanical drive loaders to 210 tonnes and 50 tonne bucket load have recently been surpassed by an electric-wheel loader at 245 tones operating weight and 72 tonnes bucket load. Cycle time improvements have also been an added benefit of recent large wheel loader development.

Large wheel loaders are often favoured as support loading equipment because of mobility advantage. They can more readily relocate to cover for unplanned downtime of a rope shovel, hydraulic shovel or excavator or to clean up small quantities of batter trimmings and like activities Where mobility, in-pit blending, multi-material selectivity or self-sufficiency for pit-floor housekeeping are major issues, particularly in shallow open-pit operations such as lateritic ores, bauxite mining and similar, wheel loaders may be applied as prime loading equipment. Final clean up to prepare faces for blasting are also an advantage as the loader can work up to within a few minutes of a time deadline for clearing a mining locality for a blast. More recently developed or upgraded large wheel loaders have the advantage of faster digging cycles that more closely approach a rope or hydraulic shovel.

Hydraulic excavators, initially in backhoe configuration and later as shovels, have provided an alternative for loading mining trucks. Initially a product developed in Europe for the construction industry, where boom and stick reach were important, hydraulic excavators; and subsequently shovels, have developed specifically for mining operations. Features of the excavator evolution include, the so-called "mass excavation" configuration - larger buckets for enhanced production capacity - necessitating shorter booms and sticks for excavators, short booms and arms for shovels. First appearing in Australia in the early 1970's with 4 – 5 CM buckets, 80 – 90 tonne operating weight, hydraulic loading equipment solutions have evolved to the current availability of 47 CM (85tonnes) bucket shovels with operating weight of some 1,000 tonnes.

A notable feature of hydraulic face shovels is the ability to pivot the bucket, changing the angle of attack of bucket teeth, at any point within the operating envelope to enable crowding into or breaking out of the face. Some hydraulic shovels have mechanisms to enhance this feature. Not available on rope shovels, this feature, allows for removal of material in the face from the top down, an obvious advantage for hydraulic shovels that facilitates selective mining and prevention of face overhang (Files, 1990).

John Wiebmer (Wiebmer, 1994) observed: “No two mines are the same. Each requires a unique system of people, machines and processes to meet production requirements profitably. Miners today, as never before, have a wide array of choices in loading tools to meet operational needs. Cable (rope) shovels, hydraulic shovels and excavators and wheel loaders … different horses for different courses.” The breadth and depth of choice imposes a need for due diligence in determining the complete characteristics of the service required and careful analysis of the options to satisfy those needs.

Early considerations should consider the special features, advantages and disadvantages of each type of loading equipment in comparison with the unique conditions and characteristics of the proposed mining operation. Some of the initial considerations are listed in Figure 3.1, with a suggested ranking for each type of equipment.

In the early stages of an open pit mining development the question will arise: “What type of loading equipment will be most applicable and deliver best economics?” This question will be addressed in more detail in a following section. The answer can only be delivered by a duly diligent analysis of the specific mining application each of which are, at least in detail, unique.

Loading Equipment	Rope Shovels	Hydraulic Shovels	Hydraulic Excavators	Loaders
Feature or Application				
Large Scale Operation	●	●	■	■
Small Scale Operation	⬢	■	●	●
Selective Mining	■	◇	●	■
High Benches	●	●	⬢	■
Low Benches	⬢	■	●	◇
Steep Dipping Ore Contacts	◇	◇	◇	◇
Mobility	■	◇	◇	●
Gradeability	■	●	●	◇
Ground Pressure	■	●	●	■
Unstable Floors	⬢	◇	●	■
Pitching Floor & Ramp Down	■	◇	●	◇
Flat-Dipping Narrow Ore Seams	⬢	■	●	◇
Hard Digging	◇	●	◇	■
Large Particle Disposal	■	■	■	●
Clean-up Support	■	◇	◇	●
Machine Life	●	◇	◇	◇
Mechanical Availability	●	◇	◇	◇
Reliability	●	◇	◇	◇
Largest Mining Trucks	●	◇	■	◇
Capital Cost	◇	◇	◇	●
Operating Cost	●	◇	◇	◇
Project Uncertainty	■	◇	◇	●
Energy Cost (Power & Fuel)	●	◇	◇	◇
Legend	**Favoured**	**Applicable**	**Questionable**	**Not Applicable**
	●	◇	■	⬢

Figure 3.1 Loading Equipment - Comparative Features and Applications

3.2.4 Intrinsic Loader Performance

Absolute hypothetical intrinsic loader performance - P_{ah} - for purposes of this discussion is defined as the hypothetical absolute production of any loader ("loader" is herein used as a generic term for any item of equipment that loads mining trucks). It is the hypothetical production rate that could be achieved by a loader continuously cycling at a specific (mean) cycle time. In practice hypothetical intrinsic loader performance is naturally discounted by a number of constraints on productivity. Exclusive of all practical constraints absolute hypothetical productivity - P_{ah} - is simply derived:

$$P_{ah} = \text{Hours per year } \mathbf{x} \text{ Seconds per hour} / C \text{ ---------- } (1)$$

Equation (1) yields loose cubic metres (LCM) per cubic metre of bucket capacity.

C is the estimated mean loader cycle time in seconds.

P_{ah} is equivalent to the displaced volume of a loader with a 1 CM bucket cycling continuously. It will be noted that the equation also yields hypothetical maximum loading cycles per year.

Productivity

The following discussion is based on a description of dragline productivity factors and calculations that are directly applicable to truck loading equipment (Humphrey, 1990).

To arrive at a practical productivity for loading equipment the following limiting factors must be applied to P_{ah}. Some of the factors have distinct categorical variations critical for loading equipment productivity calculations.

Time Factors – summarized in more detail in 3.6:

- Calendar Hours H_c – solar hours in any given period of observation, e.g., year or day based on solar time – 24 hours per day for 365 (or 366 days per leap year) generally termed "hours" in this thesis.
- Scheduled Hours H_s – the annual hours the loader is expected to operate at expected productivity – 8,760 (8,784 for a leap year is generally ignored for convenience) less allowances for public holidays, any planned interruptions to operations and daily hours when work is not scheduled.
- Available Hours A – the part of scheduled hours when the loader is mechanically and electrically ready to operate – those scheduled hours when the equipment is under the operational control of open-pit production management and not out of service for maintenance.
- Utilized Hours or Operating Hours U – the part of the available hours that the loader is actually operating at expected productivity – excludes extraneous lost time due to unplanned interruptions to operation of specific items, groups or all mining equipment and any inefficiency due to logistical mismanagement or inadequate operator response.

Material Factors – see Section 3.2.7 for more detailed discussion:

- In situ Density (as determined by laboratory density determination from samples termed Apparent Rock Density – ARD) – tonnes per BCM - an inherent property of the material to be mined that varies with the geological setting; but also may vary within any discrete stratigraphic horizon – a typically low-dispersion random variable.

- Swell Factor S_w – a variable dependent on the inherent structural and geotechnical properties; and the mechanisms and degree of disturbance of the material to be mined measured as the ratio of the volume of a specific weight of material after disturbance by blasting, ripping and pushing, loading into trucks or dumped at material disposal points – crusher, ore stockpile or waste dump and the in situ volume. That is:
    - *Swell Factor = LCM/BCM = (1 + Swell) --------------(2)*

  LCM =loose cubic metres

  BCM = Bank – in situ – cubic metres

Equipment Factors – more detailed discussion is continued in Sections 3.2.7 and 3.2.8:

- Rated Bucket Capacity B_c – generally reported as struck by rope shovel manufacturers and 2:1 heaped by hydraulic shovel and excavator manufacturers with capacity determined in compliance with an industry standard, i.e., SAE J67.

- Loader cycle time in seconds $T_{c,}$ or cycles per unit of time as convenient, that includes due allowance for non-optimum digging depth varying angles of swing – a random variable that is dependent on the condition of material to be loaded, physical characteristics of the loading operation, mechanical and electrical/hydraulic performance of the loader, efficiency of the operator and any extraneous interference that may impinge on the loading operation.

- Bucket Fill Factor F_f – a variable that typically, for full bucket loads, can be expected to have low dispersion (coefficient of variation of 0.1 or less) considered volumetrically so visual control and measurement is possible and can be effective - generally defined volumetrically as the ratio or percentage of actual volume in the bucket compared with a standard rated bucket capacity that vary for loader types – see next point.

- Bucket Factor B_f (F_f / S_w BCM or ARD . F_f / S_w tonnes) – a random variable that combines the effects of in situ density, swell and volumetric fill factor – enables analytical connection of loader performance with in situ volume and computations in terms of mass – generally expressed as a dimensionless factor that can be interpreted in terms of volume or mass depending on the context – a convenient cover-all device.

The above factors can be applied to obtain a preliminary productivity estimate P_e – BCM/hr - as follows:

For draglines; also applies to loaders delivering to surge accommodation and continuous transport systems (Humphrey, 1990):

$$P_e = (3600/T_c) \text{ x } (H_s/\text{year}) \text{ x A x U x } (B_c/S_w) \text{ x } F_f \quad \text{BCM/hr ---------- (3)}$$

For loading shovels (after Atkinson, 1992 and substituting symbols):

$$P_e = B_c \text{ x } (3600/T_c) \text{ x SF x A x U x } B_f \text{ x PF} \quad \text{BCM/hr or tonnes/hr ---------(4)}$$

Tc is the average loading cycle time - in equation (4). OEM and equipment dealers generally provide average cycle times for loading equipment items. Empirical values are tabulated for a range of digging conditions. Typically four levels of digging difficulty are specified in empirical terms viz., Easy, Medium, Hard and Very Hard (Harnischfeger, 2003). Cycle times generally tend to increase with scale of equipment and difficulty of filling the bucket. Both large rope shovels and hydraulic shovels cycle in the range 25 to 45 seconds depending on operating conditions. Actual observed cycle time data is discussed in more detail in Sections 3.2.9 and 3.3.10.

Obviously, if cycle times are manifestly increased by digging difficulty, the rewards from improvement will justify significant effort and investment in improving operating conditions by attention to drilling and blasting, particularly pattern accuracy, hole alignment, consistent depth of hole, sub-drill depth, stemming length and stemming quality, detonation sequence and delay; also by increasing drilling per unit volume and powder factor to some cost-justifiable degree. The ultimate benefit from improvement is achieving reasonable digability of material to be mined and best-practice average cycle times.

SF is a swing factor is a correction for loader cycle time to allow for variation in swing arc that will be 1.0 if due allowance has been included in T_c. Otherwise SF for rope shovels can be estimated from Table 3.1.

Table 3.1 Correction Factors for Rope Shovel Cycle Times

(Harnischfeger, 2003)

Swing Arc - Degrees	45	50	60	70	100	130	180
Indicated Productivity Factor	1.26	1.16	1.07	1.00	0.88	0.77	0.70

Empirical correction factors for varying average swing angle are provided by a number of authorities with small differences in what is adopted as average swing arc and swing factors. Average swing arcs of 90 degrees (Atkinson, 1990) and 70 degrees (Harnischfeger, 2003) are offered by two sources. A typical set of factors is provided in Table 3.1.

Productivity of loading equipment is affected by height of the working face. Cycle times recommended in literature and by OEM assume optimum digging depth. Mine design and equipment selection should ensure that loading equipment is working at or near optimum digging depth. In the unlikely event that allowance has to be made for non-optimum digging depth, the following Table 3.2 indicates suitable correction factors. This correction should not be an issue. The need to apply it implies a misapplication and the need to consider equipment type and size and bench heights that will match optimum digging depth.

Table 3.2 Non-Optimum Digging Depth -

Correction Factor for Rope Shovel Cycle Time (Atkinson, 1992)

Optimum Digging Depth %	40	60	80	100 plus
Cycle Time Correction Factor	1.25	1.10	1.02	1.00

For rope shovels optimum digging depth is approximately the same as eye level of the operator, maximum dumping height or the bottom of the stick when level. Hydraulic shovels generally have maximum crowd and break out at bottom of the face; but can exert relatively higher crowd and break out forces, compared with rope shovels, at higher levels in the face. Consequently optimum face height tends not to be as important an issue for hydraulic shovels. As a rule of thumb, faces less than double the bucket height (placed opening down) are unlikely to yield expected hydraulic-shovel productivity. Maximum *practical digging depth* for backhoes can be taken as the length of the stick. Maximum *operating reach* of backhoes is generally 20% greater than the length of the stick.

PF is a Propel Factor (<1) to allow for non-productive movement of the loader to follow the working face in the course of normal loading operations. PF does not provide for major moves between working faces, vacation of the face area for blasting, backing out for service or maintenance and any other unscheduled interruptions. These time losses are generally allowed for in utilized hours factor U or as a separate lost-time category. It needs to be clearly established that utilization U hours does not allow for PF otherwise it must be assigned a value of one or ignored. Atkinson (Atkinson , 1992) provides an estimate of 0.85 for PF for rope shovels in open pit mines. Compared with rope shovels, hydraulic shovels have less reach so move more often but propel at higher speed and do not require assistance with a trailing cable or cable bridge. In the absence of better field study data, a factor of 0.85 is considered reasonable for all excavation equipment in open pit mines.

In the following calculations P_e has dimensions and units convenient for the outcome required.

Per cubic metre of bucket space (3) yields:

$$P_e = (3600/30) \times 1.0 \times 0.9 \times 0.9 \times 1.0 \times 0.85 \times 6500$$

$$P_e = 537{,}030 \text{ LCM/bucket cubic metre per year}$$

Where:

$T_c = 30$ secs SF = 1.0 A = 0.9 U = 0.9 $B_f = 1.0$ PF = 0.85

$H_s = 6500$ hrs per year.

Allowing for truck loading specifically change over time, say, factor 0.9; and for time lost due to truck bunching (queuing) factor, say factor 0.9; also adopting a loose density (equivalent term = bulk material density) of 1.8 tonnes per LCM, then:

P_e = 537,030 x 0.9 x 0.9 x 1.8 tonnes per year per bucket cubic metre

P_e = 782,990 tonnes/bucket cubic metre; or

P_e = 284,724 BCM/bucket cubic metre per year at an ARD of 2.75 tonnes/BCM

Actual industry benchmarks provided in Table 3.3 (Tasman, 1997) appear low compared with the hypothetical indicative P_e of 782,000 tonnes per bucket cubic metre. Bench Marks in Table 3.3 reflect other time losses due to pit activities non-intrinsic to the load and haul operation.

Table 3.3 Loader Productivity Bench marks

(Tasman, 1997)

Industry Sector	**Shovel Productivity**	**All Loaders Productivity**
	‘000’s tonnes per LCM of bucket capacity	**‘000’s tonnes per LCM of bucket capacity**
US Coal Mines	**494**	**308**
NSW Coal Mines	**661**	**337**
Queensland Coal Mines	**510**	**250**
Australian Hard Rock Mines	**604**	**378**

Time losses non-intrinsic to the loading operation are discussed in more detail in Section 3.2.9 and industry practice of “over-shoveling” to overcome productivity penalties due to bunching and combinatorial effects (both of these will be discussed later in this thesis), to cover for manifestation of latent operational risks and to provide operational flexibility.

Experience by the author indicates that annual productivity targets of 650,000 tonnes per bucket cubic metre for a continuously improving operation are realistic.

Loading equipment redundancy identified by Tasman in their benchmark studies provides evidence of the comments on loading equipment over-capacity norms discussed in Section 3.1.2. US open pit coal mines practice over-shoveling in contrast to over trucking generally practiced by Australian open pit coal mines; and, generally, US coal mines were more efficient and lower cost than either NSW or Queensland coal mines (Tasman, 1997)

As a preliminary step in excavation equipment selection expected excavator productivity can be estimated from equation (4) by addressing each of the factors and assigning a value suitable for the mining project under consideration. An alternative approach is to adopt productivity per bucket cubic metre of, say, 600,000 tonnes and so determine the bucket cubic metres required. Consideration of the number of working locations then determines the bucket size for each of the chosen number of shovels. This naturally leads to an indication of the size, or sizes, of mining truck that should be considered for truck selection.

3.2.5 Single or Double Side Loading

The industry standard for truck loading is single-side loading with the truck to the left of the loader when addressing the face.

The operator's station or cab is situated to the right of the boom on rope shovels so the operator has a good view of the face on the return swing from a conventional single-side truck loading arrangement and can select the next digging point. On the loaded swing the stick is practically horizontal and generally above the operator's line of sight facilitating spotting of the bucket over the truck. Hydraulic shovels, generally physically smaller than rope shovels of equivalent performance, traditionally have the cab on the left hand side to facilitate bucket spotting. On the return swing the boom lowers and the arm folds back to draw the bucket back to the toe of the face. At this stage the operator can see over the arm and bucket to spot the digging position. Operators of excavators configured as backhoes have generally better visibility conditions compared with hydraulic shovels as the excavation envelope is well below the cab. In larger hydraulic shovels, cabs are sometimes mounted on the right side of the boom. The high cab and relatively short bucket arm provide reasonable visibility of the face to select digging positions.

Wheel loaders have centrally located cabs generally mounted on the rear module behind the articulation hitch to improve the ride for the operator and provide a safer position when working under high faces There are historical examples of a large loaders where the cab was offset to the right side as with rope shovels; also another example where the cab was mounted on the front module in an attempt to improve vision and a more conventional driving feel when articulating. Both these cab configurations for wheel loaders have been discontinued.

Truck loading time is lost when trucks exchange for loading. After the last bucket load is placed on a truck the loader returns to the face to pick up the first bucket load and then swings (or trams in the case of a wheel loader) to spot the incoming truck. Truck "turn-and-spot" time is typically some 45 seconds but can vary depending on manoeuvering safety practice and available bench space. As the truck "turn-and-spot" time generally exceeds loader cycle time, the nett time between "turn-and-spot" and loader cycling, i.e., "truck exchange time", is waiting time for the loader – optimally in the order of 10 seconds. In the previous section it was shown that some 500,000 cycles per year are practical with a loader. So, for optimal "truck exchange time" and, depending on the number of passes for truck loading, annualized change over waiting time for the loader is in the order of 350 hours for 4-pass loading and 230 hours for 6-pass loading. The implications of this loading inefficiency will be addressed in more detail in Section 3.2.6.

To eliminate, or at least reduce the changeover loader waiting time; and where operating conditions permit, double-side loading may be adopted. In this method the next truck in line spots on the other side of the shovel ready to be loaded.

The larger dumping radius of rope shovels compared with hydraulic shovels provides more margin for error for an operator positioning a truck without benefit of bucket spotting by the loader operator.

Double-side loading is not always possible, particularly as working-face length reduces. Opportunity for double-side loading with the larger range of shovels reduces to some 50% or less where working faces reduce to 150 metres or less in width. It may be unsafe or practically impossible for trucks to be spotted on both sides of the loader where materials are piled against pit walls, at bench edges or in narrow working locations.

Best applications for double-side loading are bulk-mining operations such as open pit coalmines where shovel-and-truck operations chop down ahead of a dragline operation. Pre-stripping benches in strip coalmines can have extended face lengths measured in kilometers. Iron ore and other large base metal operations, where mining of long faces is the norm; and only minimal selectivity is required, are suited to double-side loading.

In practice, first bucket loads inclusive of truck exchange time are often used by loader operators as opportunity for some face housekeeping that, even with double side loading, is necessary so there is occasionally some loading delay. Although promising significant efficiency improvement double-side loading tends to deliver only modest time benefits for those exceptional open pit mining operations where rope shovels are working on long working faces for extended and continuous periods.

3.2.6 Loader Passes and Truck Exchange Time

In Section 3.2.4 time losses for propelling to follow the face, truck exchange time and bunching (queuing) were identified as required provisions in the process of estimating loader productivity.

Time lost in propelling is more fully considered in Section 3.2.9 along with extraordinary time losses such as handling large particles and for bench cleanup (except for self-sufficient wheel loaders).

Bunching warrants, and is given substantial consideration, in separate Section 3.4.

Truck exchange time reduces loading efficiency to a degree dependent on:

- Truck “turn-and-spot” time;
- Cycle time for the first bucket load; and
- Number of bucket passes to load a truck.

It is generally concluded that increasing the number of passes reduces truck haulage productivity. The logic is that additional passes increases truck cycle time so reducing productivity. This conclusion is not necessarily based on complete consideration of all factors involved. It will be considered in more detail in Section 3.3.

Paradoxically, increasing the number of passes increases loading efficiency as shown in Table 3.4. It is apparent that there is a trade-off between efficiency of trucks and loading equipment serving them. Figure 2.2, shows the relative importance of truck haulage costs as a proportion of total mining costs. The importance of considering interrelated truck and loader productivity and costs concurrently was stressed in Section 3.2.4.

Table 3.4 Loader Efficiency - η_{LE}% - and Number of Truck-Loading Passes

Loader Cycle Time - Secs	20					30				
Exchange Time - Secs	45	30	20	10	5	45	30	20	10	5
Passes Per Truck Payload										
1	*30.77*	*40.00*	*50.00*	*66.67*	*80.00*	*40.00*	*50.00*	*60.00*	*75.00*	*85.71*
2	*47.06*	*57.14*	*66.67*	*80.00*	*88.89*	*57.14*	*66.67*	*75.00*	*85.71*	*92.31*
3	**57.14%**	**66.67%**	**75.00%**	**85.71%**	**92.31%**	**66.67%**	**75.00%**	**81.82%**	**90.00%**	**94.74%**
4	**64.00%**	**72.73%**	**80.00%**	**88.89%**	**94.12%**	**72.73%**	**80.00%**	**85.71%**	**92.31%**	**96.00%**
5	**68.97%**	**76.92%**	**83.33%**	**90.91%**	**95.24%**	**76.92%**	**83.33%**	**88.24%**	**93.75%**	**96.77%**
6	**72.73%**	**80.00%**	**85.71%**	**92.31%**	**96.00%**	**80.00%**	**85.71%**	**90.00%**	**94.74%**	**97.30%**
7	**75.68%**	**82.35%**	**87.50%**	**93.33%**	**96.55%**	**82.35%**	**87.50%**	**91.30%**	**95.45%**	**97.67%**
8	**78.05%**	**84.21%**	**88.89%**	**94.12%**	**96.97%**	**84.21%**	**88.89%**	**92.31%**	**96.00%**	**97.96%**
9	**80.00%**	**85.71%**	**90.00%**	**94.74%**	**97.30%**	**85.71%**	**90.00%**	**93.10%**	**96.43%**	**98.18%**
10	**81.63%**	**86.96%**	**90.91%**	**95.24%**	**97.56%**	**86.96%**	**90.91%**	**93.75%**	**96.77%**	**98.36%**
Loader Cycle Time - Secs	**45**					**60**				
Exchange Time - Secs	**45**	**30**	**20**	**10**	**5**	**45**	**30**	**20**	**10**	**5**
Passes Per Truck Payload										
1	*50.00%*	*60.00%*	*69.23%*	*81.82%*	*90.00%*	*57.14%*	*66.67%*	*75.00%*	*85.71%*	*92.31%*
2	*66.67%*	*75.00%*	*81.82%*	*90.00%*	*94.74%*	*72.73%*	*80.00%*	*85.71%*	*92.31%*	*96.00%*
3	**75.00%**	**81.82%**	**87.10%**	**93.10%**	**96.43%**	**80.00%**	**85.71%**	**90.00%**	**94.74%**	**97.30%**
4	**80.00%**	**85.71%**	**90.00%**	**94.74%**	**97.30%**	**84.21%**	**88.89%**	**92.31%**	**96.00%**	**97.96%**
5	**83.33%**	**88.24%**	**91.84%**	**95.74%**	**97.83%**	**86.96%**	**90.91%**	**93.75%**	**96.77%**	**98.36%**
6	**85.71%**	**90.00%**	**93.10%**	**96.43%**	**98.18%**	**88.89%**	**92.31%**	**94.74%**	**97.30%**	**98.63%**
7	**87.50%**	**91.30%**	**94.03%**	**96.92%**	**98.44%**	**90.32%**	**93.33%**	**95.45%**	**97.67%**	**98.82%**
8	**88.89%**	**92.31%**	**94.74%**	**97.30%**	**98.63%**	**91.43%**	**94.12%**	**96.00%**	**97.96%**	**98.97%**
9	**90.00%**	**93.10%**	**95.29%**	**97.59%**	**98.78%**	**92.31%**	**94.74%**	**96.43%**	**98.18%**	**99.08%**
10	**90.91%**	**93.75%**	**95.74%**	**97.83%**	**98.90%**	**93.02%**	**95.24%**	**96.77%**	**98.36%**	**99.17%**

Provided there are no extraordinary drill and blast or other costs, manifestly best mining economics will be achieved at the minimum combined loading and hauling costs. In the case of loaders the maximum hypothetical intrinsic performance is when the loader cycles at optimum cycle time continuously. Practical time losses for shift schedules, preventative and breakdown maintenance, non-utilization - both managed and job inefficiency - reduce hypothetical intrinsic performance to a practical expected performance. Considering all these time losses as fixed for our purposes there is then the efficiency of the actual truck loading operation. Further time is lost for face housekeeping by the loader or delay to allow access for a clean-up dozer or other support equipment. There is also the possibility of under trucking for a number of reasons; or truck performance variance causing bunching resulting in loader waiting time. For purposes of this current discussion let us also consider these and any other lost times as fixed and otherwise accounted for.

Time will be lost for truck change over – the focus of this discussion. Typically change over for large mining trucks is in the order of, say, 45 secs of truck time (OEM typically recommend 0.7 mins – 42 secs - as an average time). Hays advises that "Spot time for large rear dump trucks usually is between 0.40 to 0.70 minutes. Spot time for tractor-trailer units may range from 0.15 to 1.00 minutes, depending on the method and need for backing." (Hays, 1990). Truck exchange time tends to increase with truck size. Since 1990 mining trucks of 218 tonnes with more recent development up to 340 tonnes have been available from the major OEM and their dealers. Turn and spot times for these larger to ultra large trucks is considered typically higher in the range 45 to 50 seconds.

During truck "turn-and-spot" time the loader is typically filling the first bucket load. Depending on the safety protocol practiced, the loader will lose the nett exchange time - between a maximum of the "turn-and-spot" time and the difference between "turn-and-spot" time and loader cycle time. That is from 45 seconds to – for single side loading - approaching a notional minimum of 10 seconds (time taken whilst the truck is reversing to "spot") to zero for double side loading. The loading exchange efficiency η_{LE} will vary in accordance with:

$$\eta_{LE} = \frac{\textit{Number of passes to load truck} \times \textit{cycle time}}{\textit{Number of passes } \mathbf{x} \textit{ cycle time plus nett exchange time (45 seconds to approach zero).}}$$

In Table 3.4 four loading times have been selected corresponding to the practical range expected for the types of loading equipment considered in this thesis. The chosen range also generally covers cycle-time confidence interval for a shovel with a mean cycle time of 30 seconds and an assumed coefficient of variation (CV) = σ/μ of 0.25. For each of the four loader cycle times a range of five nett exchange time losses ranging from 45 down to 5 seconds has been analysed. A truck exchange time of 45 seconds has been adopted for this example.

Double-sided loading will tend to minimize nett exchange time. Double-sided loading will not necessarily totally eliminate this lost time particularly if the loader has to swing through an additional arc from last to first bucket. Also, double-sided loading is not always possible due to physical work area limitations.

It will be noted from Table 3.4 that, for any nett exchange time, the higher the number of passes the more efficient is the realisation of expected loading performance. Even for small nett exchange time losses, there is measurable loss of loading efficiency as the number of loading passes decreases.

What does this mean in practice? Three and four pass loading is not necessarily most efficient in terms of intrinsic productivity and unit cost for loading of mining trucks. Also, it may be better in terms of loading-cost benefit for several reasons, including those indicated; and others to be discussed later in this thesis (including truck coverage and loader availability) to have three loaders, say of 37 tonnes actual per pass loading 220 tonne trucks in 6 passes than two loaders of 55 tonnes actual per pass loading 220 tonne trucks in 4 passes. This assumes that it is possible to plan for at least three working faces at all times. In fact, at similar cycle times gross actual mine production may well be greater with the three smaller loaders. Of course the bottom line mining costs will reflect the additional costs of hauling for the additional truck cycle time as passes increase; and, manifestly the nett effect of marginally higher unit costs for smaller loading equipment that tends to be offset by the higher operating efficiency, $\eta_{LE,}$ experienced by the smaller loaders. But the only costs really affected by additional stationary time in a truck cycle time are fixed owning costs, small costs for engine idling and directly related maintenance, labour for operating and other relevant overheads non-intrinsic to trucks. Discussions later in this thesis will show these to be small.

3.2.7 Bucket Factor

As discussed in Section 3.2.4 Bucket Factor is a combination of in situ density, swell and bucket fill factors. It can be expressed in several ways including, as a ratio, as a percentage, as BCM per bucket load or, most convenient to measure as tonnes per bucket load. Combining all of the individual randomly variable properties into a single Bucket Factor is a common, convenient practice.

Traditionally loading productivity calculations have been based on volume as a direct consequence of the normal three-dimensional geometrical process in developing optimum (economic) open pit designs. Consequently all the productivity-modifying factors have traditionally been expressed in volumetric terms - ratios or percentages based on volumetric quotients. Typically productivity of loading equipment is estimated as BCM per unit of time as a basis for production scheduling and loading equipment comparisons.

Loader equipment selection can be based on volumetric calculations up to a point. But eventually there has to be a conversion to weight calculations to ensure compliance with rated suspended load for rope shovels, rated bucket plus load weight for hydraulic shovels and backhoes and allowable bucket payload weight in wheel loaders for stability considerations when tramming and articulating. OEM apply design limits for component life and operational criteria such as stability, steering and braking – all of which are gross machine weight (GMW) and payload sensitive.

As discussed in Section 3.3, productivity of mining trucks is a function of GMW.

To determine mining product quantities and facilitate metallurgical control and management of ore treatment process, ore volumes must be converted to tonnes by applying an estimated In situ density provided by a suitable number of ARD determinations for each ore and waste material type.

It is important to note that, in the process of using volumetric calculations in estimates for the purposes of feasibility determination or budget compilation, an error in the determined ARD compared with the actual in situ density as realised through the mining and product recovery processes will have effect equivalent to an error of similar magnitude in ore reserve grade, mill head grade or product recovery; also effect equivalent to a reduction in product price of similar proportion. The importance of determining, as accurately as possible, a suite of ARD data that can be used as a reliable basis for estimating in situ ore density is obvious. The availability of further ARD data that effectively represent the in situ density of waste materials to reliably determine productivity and operating costs is also obvious. As evidenced in some detail in discussions on in situ density to follow, fortuitously in situ density (ARD) is a low-dispersion variable so adequately representative samples for in situ density can be relatively small in number of observations (Harr, 1977).

Intrinsic performance of trucks is dependent on gross machine weight (GMW) that includes the randomly variable payload weight when loaded. So for purposes of truck selection and all relevant calculations a reliable estimate of the in situ density and

load factor (inverse of swell factor) of all materials to be mined and transported by trucks must be determined (Cat PHB, 2004).

Three variable properties influence bucket factor:

1. ***In situ* density** of material in place before free-digging or conditioning by blasting or other fragmentation method estimated from samples subjected to laboratory determinations to provide “apparent rock density” (ARD) values that can be processed to provide mean ARD for mining-performance estimating purposes and, just as importantly to provide density statistics for resource assessment and ore reserve definition. In situ density, measured as ARD, is a natural material characteristic that can only be observed and utilized in volumetric or weight estimating processes.

   Average in situ density generally increases with depth through horizons of transported rock detritus through transition zones to the so-called oxidation cut-off horizon and into practically homogeneous fresh rock basements; i.e., inversely with the degree of weathering; except that surface lateritic development can produce higher density capping with in situ density approaching but lower than fresh rock density. There can be some sub-lateral variation of in situ density in fresh rock zones due to litho-logical and mineralogy changes and different geological history of individual stratigraphic elements.

   Within any homogeneous horizon of fresh or slightly altered rock in situ density will vary dependent on the mineralogical compositional changes from point to point.

   Milton Harr has reported on variability of solid rock density investigations by others (Harr, 1977). Work by Stomatapoulas and Kitzias (1973), where 991 tests of a variety of soils types from many countries over some ten years yielded the following results for rock solids:

   - Mean – μ 2.68
   - Standard Deviation – σ 0.06
   - Coefficient of Variation - CV 2.2%

Harr also tabulated in situ density data for further testing of rock solids (attributed to a paper by G M Hammitt, 1966) as follows:

Rock Condition	**Number Of Samples**	**μ**	**σ**	**CV%**	**Comments** Taken As:
High Plasticity	65	2.63	0.115	4.4	Weathered Zone
Medium Plasticity	65	2.66	0.060	2.3	Transition Zone
Low plasticity	65	2.69	0.054	2.0	Fresh Rock

Where:

μ = Mean in situ density

σ = Standard deviation of density distribution

CV% = Coefficient of Variation as % – **(σ / μ) * 100**

Obviously, from the above results, variability of in situ density is typically small. So, relative to the other variables that influence productivity of loading and hauling operations, in situ density can be considered low sensitivity. For definitive mine planning, feasibility studies and operating budgets, it is sufficiently accurate to obtain a small number of samples of each identifiable rock type and weathering condition, determine ARD data and correlate with industry reference data from sources such as the Field Geologists Manual, Monograph 9 published by the Australasian Institute of Mining and Metallurgy (Berkman, 1989), Caterpillar (Cat PHB, 2004) or other acceptable authorities to adopt an estimated single density value for each individual rock type within each weathering horizon. For preliminary feasibility and other lower accuracy purposes it is considered sufficiently accurate to identify different rock types geologically and weathering condition visually from drill cores and reverse circulation drilling chips and consult density tables in mining standard references and handbooks.

ARD determinations of fresh rock horizons in any geological sequence are generally performed on drill cores. ARD determination is a relatively simple and reliable procedure using a time-honoured water immersion technique.

Determinations of ARD in the weathered zone of any horizon are more difficult but methods are available. Historically ARD determinations of weathered and particulate material, generically termed soils, have been considered less reliable and tended to be avoided, relying instead on experienced estimates based on tables of in situ densities for common rock material descriptions supplied by commercial and/or technical sources or experience. (Cat PHB, 2004). Acceptable methods for determination of soil density include:

- Sand cone method.
- Oil method.
- Balloon method.
- Cylinder method.
- Nuclear density moisture gauge. (Cat PHB, 2004)

The first four methods involve removing a soil sample from the bank state, weighing the sample and determining the volume of the space after removal of the sample. The nuclear method involves absorption of gamma rays that can deliver an interpreted soil density and moisture content.

The determination of soil density is more applicable to construction borrow-pit areas than mining. Generally for mining estimates the historical method of adopting industrial reference data as a source for estimating ARD of weathered and other unconsolidated materials will be sufficiently accurate.

2. **Swell** is the term for the effect of voids created in the process of, or fragmentation prior to, excavation of earth and rock material. It is an alternative term for the Voids Ratio defined as the volume of voids to the volume of solids (Harr, 1977). Swell is expressed as a ratio of voids to in situ volume or as a percentage. The following relationships define swell and other related terms and factors:

Swell	= [(In situ Volume + Voids) / In situ Volume] - 1
	= (In situ Density / Loose Density) – 1
Loose density	= In situ Density / (1 + swell)
Swell Factor	= In situ Density /Loose Density)
Load Factor	= 1/Swell Factor

“Loose density” is the term adopted throughout this thesis. An equivalent term “bulk material density” is often used as an alternative.

Many authorities have published listings of in situ and loose densities, swell, swell factor and load factor for a wide range of basic materials that have stood the test of time for application in estimates of mining and construction earth moving equipment performance. The Caterpillar Performance Handbook, a traditional industry reference, provides a typical example. (Cat PHB, 2004).

Well-graded aggregates of blasted materials have lower proportion of voids than do poorly graded materials consisting predominantly of a narrow size range. Poor grading, particularly a predominance of large particles is a function of individual mechanical material characteristics, efficacy of blasting or other fragmentation process and any extraordinary influences such as in situ voids, open faces and weathering profiles. Manifestly swell is a random variable dependent on complex and discrete influences that may be managed and modified. But many of the influences are intrinsic characteristics. Only efficacy of drilling and blasting or other fragmentation processes can be substantially managed and modified.

In summary swell variability (alternatively voids ratio), as evidenced by the effect on load factor, depends essentially on efficacy of blasting or other fragmentation method, the degree of fragmentation, grading of fragmented materials and the size and shape of particles, particularly the larger size particles.

Traditionally swell of blasted material has been assumed to apply as a constant through the truck loading process. Intuitively, vigorous treatment of materials throughout the mining process results in swell changing during the various sequential mining activities. Certainly when material is dumped onto a waste dump or an ore stockpile there is some degree of segregation of particle sizes and compaction. So any discussion on swell should identify the stage of operations and status of material being mined.

Assessing fragmentation and shape criteria at the scale of particles being considered is possible but time consuming and difficult. Production of dimension stone for marine structures and under-water pipeline protection has justified development of techniques and standards for measuring and comparing particle shape and form characteristics. Some applications of photographic techniques have been developed and are available. But such improvement activity is currently generally in the research and testing phases and not general practice.

Of the factors that affect the BCM or tonnes in bucket loads, estimation or assigning a swell value is intuitively the most imprecise, tends to be subjective, and is considered to be the least reliable and to have the greatest potential for error. Certainly particle size and grading of blasted material is manageable to some degree by quality attention to drilling and blasting; but the over-riding natural fragmentation characteristics of rock materials both in size and shape at the micro level of preparing material for loading and hauling tend to be relatively random and unpredictable. Inherent fracture patterns in rock units within a stratigraphic group can be a valuable macro indication of natural failure characteristics in the geotechnical context. In preparing rock materials for loading and hauling by drilling and blasting the natural fracturing and incipient jointing patterns influencing sizing and particle shape within the required fragmentation grading is relatively micro in scale. Whereas jointing surveys and subsequent technical assessment prediction are analytical, at the micro level of rock fragmentation in terms of size and shape for determinations of load and haul productivity, research and investigation tends to be empirical. We seem to be reliant on experience for much of the understanding of how drilling and blasting and loading and hauling interact.

Swell is discussed above in the context of bucket loads. References to swell need to be identified with a specific situation or appropriately qualified. That is, swell after blasting, into bucket, into truck, onto waste dump and long term waste dump or ore stockpile volume. Typically the greatest swell-effect is after blasting or otherwise fragmenting rock materials; likely increases but slightly into bucket, remains practically constant or slightly reduces into trucks, reduces on tipping onto dumps and stockpiles and tends to be further compacted in time by traffic and moisture ingress.

From the above discussion it is concluded that single value estimates for swell, or predictions of how swell varies statistically, are questionably reliable and should be treated with caution.

3. **Bucket Fill** is a variable, generally volumetric, measure of the efficiency of loading operations. Bucket capacity is designed to a standard as discussed in Section 3.2.4. Operators endeavour to lift full bucket loads for each pass except where, by judgment of volumetric appearance of truck loading, or by feed-back from truck payload sensing systems, truck overloading is likely. Buckets are designed so that, when filled, excess initially spills over the teeth or cutting lip and finally over the ends of the bucket and, by design, never over the back of the bucket for obvious safety reasons. Backhoes dig below the bench but similar bucket design criteria are used. One common problem with backhoes in hard rock materials is damage to the underside of the stick from over filled buckets – particularly where bucket fill is achieved low in the face, when boom and stick are extended reaching for the load and the full bucket tucks up under the backhoe stick. Bucket fill for all loader types is significantly influenced by the stacking characteristics of the material that increase voids within the bucket, i.e., the variability of swell – or voids ratio. Obviously material that breaks into large slabs and elongated particles will tend to increase voids within blasted material, a phenomenon that will persist, possibly with some variation, into the loader bucket. The degree of increase or decrease in swell within the loader bucket is considered to be dependent on the relationship between maximum particle dimensions, internal bucket dimensions and bucket shape.

   Bucket fill is also, to a large degree, dependent on the skill and application of the operator. Generally the first bucket load is a full load as the operator takes advantage of the truck exchange time for face house keeping, including setting up a heaped full first-load for the next truck. Intermediate loads are also as full as possible; but the operator is sustaining best possible cycle times with the last load sometimes a partial load to top up the truck. These first and last-load biases must be considered in any statistical analysis of bucket load data.

Bucket fill factor is a random continuous variable that is not strictly independent due to the degree of bias that operators can exercise. But for large sample of bucket loads and related truck loads the application of bias tends to be random and continuous. In effect small samples of bucket loads that coalesce as truck payloads have representative records in separate sub samples of first loads, intermediate and last loads that exhibit differing descriptive statistics. It is possible to analyze such data and draw valid conclusions from the trends so determined.

3.2.8 Bucket Loads

Preliminaries

Bucket loads of material prepared by drilling and blasting or ripping and pushing by dozer for loading into mining trucks are basic fundamental quantities for open pit production. Research of the volumetric and mass movement characteristics of load and haul operations involves statistical analysis of relatively large samples of bucket loads that are transported as small samples - truck payloads.

Where load and haul activities involve different loading equipment and materials significantly different in physical characteristics or preparation by way of fragmentation for loading, each identifiable set of conditions must be viewed as separate sub-samples of bucket loads and related truck payloads.

The actual load in each bucket pass in the process of loading mining trucks is determined by the several factors discussed in Section 3.2.7. These factors are considered to be significantly interactive. The underlying fragmentation characteristics that determine swell likely influence the ability of operators to achieve consistently high bucket fills and may also inhibit digability by effecting penetration of the face when crowding to achieve bucket load. Generally factors influencing bucket load variability are random, difficult to both predict and to control.

In the process of loading trucks the underlying variability of individual bucket loads will determine the variability of truck payloads.

Practical experience indicates that the following fundamentals need to be kept in view:

1. In situ density is a natural given. Variability down through the weathering profile or across the stratigraphic members associated with any potentially economic mineral occurrence is pre-determined and only requires acceptably accurate measuring procedures.

2. Loading equipment operators will use their best efforts to fill the bucket on every occasion, except when visual assessment or truck-payload sensing facilities indicate a partial bucket load is required. Experienced operators are consistently good at loading trucks. It is optimistic to attempt control of payload dispersion through application of a subjective loading technique. The relationship between payload dispersion and material preparation for loading is discussed in Section 3.3.6. If material to be loaded is well prepared payload dispersion is unlikely to be excessive. Influence of loader operator skill on truck payloads is revisited in Section 3.3.5 and Section 3.3.8.

3. Any excessive payload is a direct result of the last bucket load (or loads in the absence of a payload indication facility) placed on a truck.

4. Potential for over loading trucks is dependent on the proportion of a truck payload that each bucket load represents. That is, truck payload dispersion has an inverse relationship to the number of bucket passes to load each truck. Statistical analysis shows that the relationship between bucket and payload data is consistent with Sample Theory and the Central Limit Theorem (MPN 3, Mathematical Principles – Notes, Volume 2, Appendices}; and also confirms the intuitive concept that the lower the number of passes the more disperse will be the distribution of resulting truck payloads..

It is most important to be able to predetermine the variability of truck payloads to:

- Comply with design standards for steering and braking safety that set limitations on gross machine weight (GMW) for mining trucks – conveniently interpreted in terms of payload limitations by considering the truck tare as generally constant; but with due allowance for debris, carry-back and added mass due to body repairs;

- Comply with target truck-payload limitations particularly to accommodate overloading and so avoid latent accelerated wear-and-tear of driveline componentry due to increased fuel burn; also suspension components due to excessive working stress levels;

- To limit the dispersion of payload distribution within reasonable limits so reduce variability of truck performance due to variation in GMW that in turn will ameliorate inefficiency due to bunching of trucks.
- To consistently achieve expected tyre life – tyres being considered the final driveline and suspension component for mining trucks.

So truck payloads should be retained within a specific confidence interval. A suitable key performance indicator (KPI) for purposes of analysis and control is the descriptive statistic Coefficient of Variation (CV = Standard deviation/Arithmetic Mean). The necessary CV limitation on truck payloads for the several operating standards is exactly similar to the CV limitation on the distribution of average bucket loads in each payload for a constant number of bucket passes per payload.

Truck payloads are small samples of the sub sample of bucket loads for an identifiable mining operational phase. Each small sample consists of the number of bucket passes per truck payload. Consequently the descriptive statistics of truck payloads are related to all bucket loads in the sub-sample.

Relevant practical truck payload issues are covered in additional detail in Section 3.3 and supporting descriptive statistical mathematics are provided in Mathematical Principles – Notes, MPN 1 to MPN 3, appended in Volume 2.

Previous research on truck payload dispersion and number of loading passes identified a need to obtain actual bucket load data to confirm the hypotheses developed using statistical mathematical relationships (Hardy#1, 2003).

Direct collection of bucket load data from wheel loaders has been in process of development over many years and has been of particular interest for load-and-carry operations in stockpile reclaiming. In open pit mining applications the ultimate objective is to be able to predict and control truck payloads. Consequently application of wheel loader weighing systems in mining applications is not as favoured as truck payload sensing systems. Currently an alliance between an OEM and a research organization is testing a bucket weighing system for rope shovels. The system is reported to be close to commercial release.

Ready collection of bucket load data has not been practically possible until recent refinements of on-board load sensing systems has facilitated collection of truck-payload building data. Cumulative truck payload is recorded after each bucket pass. OEM and their dealers offer information products, specifically onboard sensing and information management that integrated with mining information systems have the facility to provide constant productivity information from trucks, loaders and other earthmoving equipment. Amongst other things, these information systems can record and make available, in real time, payload increase for each bucket load. Onboard data processing and recording downloads to local data servers that can be accessed, given security authorization, by telephone connection anywhere in the world. Recent broadband transmission facilities have further enhanced speed and reliability of data transfer.

Caterpillar, supplier of the VIMS/TPMS (Vital Information Management System/Truck Payload Management System, hereinafter referred to generically as VIMS) used to collect bucket load and truck payload data for this research, states accuracy of VIMS as 95% of loads +/-5%. It is believed that this is typically representative of the current accuracy-status of on-board truck payload sensing systems offered by all OEM of mining trucks. Although limited in precision these systems do provide data sufficiently accurate and useful for comparisons and to develop understanding of the underlying relationships between the various loading factors that influence truck performance. A more detailed discussion on accuracy limitations of truck payload measuring using available technology is provided by earlier research on four-pass loading (Hardy, 2003). A copy of the relevant paper is provided in Supplementary Information appended.

Further discussion on reasons for inaccuracy of VIMS and equivalent systems is provided in Section 3.3.7.

To facilitate the research, access was obtained to data and information from a 220 tonne truck fleet as described below.

Bucket load data observations were conducted in two phases:

1. Real time observations using Caterpillar's Minestar software to access VIMS truck payload data were recorded in real time over two sessions each of several hours duration. This provided valuable insight into the

detailed working of the system and highlighted the significant differences between initial, intermediate and final bucket loads. Some limitations of the system were also identified and consideration given to management of the limitations. Results of this study are provided by Table 3.5; also Tables 3.6 and 3.7 - loading by 700 tonne hydraulic shovel: - and Table 3.8; also Tables 3.9 and 3.10– loading by 190 tonne wheel loader.

2. Distribution histograms for bucket loads have also been developed in Figures 3.2 and 3.2A. This phase of the research proved to be time consuming, produced only an unacceptably small number of observations; and inconclusive, but promising, results.

3. Data from a number of operating days was down loaded from VIMS records for 700 tonne hydraulic shovels loading 220 tonne trucks, summarized in Table 3.11 and analysed to provide the descriptive statistics in Table 3.12.

Table 3.6 - Bucket Load Summary 16 February 2004
700 tonne Hydraulic Shovel Loading 220 tonne Trucks

First Loads	Intermediate Loads			Last Loads	
1	**2**	**3**	**4**	**4**	**5**
69.1	51.8	60.3		40.2	
52.8	56.2	51.8		42.4	
43.8	41.8	54.3		60.6	
70.6	54.5	49.7		47.2	
71.7	42.7	53.0	54.9		40.6
83.7	55.2	52.4		47.0	
58.0	57.5	71.4		46.5	
88.7	57.3	58.5		35.1	
62.3	49.7	38.9		51.2	
		56.5	43.5		32.7
60.7	35.8	45.0	38.8		27.9
69.1	51.8	60.3		40.2	

Table 3.7 – Statistical Analysis of Bucket Load Data 16 February 2004
700 tonne Hydraulic Shovel Loading 220 tonne Trucks

	First Loads	Inter-med. Loads	Last Loads	Except First Loads	All Loads	Pay-loads	Number Bucket Loads
Number of Records	10	24	11	35	45	11	47
Maximum Value	88.7	71.4	60.6	71.4	88.7	262.9	5.0
Minimum Value	43.8	35.8	27.9	27.9	27.9	200.5	4.0
Range	44.9	35.6	32.7	43.5	60.8	62.4	1.0
Average of Range	66.3	53.6	44.3	49.7	58.3	231.7	4.5
Arithmetic Mean	**66.1**	**51.3**	**42.9**	**48.7**	**52.5**	**224.0**	**4.3**
Median	65.7	52.7	42.4	49.7	52.4	222.0	4.0
Std Dev Sample	13.59	8.17	9.13	9.25	12.56	19.62	0.47
Coefficient of Variation	**0.21**	**0.16**	**0.21**	**0.19**	**0.24**	**0.09**	**0.11**
Skewness	0.16 To Right	0.02 To Right	0.24 To Right	-0.09 To Left	0.69 To Right	-0.09 To Left	0.12 To Right
Kurtosis	-0.25 Platy-kurtic	0.40 Lepto-kurtic	0.28 Lepto-kurtic	0.01 Lepto-kurtic	0.94 Lepto-kurtic	0.01 Lepto-kurtic	0.14 Lepto-kurtic

Table 3.9 - Bucket Load Summary - 16 February 2004
190 tonne Wheel Loader loading 220 tonne Trucks

First Loads	Intermediate Loads								Last Loads		Number Bucket Loads
Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	
1	2	3	4	5	6	7	8	7	8	9	
The only four first loads recorded were anomalously high – likely combined in the second load. So both first and second loads were filtered out.											
		18.3	21.3	28.2	24.1	24.0			21.4		8
		29.3	26.0	26.7	25.7	26.3			22.6		8
			21.9	22.7	21.8	16.7			16.5		8
		20.0	25.9	21.7	23.3	24.6			23.4		8
			20.9	20.1	20.9	25.2			14.4		8
		23.5	20.9	17.7	19.8	24.0	19.5			17.3	9
		21.6	23.7	26.0	24.0	24.3	17.2			21.4	9

Table 3.10 – Statistical Analysis of Bucket Load Data - 16 February 2004
190 tonne Wheel Loader loading 220 tonne Trucks

	First Loads	Inter-med. Loads	Last Loads	Except First Loads	All Loads	Pay-loads	Number of Bucket Loads
Number of Records		35	7	42	42	7	7
Maximum Value		29.3	23.4	29.3	29.3	220.0	9.0
Minimum Value		16.7	14.4	14.4	14.4	189.5	8.0
Range		12.6	9.0	14.9	14.9	30.5	1.0
Average of Range		23.0	18.9	21.9	21.9	204.8	8.5
Arithmetic Mean		**22.8**	**19.6**	**22.3**	**22.3**	**204.3**	**8.3**
Median		23.3	21.4	22.3	22.3	206.9	8.0
Std Dev Sample		**3.08**	**3.46**	**3.33**	**3.33**	**10.88**	**0.49**
Coefficient of Variation		**0.14**	**0.18**	**0.15**	**0.15**	**0.05**	**0.06**
Skewness		-0.08 To Left	-0.46 To Left	-0.24 To Left	-0.24 To Left	-0.06 To Left	1.23 To Right
Kurtosis		-0.46 Platy-kurtic	-1.66 Platy-kurtic	-0.22 Platy-kurtic	-0.22 Platy-kurtic	-1.07 Platy-kurtic	-0.84 Platy-kurtic

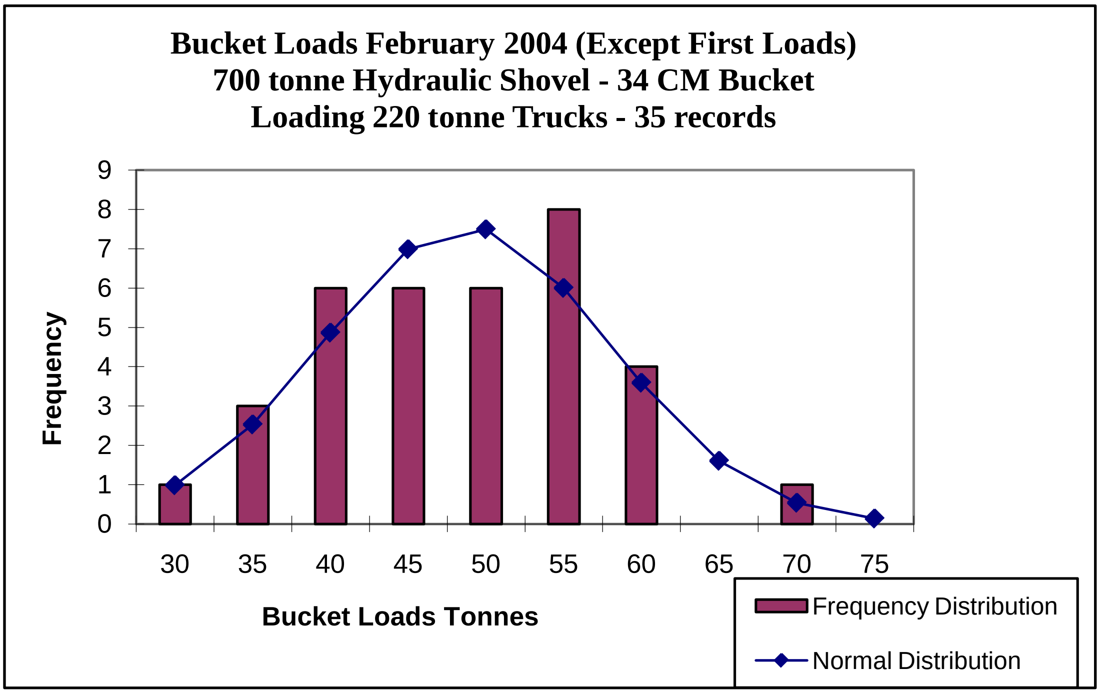


Figure 3.2 From Table 3.7

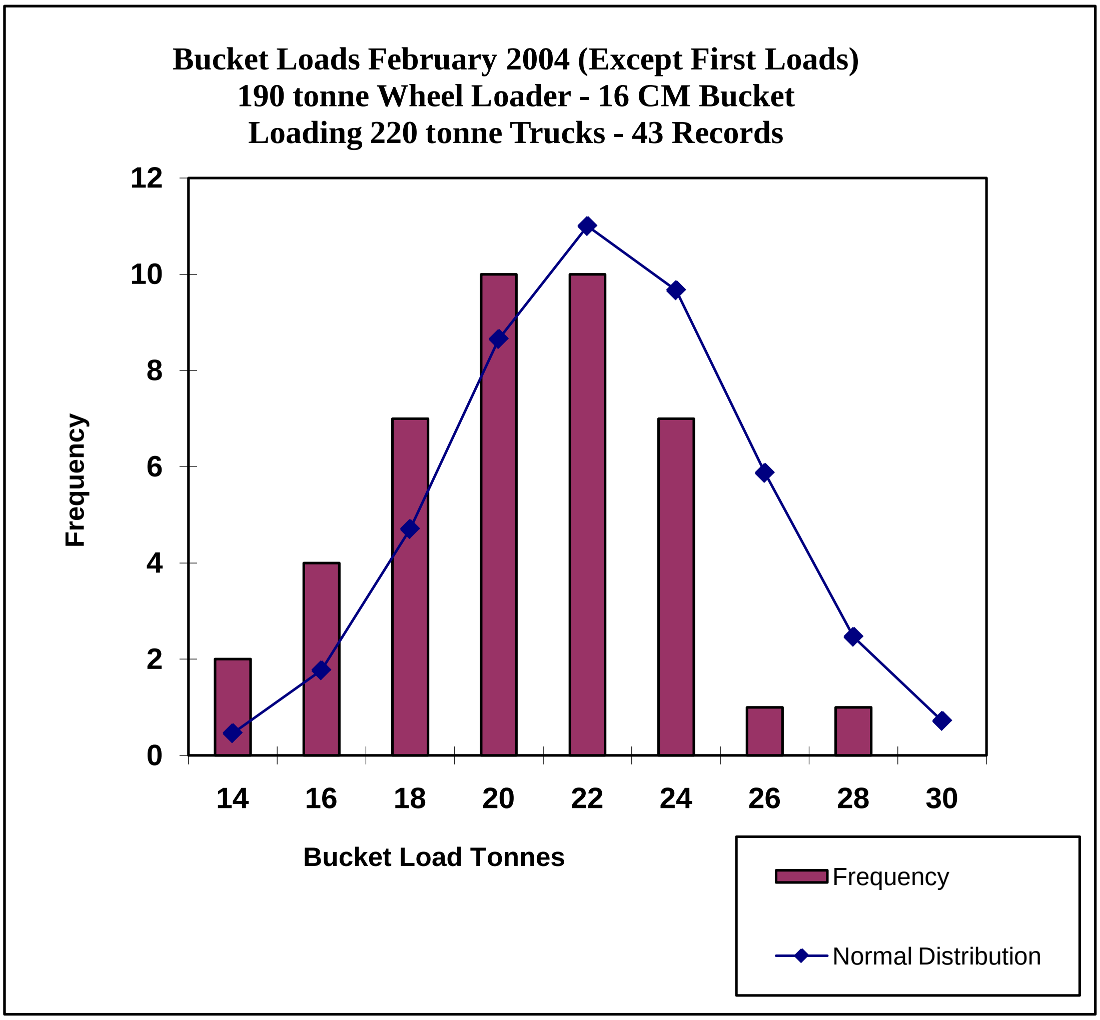


Figure 3.2A From Table 3.10

Table 3.11 Bucket Load Summary - April 2004 – Part 1
700 tonne Hydraulic Shovel Loading 220 tonne Trucks

First Loads	Intermediate Loads					Last Loads		Number Bucket Loads	Average Bucket Loads
Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes		Tonnes
63.8	60.3	49.3			39.8			4	53.3
55.6	44.4	60.7	55.3			27.8		5	48.8
80.1	41.7	61.8			40.7			4	56.1
68.6	15.8	55.8	68.7			32.3		5	48.2
64.8	50.3	31.2	47.2			25.1		5	43.7
47.5	30.6	63.9	69.5			20.0		5	46.3
		55.5	72.2			19.4		5	51.8
77.1	67.5	49.4			41.2			4	58.8
51.3	58.8	51.4	50.0			20.8		5	46.5
64.3	47.4	59.8	31.7			31.7		5	40.6
		49.1	52.9			31.6		5	45.8
57.6	48.0	33.5	33.4	15.2			17.9	6	34.3
		59.0			58.7			4	59.2
60.7	49.1	52.2			61.7			4	55.9
75.8	43.6	43.5	59.2			27.3		5	48.2
		73.7			24.9			4	55.0
75.1	45.3	32.6	42.5			36.5		5	46.4
60.4	59.0	45.3	43.3			37.1		5	47.1
73.3	47.8	77.2			46.5			4	61.2
64.6	53.2	46.5			42.5			4	51.7
72.8	59.6	62.8			46.2			4	60.4
		73.7			39.3			4	59.1
64.0	58.8	75.1			53.4			4	62.8
58.2	59.0	54.4			48.4			4	55.0
56.1	38.1	68.4	32.7			33.2		5	44.0
63.1	63.7	67.2			51.8			4	61.5
57.3	49.4	64.7	43.2			19.0		5	45.8
		32.9	38.5			35.6		5	41.7
33.3	24.7	36.2	39.0	47.8			53.2	6	39.1
58.8	68.1	38.7			35.9			4	50.4
68.1	54.5	47.9			37.0			4	51.9
88.5	62.4	55.8			48.1			4	63.7
48.4	19.9	58.0			54.5			4	45.2
69.1	57.3	39.9			52.8			4	54.8
69.0	55.5	58.5			45.3			4	57.1
59.0	48.8	57.5			41.6			4	51.8
63.5	29.4	30.5	38.2	30.5			22.4	6	35.7
85.5	60.2	60.3			32.3			4	59.6
		69.6			41.2			4	61.4
62.2	52.7	49.4			35.1			4	49.8
		62.7			47.2			4	62.0
78.5	36.0	44.9	50.8			30.0		5	48.1
		48.7	50.0			31.7		5	46.9
35.0	35.4	49.3	43.5			41.1		5	39.4
70.8	81.9	56.8			42.0			4	62.9

Part 2									
700 tonne Hydraulic Shovel Loading 220 tonne Trucks									
First Loads	**Intermediate Loads**					**Last Loads**		**Number Bucket Loads**	**Average Bucket Loads**
Tonnes	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**		**Tonnes**
67.3	41.2	57.4			58.4			4	56.1
		58.7			52.3			4	55.1
56.0	50.9	53.5			47.1			4	51.9
		51.7	49.3			46.9		5	45.7
67.1	50.9	51.5			49.7			4	54.8
		55.8	47.2			26.4		5	50.9
51.8	48.8	56.0			48.8			4	51.3
61.7	44.9	45.5	43.6			24.7		5	42.7
60.4	44.7	63.1	50.4			39.3		5	49.7
		50.8	47.3			39.6		5	46.4
78.6	41.1	51.3	35.3			14.1		5	43.1
71.4	46.1	32.6			51.9			4	50.5
71.5	48.4	43.5	39.6			53.6		5	48.1
60.0	49.8	53.4			46.5			4	52.4
53.6	50.8	45.7	37.3			28.8		5	41.8
72.3	50.7	53.4			37.3			4	53.4
		44.4			46.2			4	51.9
67.3	43.7	33.2			42.9			4	46.8
65.3	51.0	46.3	33.0			46.4		5	45.8
63.6	58.5	54.7			45.1			4	55.5
73.1	52.8	44.2	46.1			23.1		5	46.4
62.7	60.2	55.6			48.9			4	56.8
65.7	52.7	50.0	53.3			29.0		5	48.6
59.8	57.8	56.2			35.9			4	52.4
80.5	63.3	62.8			52.2			4	64.7
		50.4			37.8			4	49.9
		63.4			27.5			4	59.3
61.5	53.0	65.6			45.0			4	56.3

Related illustrative distribution histograms have been developed as shown in Figures 3.3 (Bucket Loads), 3.4 (Truck Payloads) and 3.5 (Shovel Passes Per Payload).

Table 3.12 – Statistical Analysis of Bucket Load Data - April 2004
700 tonne Hydraulic Shovel Loading 220 tonne Trucks

	First Loads	Intermed. Loads	Last Loads	Except First Loads	Except Last Loads	All Bucket Loads	Pay-loads	Bucket Loads	Average Bucket Loads
Number of Records	57	164	73	237	221	294	73	326	73
Maximum Value	88.5	81.9	61.7	81.9	88.5	88.5	259.0	6.0	64.7
Minimum Value	33.3	15.2	14.1	14.1	15.2	14.1	180.7	4.0	34.3
Range	55.2	66.7	47.6	67.7	73.3	74.4	78.3	2.0	30.4
Average of Range	60.9	48.5	37.9	48.0	51.9	51.3	219.9	5.0	49.5
Arithmetic Mean	**64.4**	**50.3**	**39.0**	**46.8**	**53.9**	**50.2**	**225.4**	**4.47**	**51.2**
Median	64.0	50.4	46.5	47.9	53.5	50.4	227.3	4.00	51.3
Std Dev Sample	**10.54**	**11.64**	**11.17**	**12.60**	**12.93**	**14.07**	**17.56**	**0.58**	**6.90**
Coefficient of Variation	**0.16**	**0.23**	**0.29**	**0.27**	**0.24**	**0.28**	**0.08**	**0.13**	**0.13**
Skewness	-0.45 To Left	-0.23 To Left	-0.22 To Left	-0.19 To Left	-0.16 To Left	-0.10 To Left	-0.16 To Left	0.80 To Right	-0.09 To Left
Kurtosis	1.29 Lepto-kurtic	0.48 Lepto-kurtic	-0.72 Platy-kurtic	0.02 Lepto-kurtic	0.17 Lepto-kurtic	-0.13 Platy-kurtic	-0.54 Platy-kurtic	-0.33 Platy-kurtic	-0.43 Platy-kurtic

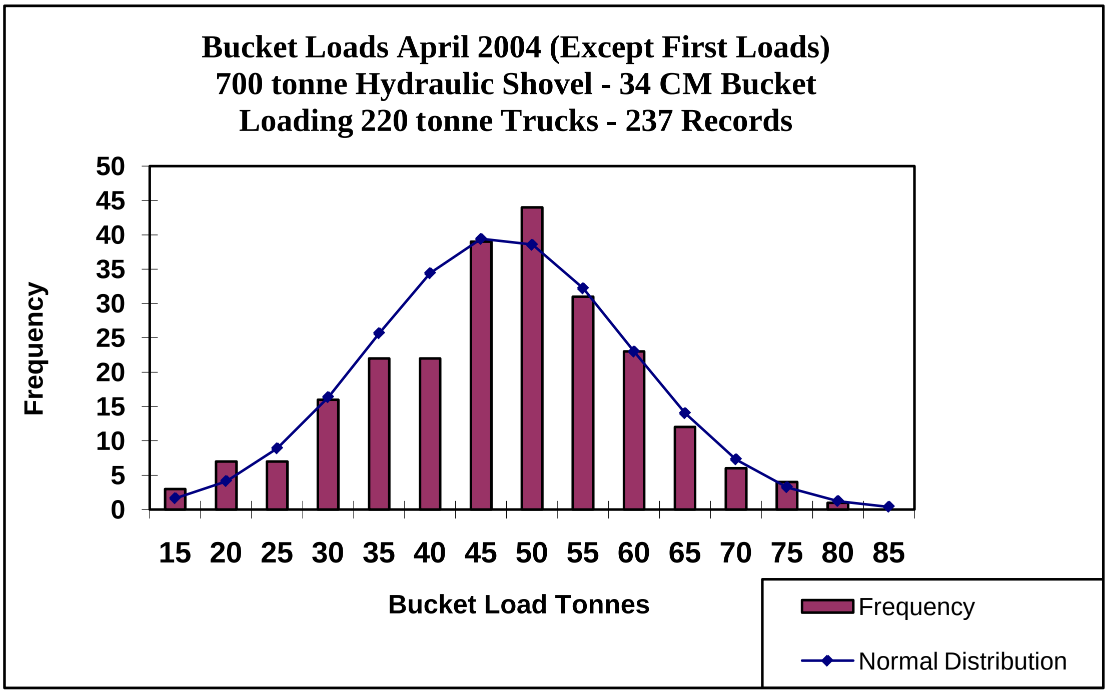


Figure 3.3 From Table 3.12

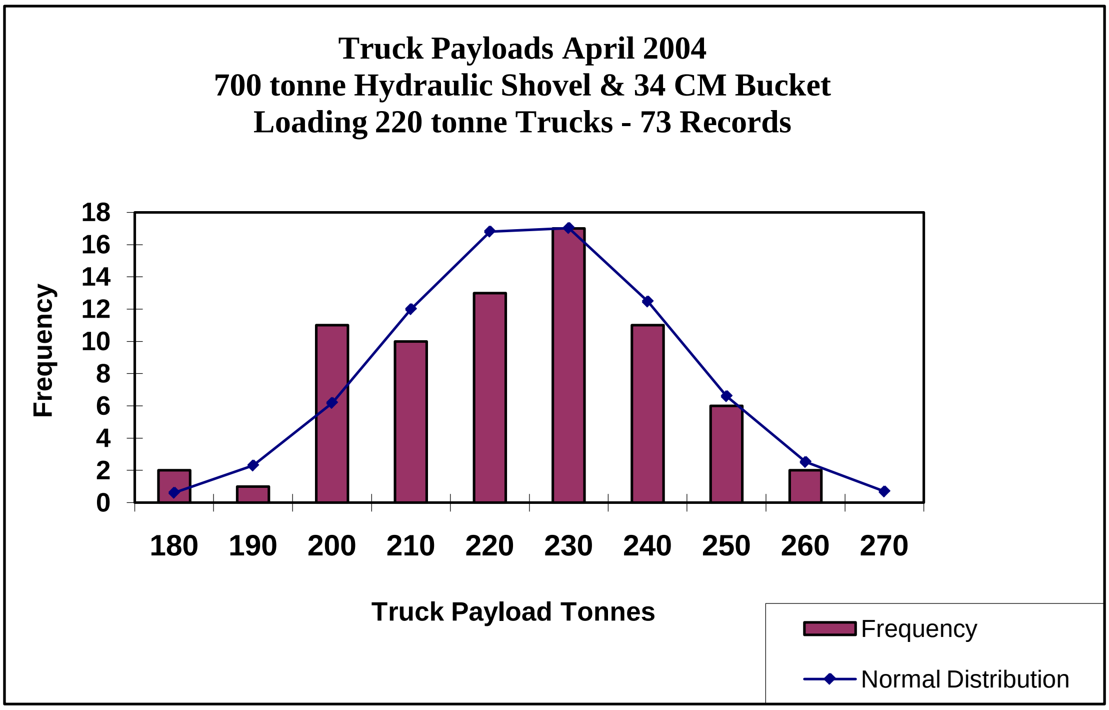


Figure 3.4 From Table 3.12

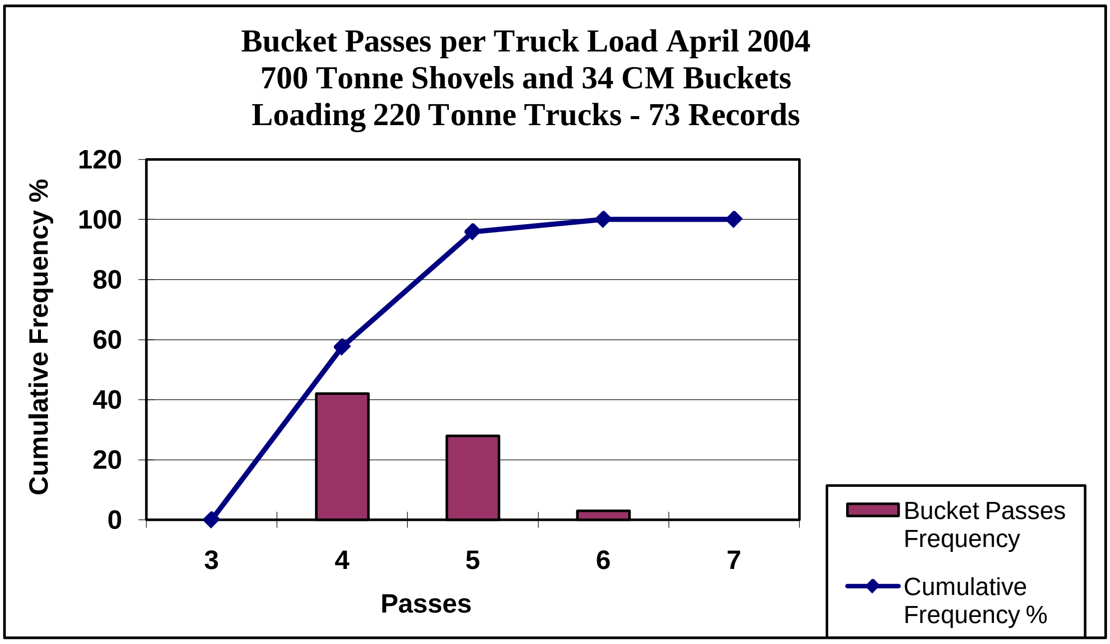


Figure 3.5 From Table 3.12

Data was also downloaded for a 190 tonne wheel loader; also loading 220 tonne trucks. A Summary of Bucket Loads is provided in Table 3.13, and the results of Statistical Analysis in Table 3.14. A histogram of the bucket load data (excepting first loads) is provided by Figure 3.6.

Table 3.14 – Statistical Analysis of Bucket Load Data - April 2004
190 tonne Wheel Loader loading 220 tonne Trucks

	First Loads	Intermed. Loads	Last Loads	Except First Loads	Except Last Loads	All Loads	Truck Payloads	Number Bucket Passes	Average Bucket Loads
Number of Records	4	51	8	59	55	63	8	8	8
Maximum Value	49.9	37.1	28.4	37.1	49.9	49.9	239.5	12.0	28.5
Minimum Value	33.8	12.6	13.5	12.6	12.6	12.6	194.4	7.0	20.0
Range	16.1	24.5	14.9	24.5	37.3	37.3	45.1	5.0	8.5
Average of Range	41.8	24.9	20.9	24.9	31.3	31.3	217.0	9.5	24.2
Arithmetic Mean	40.84	23.41	19.19	22.84	24.68	23.98	213.08	8.88	24.41
Median	39.8	22.5	19.0	22.3	22.8	22.5	213.6	9.0	23.9
Std Dev Sample	6.82	4.37	4.78	4.62	6.41	6.46	13.47	1.55	2.67
Coefficient of Variation	0.167	0.187	0.249	0.202	0.260	0.269	0.063	0.175	0.109
Skewness	0.79	0.60	0.91	0.44	1.66	1.52	0.73	0.88	0.16
	To Right	To Right	To Right	To Right	To Right	To Right	To Right	To Right	To Right
Kurtosis	0.88 Lepto-kurtic	2.03 Lepto-kurtic	0.85 Lepto-kurtic	1.33 Lepto-kurtic	4.25 Lepto-kurtic	3.99 Lepto-kurtic	1.82 Lepto-kurtic	2.18 Lepto-kurtic	0.49 Lepto-kurtic

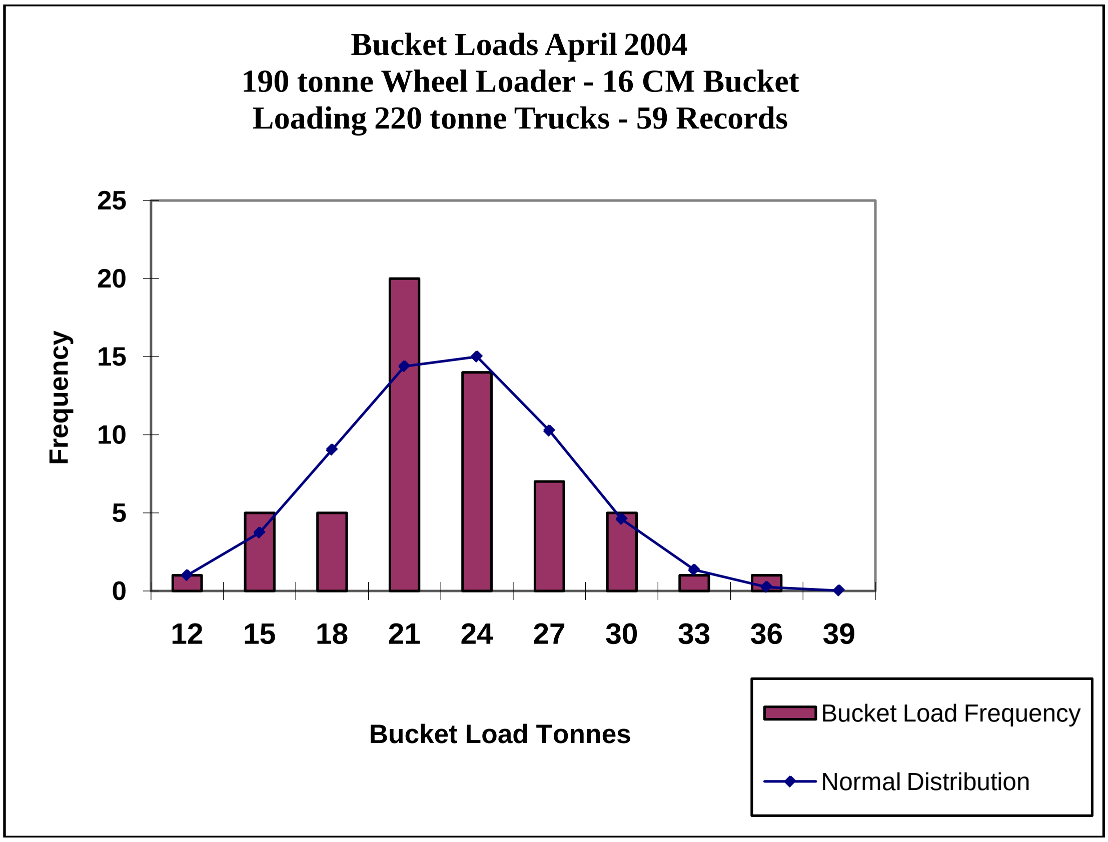


Figure 3.6 From Table 3.14

The second exercise, based on a larger sample of data, provided more consistent, reliable results from descriptive statistical analysis. This has provided confidence in the conclusions and inferences, derived from the bucket-load research.

It is important to keep in view that trucks of the same capacity – 220 tonne nominal payload - were loaded in every data observation. Also the Coefficient of Variation (CV), a most useful descriptive statistic to compare distributions of different sets of continuous random variables with different mean values, is used as a basis for comparison of the data sets and the results of statistical analysis.

Intuitively, for any range of bucket capacities, the CV should be the same for all sub-samples of bucket loads taken from material in the same condition. It was noted that generally this was true for the range of bucket-load data sets with the exception of data observed in February 2004 for the 190 tonne wheel loader loading 220 tonne trucks. It was realised that the descriptive statistics in Table 3.10, illustrated by Figure 3.2A were anomalous in comparison with other bucket-load data sets. Particularly the CV of 0.15 (Table 3.10) was unrealistically low compared with the equivalent descriptive statistic for other bucket-load data sets. At the time of recording in February 2004, the duty of the 190 tonne wheel loader was significantly different to that of loading equipment operating for the other data sets. It was re-handling stockpiled ore when the data represented by Table 3.10 and Figure 3.2A was recorded. The balance of the load and haul fleet was not operating due to heavy rainfall that forced suspension of pit operations. Data collected at this time was excluded from subsequent data consolidation and interpretations. All other data recorded was from similar operating conditions, particularly type and condition of material loaded and hauled; and forms the basis for comparisons and interpretation of the results.

The VIMS system records truck loading in incremental bucket loads but is not designed to provide accurate bucket load information (Rea David, personal communication, 2005). Best accuracy of data from VIMS is payload reweigh whilst travelling in second transmission range. Notwithstanding this implied limitation on bucket load data from VIMS for the purposes applied in the research, bucket load and payload analysis results were reasonably consistent providing confidence in the comparative facility of the data that has been utilized in this section. The relative accuracy of VIMS data in various situations is covered in more detail in Section 3.3.7.

Bucket Load Data Characteristics

It was noted that, for all truck-loading records, first loads were anomalously high. This is considered to be the result of a combination of influences including:

1. Loading-equipment operators utilize truck exchange time to ensure that the first bucket load is full – in excess of 100% bucket factor where possible. The first bucket load may be partially re-handled (utilizing part of the truck-changeover time) to ensure that the best possible bucket load is achieved.

2. The way in which the load sensing and recording system that provided bucket-load data described below accounts for truck tare.

3. Placement of the first bucket load, or any bucket load, displaced from the designed centre of gravity (CG) position of the truck payload. The load sensing system within Caterpillar's VIMS is sensitive to location of the CG of truck payloads within the truck body. This is discussed in more detail in Sections 3.3.5 and 3.3.7. Essentially, if the payload CG is forward of the design position:

    - Load on front wheels is increased and rear wheels decreased – with implications for reduced tyre life on front wheels.

    - The payload sensing system reports lower than actual payload i.e., the value that would be derived from weighing all four truck wheels and deducting pre-weighed truck tare.

    - Conversely, and importantly in the case of first bucket loads, rearward displacement behind design payload CG position of total or partial payload will be sensed by the VIMS system as lower than actual incremental or total payload in the truck body.

Notwithstanding the potential for errors in incremental and total truck payload data due to displacement of payload CG from the designed location, The data collected and subsequent analysis is valuable in providing insight and understanding of the bucket load**:** truck-payload relationship discussed in some detail below in this section.

The truck payload system is calibrated initially by manually setting a truck tare weight from sensed loads. The system can of automatically calibrate, but only within a limited range around the initial tare weight. Each time the configuration of the truck is changed, e.g. exchange of a ride strut, manual recalibration should be implemented. The reliability of the system to auto-calibrate is dependent on equipment operators and maintainers observing anomalous first loads and effecting a recalibration. So, in the absence of current correct calibration or if there is a mechanical fault in the suspension group, the payload weighing first sensed after the first bucket pass can include any error in the preset tare weight, any debris attached to the truck frame and any carry-back material in the truck body.

Caterpillar recommends an allowance of 4% of the frame weight fully-equipped ready-to-operate, equivalent to some 1.2% of GMW – amounting to some 5 tonnes for a 220 tonne truck. An on-site weighing study of 220 tonne trucks (Maloney, 2002) measured total “debris” and “carry back”. More than 100 trucks were weighed before and after washing down and clearing “carry back” from body. Results yielded average “carry back in the tray ” of 4.2 tonnes with a maximum value near 8 tonnes, equivalent to 2% of GMW. Experience indicates that the combined weight of “debris” and “carry back” varies from–zero to more than 10 tonnes depending on truck scale and site conditions. In all performance and productivity calculations for mining trucks contribution to GMW by “debris” and “carry back” requires consideration in terms of site and operational specifics. This is further considered in Section 3.3.

Caterpillar has recently amended the VIMS payload measuring software to provide for application of a load factor derived from weighbridge data. This facilitates accommodation by VIMS of body exchange to third party designs or a MSD body (described in some detail in Section 3.3.5) by correcting for errors due to fore or aft displacement of CG of truck payload.

Means of recorded last bucket loads are comparatively lower than the sample mean believed to be due to operators using the last load to top-up truck loads.

Filtering of Data

As explained above, initial review of bucket load data revealed obvious anomalously high values for first bucket loads. These anomalies and any related second bucket loads were filtered out before analysis. Because of the uncertainty of both occurrence and magnitude of error in first bucket loads, derived by difference between successive cumulative truck payloads, the remainder of first bucket loads (after filtering) were recorded and analysed separately and in combination with intermediate and final bucket loads. All logical groupings of bucket loads have been tabulated and provided as a total presentation of the data. For purposes of comparing bucket-load data sets, constructing histograms and adopting descriptive statistics as a basis for further analysis and interpretation, only data excluding first bucket loads were selected.

In addition to first and related second pass bucket loads; obviously anomalous bucket loads were filtered out of the overall bucket load samples recorded. Truck payloads are small samples of the total bucket load sample and consequently are related to all bucket loads whether first, last or intermediate and may contain bucket loads for which values have not been recorded. Similarly the number of bucket passes for each truck payload is easily derived from the data even if some bucket load values are not recorded or have been filtered out.

Interpretation and Conclusions

Data from the two phases of bucket load investigations for the 700 tonne hydraulic shovels loading 220 tonne mining trucks were combined and descriptive statistics summarized in Table 3.15 and illustrated by Figure 3.7.

Descriptive statistics derived from filtered data for the 190 tonne wheel loader adopted for the following conclusions and interpretations exclude the records from the initial collection in February 2004 for the reasons described earlier in this section. Accordingly descriptive statistics adopted for the 190 tonne loader are from Table 3.14 and Figure 3.6.

Review of the descriptive statistics in Tables 3.14 and 3.15 indicate CV’s for average bucket loads that are higher than CV’s for truck payloads in each case when it could be expected that these comparative statistics should be identical. It was realised that recorded truck payloads were small samples of bucket loads; but varying in number of bucket loads per truck payload. So the simple arithmetical relationship between average bucket loads for identical-number small samples as truck payloads is not applicable

Table 3.15 – Summary - Statistical Analysis of All Bucket Load Data
700 tonne Hydraulic Shovel Loading 220 tonne Trucks

	First Loads	Intermed. Loads	Last Loads	Except First Loads	Except Last Loads	All Bucket Loads	Truck Payloads	Number Bucket Passes	Average Bucket Loads
Number of Records	67	188	84	272	255	339	84	84	84
Maximum Value	88.7	81.9	61.7	81.9	88.7	88.7	262.9	6	64.7
Minimum Value	33.3	15.2	14.1	14.1	15.2	14.1	180.7	4	34.3
Range	55.4	66.7	47.6	67.7	73.5	74.6	82.2	2	30.4
Average of Range	61.0	48.5	37.9	48.0	51.9	51.4	221.8	5	49.5
Arithmetic Mean	64.7	50.4	39.5	47.0	54.2	50.523	225.202	4.440	51.40
Median	64.0	50.8	40.6	48.1	54.3	50.8	226.2	4.0	51.5
Std Dev Sample	10.94	11.25	10.95	12.22	12.80	13.88	17.72	0.57	6.75
Coefficient of Variation	0.169	0.223	0.277	0.260	0.236	0.275	0.079	0.128	0.131
Skewness	-0.27 To Left	-0.24 To Left	-0.24 To Left	-0.21 To Left	-0.07 To Left	-0.04 To Left	-0.07 To Left	0.85 To Right	-0.14 To Left
Kurtosis	0.87 Lepto-kurtic	0.60 Lepto-kurtic	-0.61 Platy-kurtic	0.10 Lepto-kurtic	0.28 Lepto-kurtic	0.00 Lepto-kurtic	-0.57 Platy-kurtic	-0.28 Platy-kurtic	-0.42 Platy-kurtic

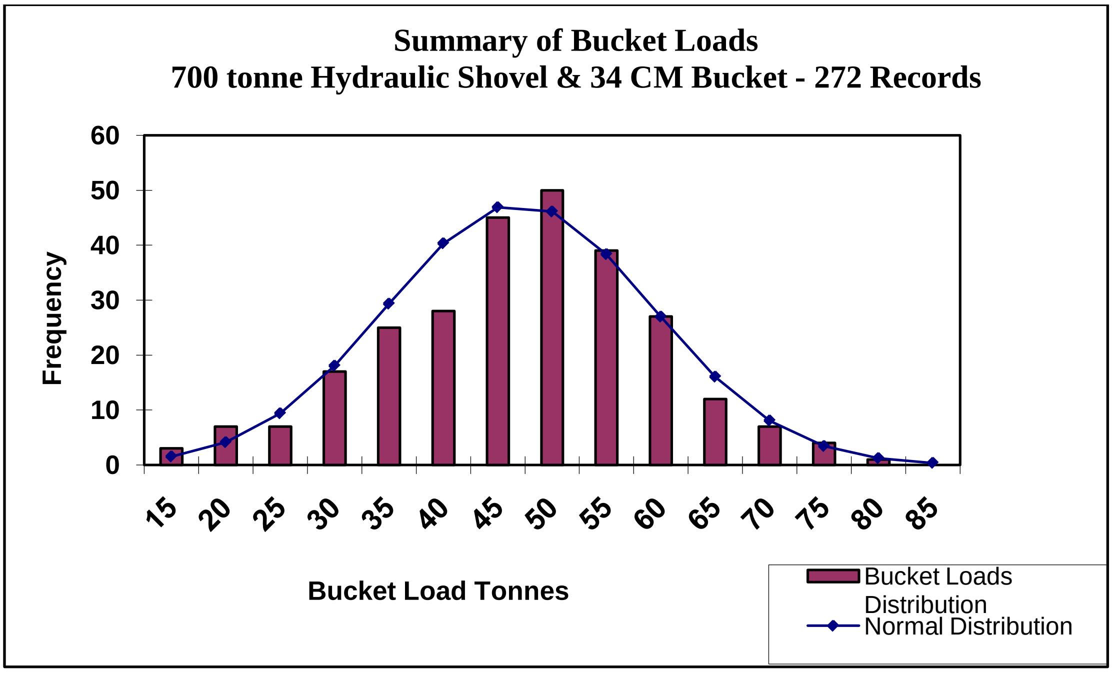


Figure 3.7 From Table 3.15

Bucket load data for the 700 tonne hydraulic shovels were sorted and two data subsets of truck payloads were selected with only 4 passes and only 5 passes. Descriptive statistics for these two sets of data are provided in Table 3.16 and 3.17; and illustrated by Figures 3.8 and 3.9 respectively.

Table 3.16 - Selected 4 Pass Bucket Load Data
700 tonne Hydraulic Shovel Loading 220 tonne Trucks

FOUR PASS SHOVEL LOADS ONLY	First Loads	Intermed. Loads	Last Loads	Except First Loads	Except Last Loads	All Bucket Loads	Truck Payloads	Average Bucket Loads
Number of Records	41	90	50	140	131	181	50	50
Maximum Value	88.7	81.9	61.7	81.9	88.7	88.7	258.8	64.7
Minimum Value	43.8	32.6	24.9	24.9	32.6	24.9	180.7	45.2
Range	44.9	49.3	36.8	57.0	56.1	63.8	78.1	19.5
Average of Range	66.3	57.2	43.3	53.4	60.6	56.8	219.8	54.9
Arithmetic Mean	66.51	55.54	45.04	51.79	58.97	55.12	222.34	55.59
Median	64.6	55.5	46.2	51.9	58.5	54.5	221.7	55.4
Std Dev Sample	10.22	9.10	7.85	10.01	10.72	11.78	18.37	4.59
Coefficient of Variation	0.154	0.164	0.174	0.193	0.182	0.214	0.083	0.083
Skewness	0.32	0.18	-0.18	0.14	0.36	0.31	0.02	0.02
	To Right	To Right	To Left	To Right	To Right	To Right	To Right	To Right
Kurtosis	0.16	0.76	0.22	0.39	0.47	0.27	-0.66	-0.66
	Leptokurtic	Leptokurtic	Leptokurtic	Leptokurtic	Leptokurtic	Leptokurtic	Platykurtic	Platykurtic

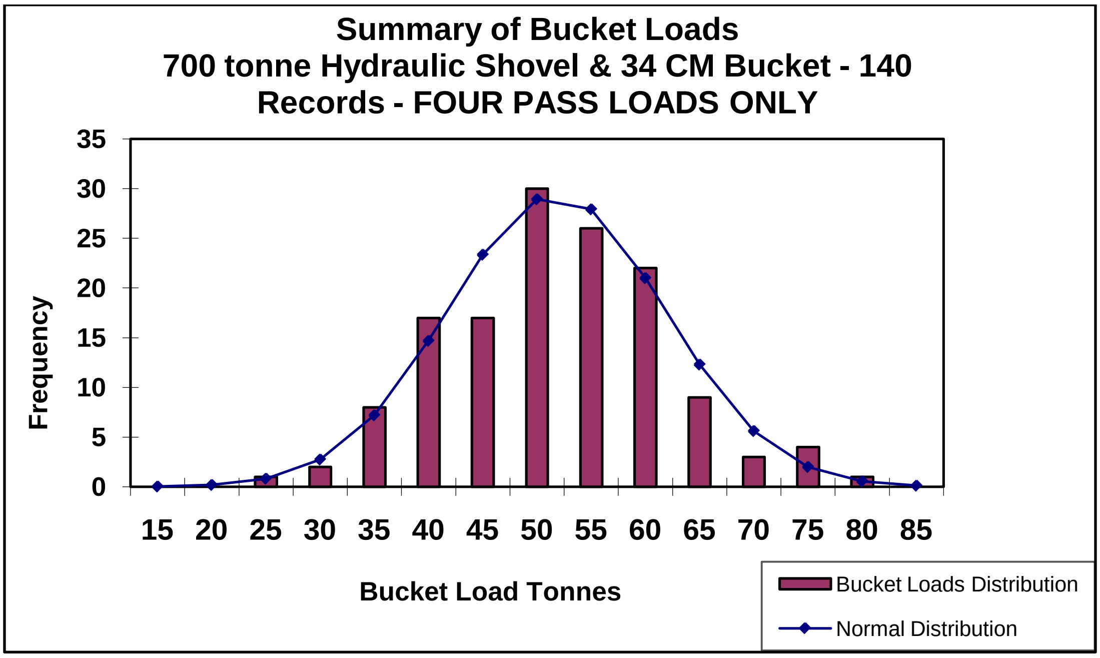


Figure 3.8 From Table 3.16

Table 3.17 Selected 5 Pass Bucket Load Data
700 tonne Hydraulic Shovel Loading 220 tonne Trucks

	First Loads	Intermed. Loads	Last Loads	Except First Loads	Except Last Loads	All Bucket Loads	Truck Payloads	Average Bucket Loads
Number of Records	23	84	31	115	107	138	31	31
Maximum Value	78.6	72.2	53.6	72.2	78.6	78.6	262.9	52.6
Minimum Value	35.0	30.6	14.1	14.1	30.6	14.1	197.0	39.4
Range	43.6	41.6	39.5	58.1	48.0	64.5	65.9	13.2
Average of Range	56.8	51.4	33.9	43.2	54.6	46.4	230.0	46.0
Arithmetic Mean	63.15	48.03	31.40	43.54	51.28	46.81	230.50	46.10
Median	64.3	47.4	31.6	44.7	50.0	46.6	232.0	46.4
Std Dev Sample	10.66	9.25	8.99	11.77	11.38	13.69	15.99	3.20
Coefficient of Variation	0.169	0.193	0.286	0.270	0.222	0.292	0.069	0.069
Skewness	-0.67 To Left	0.35 To Right	0.41 To Right	-0.06 To Left	0.40 To Right	0.11 To Right	-0.14 To Left	-0.14 To Left
Kurtosis	0.73 Leptokurtic	0.11 Leptokurtic	0.12 Leptokurtic	-0.15 Platy-kurtic	-0.30 Platykurtic	-0.30 Platykurtic	-0.22 Platykurtic	-0.22 Platykurtic

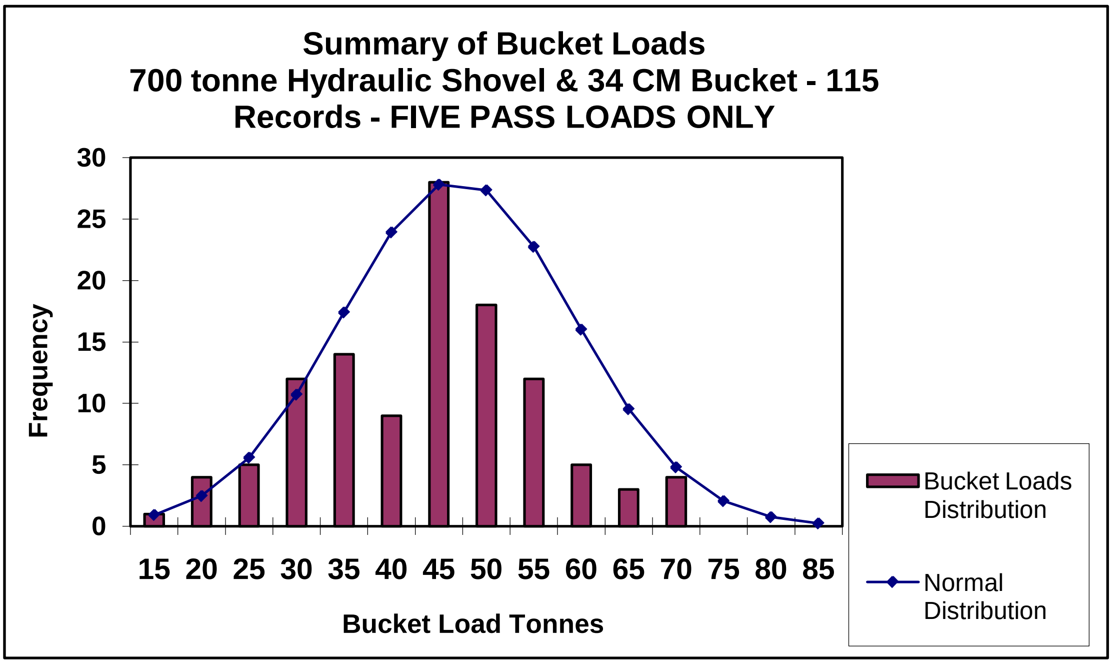


Figure 3.9 From Table 3.17

Distributions of bucket loads and truck payloads were examined qualitatively by constructing histograms with superimposed model distribution curves as shown by Figures 3.2 through 3.9 excepting Figure 3.5 for number of bucket passes. The distribution model for bucket passes was not an important issue for the research. The essential interest in bucket passes was the relationship with payload dispersion. These histograms were developed within Microsoft's Excel application using value intervals selected by methods proposed by Devore (Devore, 1999), rounded up to logical intervals and adjusted by trial and error. Normal distributions were generally exhibited for bucket load data. Consequently truck payloads, as small samples of bucket loads, and as expected, were also normally distributed.

To test the form of data distributions throughout the research, Kolmogorov-Smirnov (K-S) non-parametric tests were applied within the SPSS application (access provided by Curtin University of Technology). K-S tests were run for the above listed figures; also histograms were generated together with probability P-P and quantile Q-Q plots (terms simplified to Pplots and Qplots herein).

Results of the K-S tests with interpretive comments are provided in "Distribution Testing ", Volume 2, Appendices. Each of the K-S tests and accompanying plots are generally assigned the figure number to which the specific test relates. Table KS 6.1 in Distribution Testing appended summarises test results.

In each case, for Figures 3.2 to 3.4 and Figures 3.6 to 3.9 confirmed normal distributions and the Pplots and Qplots verified correlation. The following were noted:

- Pplot and Qplot correlations were positive for bucket load data samples and, as expected, increasingly positive for truck payloads.
- Verification of normality seemed to be enhanced by the number of data observations in the sample – see Figure 3.7 for 272 observations compared with other figures.

This last point is consistent with Harr's concept of "settling down or statistical regularity" – improvement (of fit to model or test distributions) with increase in sample size (Harr, 1977).

The form of distribution of bucket loads and truck payloads is of value in that a normal probability model can be used to predict the behaviour of all bucket load/truck payload activities. Bucket load and truck payload data and related descriptive statistics identify reliable, comparative measures of the variability of truck loading that affects productivity and cost of load and haul operations in open pit mines.

In summary:

- Bucket load data is normally distributed.
- Accordingly, truck payloads that are small samples of bucket loads are also normally distributed.
- Truck payloads have reduced dispersion compared with bucket loads that is an inverse 0.5 power function of the number of passes per truck payload.

These mathematical outcomes that are supported by empirical data and their potential as operating criteria and tools for selection of equipment and control of operations, is a significant focus of the research.

More detailed treatment of the statistical theory and relationships applied during the research is provided in "Mathematical Principles – Notes" appended in Volume 2.

Relevant descriptive statistics have been extracted from four data sets, summarized in Table 3.18 and illustrated by Figure 3.10.

Table 3.18 - Summary Of Bucket Load Data Recorded February & April 2004

From Table	3.16	3.15	3.17	3.14
Load. Equip. Tonnes & Average Passes	**700/4**	**700/4.44**	**700/5**	**190/8.88**
Mean of All Bucket Loads Recorded	55.12	50.52	46.81	23.98
Mean of Intermediate Bucket Loads	55.54	50.40	48.03	23.41
Mean of Truck Payloads	222.34	225.20	230.50	213.08
CV of All Bucket Loads Recorded	0.214	0.275	0.292	0.269
CV of Intermediate Bucket Loads	0.164	0.223	0.193	0.187
CV of Truck Payloads	0.083	0.079	0.069	0.063
Theoretical Projected CV Based on 4-Pass Truck Payloads as Standard	0.083	0.078	0.074	0.055

Observations from Table 3.18 include:

- CV of bucket loads recorded for the selected sets of data are approximately of the same order but are comparatively variable without any trend correlation across the data sets for increasing number of passes per truck payload.
- CV of intermediate bucket loads exhibit similar characteristics.

- For the sets of data selected with a constant number of passes, as expected, the CV's of average bucket loads and truck payloads are identical (Tables 3.16 and 3.17).

- There is a subtle but significant, consistent reduction in CV as number of passes per truck payload increase – as illustrated by Figure 3.10.

The manifest influence on truck-payload dispersion of number of bucket passes is essential evidence supporting realisation of a fundamental objective of the research – providing means for truck payload control and management in the quest for safe best practice as well as optimum load and haul productivity and lowest possible cost.

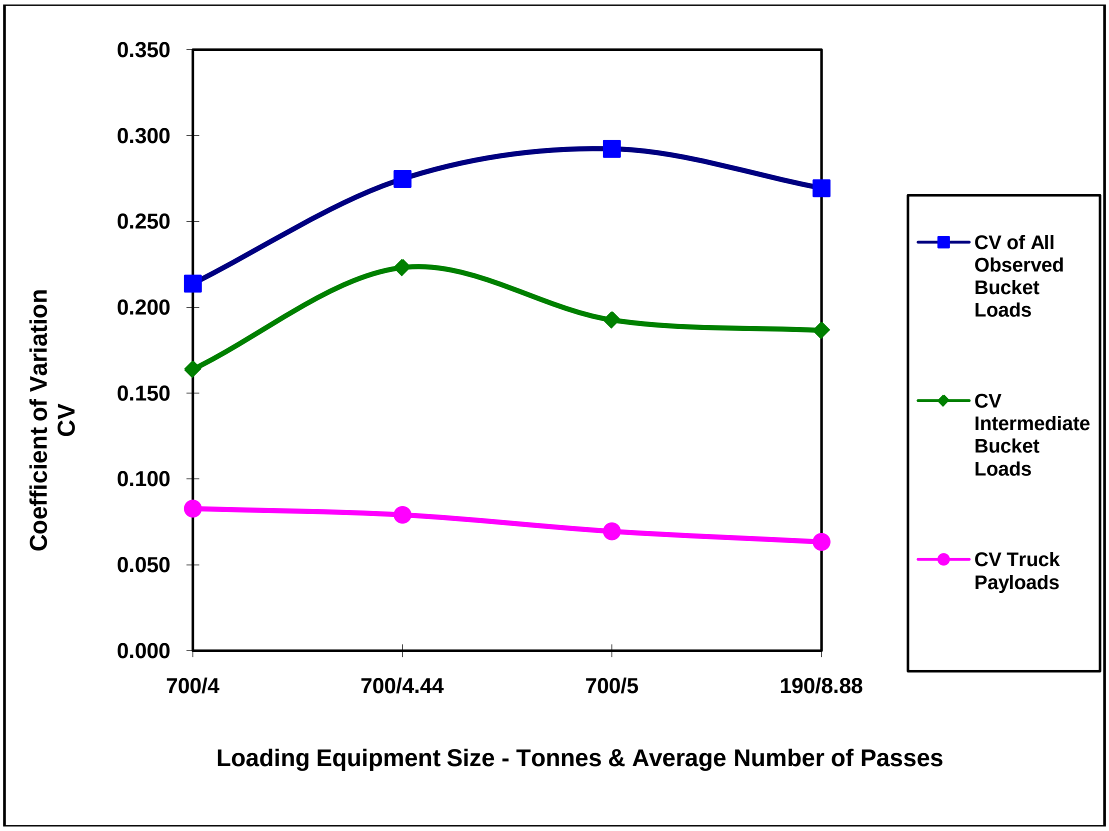


Figure 3.10 Summary – Coefficients of Variation for Selected Bucket Load and Payload Data Sets From Table 3.18

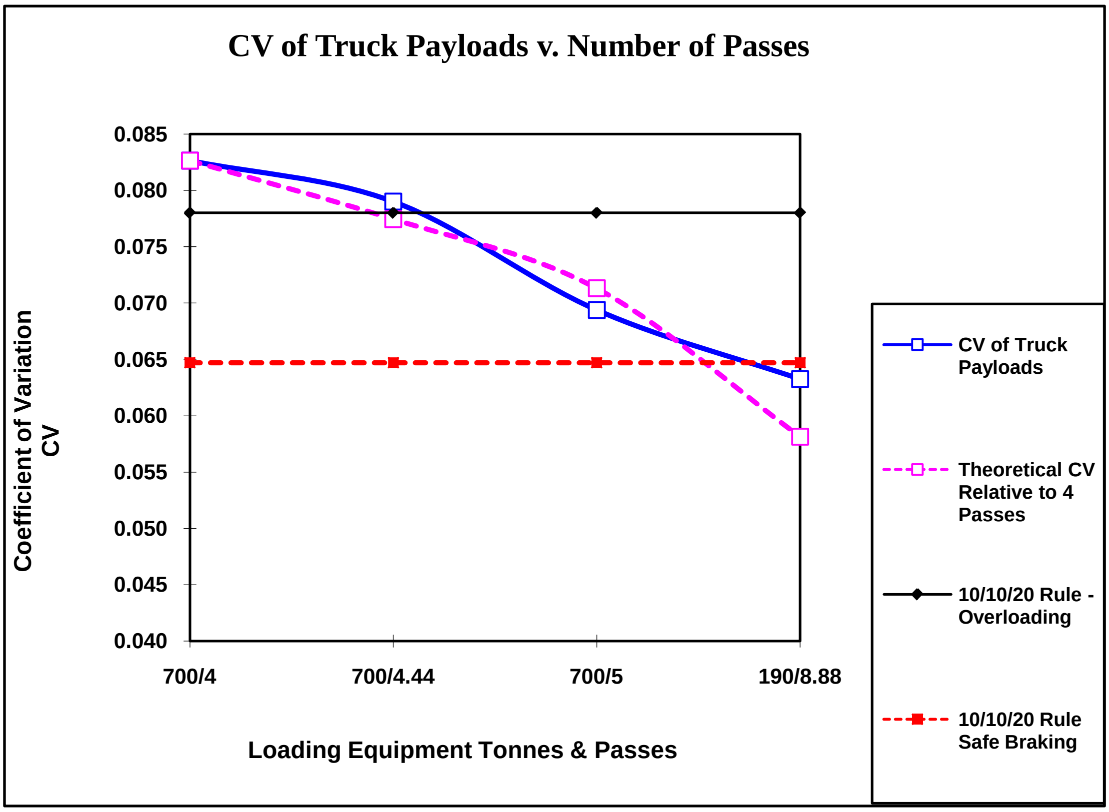


Figure 3.11 From Table 3.18

In Figure 3.11, the CV for the selected data sets are compared with:

- The theoretical reducing CV based on adopting the 4-pass CV as a standard and invoking sampling theory that should result from the increasing number of loading passes for the data sets analysed.
- Caterpillar's 10/10/20 Policy (Overloading) where not more than 10% of truckloads may exceed 110% of target payload (compliance will keep GMW within acceptable maximum limits).
- Caterpillar's 10/10/20 Policy (Safe Steering and Braking) where no truck payload should exceed 120% of the target payload to ensure safe steering and braking at the commensurate GMW. The confidence level adopted for this rule application is 1:1,000 (approx. +/- 3σ).

Caterpillar's 10/10/20 Policy states "The mean of the payload distribution shall not exceed the target payload and no more than 10% of payloads may exceed 1.1 times the target payload and no single payload shall ever exceed 1.2 times the target payload." The meaning and derivation of "target payload" and operating implications are dealt with in some detail in Sections 3.3.5 and 3.3.9.

The implications and conclusions of the above comparisons are also expanded in some detail in Sections 3.3.5, 3.3.6, 3.3.7 and 3.3.9.

Comments

Descriptive statistics for four-pass loading in Table 3.16 reveal:

- Relatively even mean values for subsets of first, intermediate, last and all bucket loads that are also reflected by the CV in each case.
- Compared with CV's of other data sets the four-pass CV's are generally lower as illustrated by Figure 3.10.
- As illustrated by Figure 3.11, notwithstanding the lower bucket-load CV's the CV for four-pass truck payloads is the highest value at 0.083 compared with data sets for higher numbers of passes.

Review of Table 3.17 for five pass loading indicates:

- A relatively high mean truck payload of 230.50 tonnes.
- Relatively low mean value for last bucket loads.
- A consequently higher than expected CV for the data set of all bucket loads – as illustrated by Figure 3.10.
- Notwithstanding the high CV for all bucket loads the five-pass truck payload CV reduced to 0.069 – a natural consequence of the increased number of bucket loads compared with lower-pass data sets. This natural phenomenon can be explained by statistical mathematics, including sample theory summarized from a number of references in MPN 3, Mathematical Principles – Notes appended in Volume 2.

Review of Table 3.14 for the 190 tonne wheel loader, averaging 8.88 passes per truck payload, shows:

- A relatively high CV for all bucket loads – illustrated by Figure 3.10
- A low CV for truck payloads of 0.063 – also shown by Figures 3.10 and 3.11.

- But, as illustrated by Figure 3.11, the reasonable correlation, based on sample theory, between actual recorded data and theoretical projection of CV for truck payloads from data sets recorded from the 700 tonne hydraulic shovel – – is not sustained for the 190 tonne loader statistics.

In this last case it is not clear why the correlation between actual and theoretical CV's across the range of data sets does not persist through to an average of 8.88 passes per truck payload by the 190 tonne wheel loader. Review of descriptive statistics in Table 3.14 for 190 tonne wheel loader data confirms that the mathematical relationship between standard deviations and CV for bucket loads and for truck payloads exhibits reasonable correlation.

Features of the statistics include:

- The mean of last bucket loads was relatively low.
- The outlying low values substantially reduced the truck payload mean but not significantly affecting the bucket load dispersion – resulting in elevating the CV for all bucket loads to 0.269 as shown in Table 3.14.
- In turn CV of truck payloads might be expected to be elevated. But, as noted in the comments on 4-pass loading, and as illustrated by Figure 3.10, reduced bucket load CV does not appear to significantly influence CV of truck payloads; so, conversely, high CV of bucket loads in this case do not necessarily significantly influence CV of truck payloads averaging 8.88 passes.
- The small sub-sample of data records, 63 bucket loads, 71 passes; and, likely most significantly, 8 truck payloads in this data set may be insufficient to provided sufficiently accurate and reliable descriptive statistics.
- Although actual to theoretical CV correlation was not sustained, the downward trend with increasing number of loading passes continued – as confirmed by Table 3.18 and illustrated by Figure 3.11.

Comparison of statistical results with the 10/10/20 Policy, promoted by Caterpillar Inc through Caterpillar Global Mining, Peoria, Illinois, USA, in Figure 3.11 indicate that:

- Compliance with the “10/10” part of the Policy, applying to truck overloading, is facilitated as CV of truck payloads decreases – an inter-active result of specifically, the increasing number of truck-loading passes; also, ensuring consistent digability (voids ratio, fragmentation and size distribution) of material being loaded.
- Compliance with the more stringent “20” part of the Policy (as shown by Figure 3.11) relating to safe steering and braking is also facilitated, albeit at lower CV, for similar reasons.
- Compliance with the “20” part of the Policy – no truck to exceed 120% of the target payload – appears to automatically ensure compliance with the “10/10” part of the Policy for truck overloading.

The issues generally identified in the above comments are revisited in more detail in Section 3.3.6.

3.2.9 Bucket Cycle Time

Preliminaries

Bucket-cycle and truck-loading times substantially determine productivity of all loading equipment. Bucket cycle time data are:

- In practice generally rounded up or down to unit-second intervals for recording purposes.
- Selected in small samples accumulating to truck loading time.

Condition of the material to be loaded, physical operating conditions and efficiency of loading equipment operators substantially influence bucket cycle times. Influence of operators and material condition tend to be random in effect. As small accumulations of continuously random bucket cycle time so truck loading times are continuously random with collected data similarly rounded up to unit-second intervals of time.

All of these variables are “continuous and random” since any of the outcomes within the total distribution of outcomes is possible to be observed individually or as a sample for any event.

Absolute productivity of cyclical truck-loading equipment is a function of the total number of bucket cycles in any chosen period of time. Also, absolute productivity of cyclical haulage equipment such as mining trucks is, to a significant degree, a function of the time taken to load each truck that, in turn, is a function of the bucket cycle time, the number of bucket passes per load and, as described in Section 3.3.9, to a degree, dependent on the payload per bucket cycle.

Consideration of the above, along with the influence of number of passes of bucket loads on the dispersion of truck payload distributions introduced in Section 3.2.8, initiated the thought that a similar concept applies to loading cycle times. That is:

- Bucket cycle times, as small samples, constitute truck-loading time.
- Sample Theory and the Central Limit Theorem (summarily described in Mathematical Principles – Notes, MPN 1 to MPN 3, appended), seen to apply to data from bucket loads and resulting truck payloads, likely are also

relevant to bucket cycle times and truck loading times, albeit to a differing degree.

- Increasing passes to load a truck likely tend to reduce the variability of the truck loading time (albeit longer by the additional passes) that will consequently provide more consistency and "rhythm" to hauling operations.
- The concept of “rhythm” as a desirable operational state developed in this section and Section 3.4, is an incentive to seek further evidence to support the hypothesis advanced in earlier sections that pursuit of the expected productivity and cost benefits of 4 or even 3 pass loading is not all positive due, in the case of cycle times, to increased bunching (queuing) inefficiency. Any expected benefits of short truck loading times will be, to some degree, offset by the increase in bunching inefficiency - the question is - how much? This question is addressed in Section 3.4.

Data and Filtering

Table 3.19 summarizes a sample of bucket-cycle-time data acquired from an open pit mining operation where 550 tonne hydraulic shovels with 34 cubic metre buckets load 220 tonne mining trucks. It was noted that the truck exchange time was included in the first bucket cycle time. This complication required that analyses consider two sub-samples of the data, i.e., first bucket cycles and bucket cycles exclusive of the first. Truck loading times calculated from the data necessarily included the first pass cycle time. The effect of the extended-time first cycles including truck exchange time as high-end outliers in the bucket cycle time distribution also required consideration.

Truck exchange time in the context of truck loading time is that part of the fixed “turn-and-spot” time for a truck nett of the continuously variable first bucket cycle time that occurs concurrently as the truck is maneuvering into position for loading. It was indicated that “turn-and-spot” time, also a continuous random variable, generally exceeds the time for the first bucket cycle. Truck exchange time is a continuous random variable, being the difference between the two continuous random variables of “turn-and-spot” time and nett cycle time for the first bucket load.

A hypothetical case is worth considering where “turn-and-spot” time is generally constant in operation; but variable only to the extent of the intrinsic performance of a mining truck in specific operating conditions. Any changes in the hypothetical constant (intrinsic) “turn-and-spot” time:

- Will adjust the truck exchange time by a like amount.
- In turn the mean loading time will be adjusted by a similar time interval.
- The variation and standard deviation of truck loading time distribution will not be affected as the constant adjustment does not affect the dispersion of the distribution.

So, if the truck-exchange time is reduced by a practically constant value the mean truck loading time will be so reduced but with no effect on the measures of dispersion of the truck loading time distribution. (Devore, 1999). Deterministic calculations of truck loading time and total haul cycle times; and subsequent deterministic productivity estimates rely on this mathematical proposition.

The relationship between loading equipment performance efficiency and number of passes per truck payload was introduced in Section 3.2.6 and demonstrated by Table 3.4. Particularly the significant affect of truck exchange time was demonstrated. In the context of mining truck performance, “turn-and-spot” time is the fixed time dedicated to locating the truck ready for loading. In the context of loading equipment performance, this truck-cycle time increment, nett of the time taken to first bucket cycle time to the point of readiness to dump into the truck, is, the “truck exchange” time. The effect of this idle time is reflected in loading equipment productivity.

The raw bucket cycle time data was reviewed and three levels of filtering successively applied to deliver:

1. Filter Level 1 – nett of obviously non-comparable, erroneous or anomalous records.
2. Filter Level 2 – nett of the bulk of time anomalies due to causes non-intrinsic to the capability of the loading equipment, operator efficiency or loading equipment operating techniques – facilitated by deleting complete truck loading time records containing an anomalous bucket cycle time.
3. Filter Level 3 – where complete truck loading time records containing first cycle times greater than 60 seconds and/or subsequent cycle times greater than 45 seconds were deleted to provide “smoother” loading time data and an

improved understanding of the characteristics and descriptive statistics of the fundamental bucket-cycle-time data.

Filter Level 3 is an hypothetical treatment of the data and can be viewed as predictive of the benefits of elevating the loading operation to best possible practice by improved practices. The outcomes of descriptive statistical analysis at Filter level 3 are indicative of suitable targets for continuous improvement in the operational case from which the data was sourced.

The general outcomes of this staged filtering are summarized in Table 3.19.

Table 3.19 – Summary of Data Residuals at Each Filter Stage

Filter Stage	Truck Loading Time Records	Bucket Loading Time Records	Average Bucket Loads/Truck Load
Raw Data	436	2,508	5.77
Level 1	428	2,444	5.71
Level 2	368	2,093	5.69
Level 3	188	1,070	5.69

The original data consisting 436 truck-loading times included.

- Truck loading times requiring more than 7 (8 up to 12) passes – known to be from a smaller hydraulic excavator wall scaling and cleaning up – so not necessarily comparable with the bulk of the data.
- Records where operations were interrupted by management-caused stoppages, also mechanical breakdowns, i.e., events for which alternative, appropriate-level discounting allowances are made or factors applied to performances.

Filter Level 1 eliminated these anomalies.

Filter Level 2 was applied on the basis of analysis of comments recorded against each truck loading activity. The filtering criteria and indicated relative frequency for the following major lost time occurrences included:

- Clean up by shovel or waiting on support equipment to effect house keeping at the working face – 32%.
- Waiting on trucks – 30%.
- Re-locating from the working area to a new digging location on the same face or relocating to an entirely new face – 28%.
- Some 4% of the time-loss incidents were for shovel manoeuvering - but not leaving the working face.
- A further 4% of time loss occurrences were caused by trucks bogging in the bench with the shovel required to manoeuvre behind the truck and lift/push the truck out of the boggy area.
- The balance of lost-time incidents included isolated occurrences of face scaling to avoid truck damage from collapses, dealing with face collapses, large particles in the face, bridging in the bucket and water truck intrusions.

The above-listed time-loss incidents are due to causes independent of the capability of loading equipment, the operator or method of operation. Accordingly, it is logical to separate such occurrences from cycle time estimation or selection, to assess and allow for the impact on load and haul productivity of such time losses independently. There are three significant reasons for separate treatment of such time losses:

1. For operational monitoring and control, transparency of time contingency or separate discount factoring provides understanding of the nature and magnitude of allowances.
2. Such extraordinary time losses are an indication of the degree of inefficacy of pit development plans and operating deployment strategies, adverse influence from interrelated activities such as drilling and blasting and the physical and geotechnical characteristics of each individual ore deposit and its host stratigraphy.
3. Any allowances for these time losses are, of necessity based more on experience and operational expectation, i.e., “crystal-ball gazing” and unrelated to intrinsic capability of the loading equipment, the operators and operating techniques.

Some causes of minor time losses including operator changeovers and minor stoppages for mechanical inspection are closely interrelated with performance of the equipment and/or operator inefficiency/maintenance support and rightly should be included in time categories such as equipment management efficiency and maintenance downtime. For the research analysis these minor time losses were considered inconsequently small and were ignored.

The data was collected from a base metal open-pit mine where ultramafic rock types are chemically altered to chloritic schists and talcose horizons in the weathered zone. There is a relatively deep weathering profile with:

- Shallow tertiary-lateritic capping over an incompetent highly weathered zone that, in some areas is free-digging massive leached-out clays; over
- A transition zone containing some massive clay inliers with a mixture of highly-weathered to fresh rock risers and floaters including some chemically-altered ore lenses; where, generally, drilling and blasting quality and consistency is difficult to achieve; over
- A fresh ultramafic rock zone where the bulk of ore suitable for froth-flotation concentration is mined; also where drilling and blasting quality is more consistent.

At the time of data recording, load and haul operations were in the transition zone where occasionally areas of incompetent clays are exposed on benches – so some bogging of trucks was experienced.

Data records that were obviously affected by operating conditions characteristic of the transition zone were filtered out on the basis that separate allowances would be made for resulting time losses.

It is not represented that the extraordinary time losses listed; and accepted as a basis for filtering the data from Level 1 to Level 2 to minimize their effect; are in any way typical of open pit mining operations. There would certainly be some similar time-loss occurrences in other open pit mining operations. But it is generally accepted that in considering individual mining properties there are “No Two The Same”. This was an observation of A. A. C. (Bert) Mason adopted as the title of his autobiographical history of forty years of evaluating and developing Australian mining properties (Mason, 1994).

But it is represented, and analysis shows, that the data collected provides an insight into the stochastic relationship between bucket cycle times, truck loading times and the effect of number of passes on variability of truck loading time.

Filtering to Level 3 was based on descriptive statistics resulting from Level 2 as follows:

- For bucket cycles other than first cycles, by assigning a maximum value of 45 seconds that, on the basis of experience, is consistent with a mean of 28 to 30 seconds; and
- Assigning a maximum of 60 seconds to first bucket-loads to allow for a reasonable, inclusive truck-exchange time of 15 seconds included in truck loading time.

Comments on criteria for Filter Level 3:

1. First passes – observed minimum of 33 seconds – expected reasonable mean value assumed at 45 seconds – so set maximum at 60 seconds for an approximate 27 second range.
2. Passes after the first – initial minimum observed at 9 seconds so filtered out at Filter Level 2 to a minimum of 15 seconds – assumed an expected reasonable mean time for passes nett of truck exchange time of approximately 30 seconds – so set maximum at 45 seconds for an approximate 30 second range.

That the distribution for all bucket passes exhibits a negative (to left) skew, Table 3.25 (left hand column, albeit relatively small skewness) indicates that the centralizing effect of Filter Level 3, although only a hypothetical case, is slightly over-enthusiastic. Table 3.27 and Figures 3.22 and 3.23 illustrate that the mode, median and mean are practically coincident. (Tables 3.25 and 3.27, Figures 3.22 and 3.23 are included in this section)

The assumptions for filtering and the outcomes are further revisited in more detail in the interpretation of descriptive statistics.

Statistical Analysis and Results

The overall objective of the analysis was to:

- Derive an understanding of the nature of the underlying distribution of bucket load cycle times and truck loading times at selected filter levels.
- To determine a measure of the variability of bucket load cycle times and related truck loading times both collectively and in separate data sub-groups for selected number of passes at various filter levels – as input to examination of shovel and truck fleet matching and bunching and queuing in general described later in this thesis.
- To provide a basis for assessing potential improvement benefits to loading productivity from reduced variability in bucket cycle times and truck loading times.

Data derived at each level of filtering was analysed to provide descriptive statistics to realise the objectives.

In the process of selecting Filter Level 1, bucket-pass data was examined for a preliminary understanding of the nature of the loading operations. After filtering out data from clean up operations using smaller loader equipment, raw bucket-pass data was analysed with results shown in Table 3.20 and illustrated by a histogram and plot of cumulative frequency % in Figure 3.12.

Table 3.20 – Raw Data Bucket-Pass Analysis

Number of Passes	3	4	5	6	7	8	9	Totals
Frequency	1	19	146	206	51	3	2	428
%	0.23	4.44	34.11	48.13	11.92	0.70	0.47	100.00
Cumulative %	0.23	4.67	38.79	86.92	98.83	99.53	100.00	

It was noted that, for this operation, after ignoring the small number of loads from passes exceeding 7 (1.2%) that only 12% of the truckloads required 7 passes.

The small number of 7-pass loads raises the question, for this essentially 5 or 6-pass application (as operating when the data was observed), whether foregoing production of the 7th passes could be traded off against reduced truck loading times that benefit truck productivity and improve the “rhythm” of the operation. This issue is revisited in more detail in terms of productivity in Section 3.3.9 with cost considerations in Section 5.4.

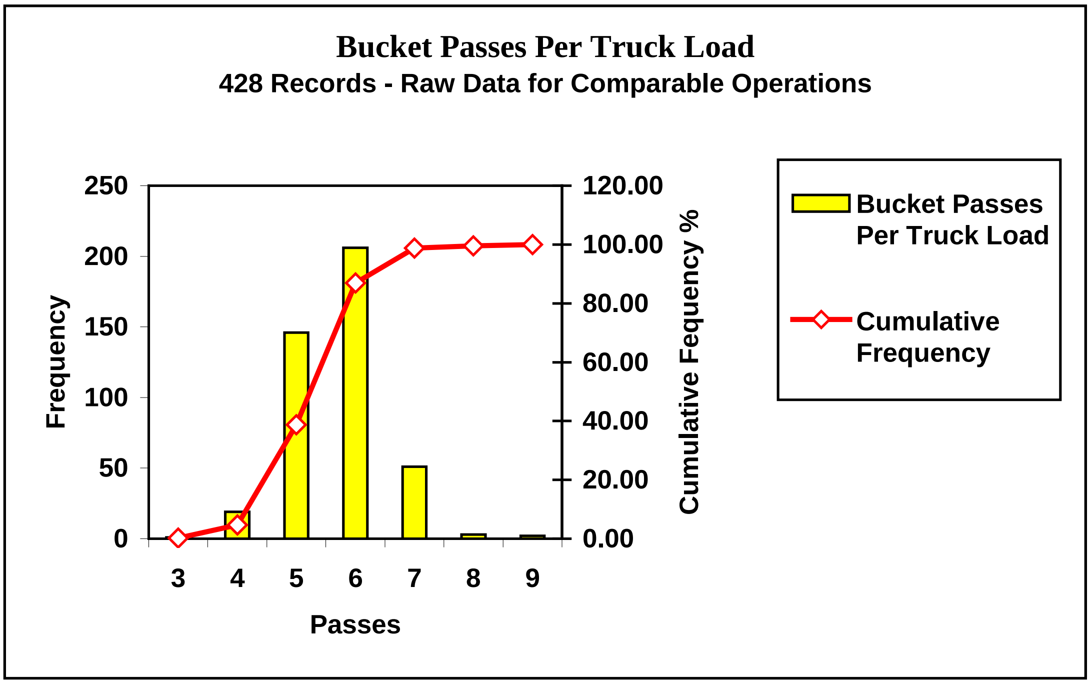


Figure 3.12 From Table 3.20

Service-time data is generally positively (right) skewed and normal distribution models are not applicable. Positive skewness can be understood intuitively as minimum values are practically limited in range but maximum values are less inhibited. For example, with a mean bucket cycle time of 30 seconds minimum values below 15 seconds would be impractical and anomalous. But maximum values are less inhibited, particularly as, in practice, time losses from causes both intrinsic and non-intrinsic to loading operations may add to cycle times. Distributions of continuous random service-time variables can typically be modeled by one of the family of gamma distributions. On the basis of interrelationships between descriptive statistics, particularly measures of central tendency, and qualitative evidence from the preliminary histograms it was decided that bucket cycle data was not normally distributed. Kolmogorov Smirnov (K-S) testing of the raw bucket-pass data generally confirmed this decision.

Kolmogorov – Smirnov (K – S) tests, Pplots and Qplots in modelling of bucket cycle times and truck loading times for a selection of cases across the range of filtering are summarized in Table KS 6.1.

As discussed, bucket cycle times; and truck loading times, as accumulated bucket cycle times, have intrinsic and non-intrinsic time components. Intrinsic bucket cycle and truck loading times appear to have central tendency and likely can be modelled by a normal distribution, albeit positively skewed. Normal probabilities could likely be applied to intrinsic times. But non-intrinsic time in bucket cycles and accumulated in truck loading times result in positively skewed empirical distributions that can likely modelled by a gamma distribution including the less positively skewed Weibull distribution.

In the process of productivity planning and equipment selection it may be a more transparent practice to develop bucket cycle times and accumulated truck loading times in terms of intrinsic time and deal with the non-intrinsic bucket cycle and truck loading time separately as discussed.

Data from Level 1 and Level 2 Filter sub-samples of data were analysed for each successive bucket cycle in sequence. The resulting mean values are summarized in Table 3.21 and illustrated by Figure 3.13.

Table 3.21 - Mean Cycle Times for Each Pass

Passes	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**Total Mean**	**Mean Except First Pass**
Level 1 Filtering	80.52	33.37	33.84	32.27	31.70	30.25	29.38	**40.80**	**32.39**
Level 2 Filtering:									
5.00	62.75	33.77	33.53	33.51	32.02			**39.12**	**33.21**
6.00	62.36	32.59	32.49	31.76	32.13	30.62		**36.99**	**31.92**
7.00	64.44	29.88	31.14	30.09	29.14	29.02	28.98	**34.67**	**29.71**
Interpolated Allowance for Changeover									
5.00	34.00	33.77							
6.00	32.70	32.59							
7.00	31.30	29.88							

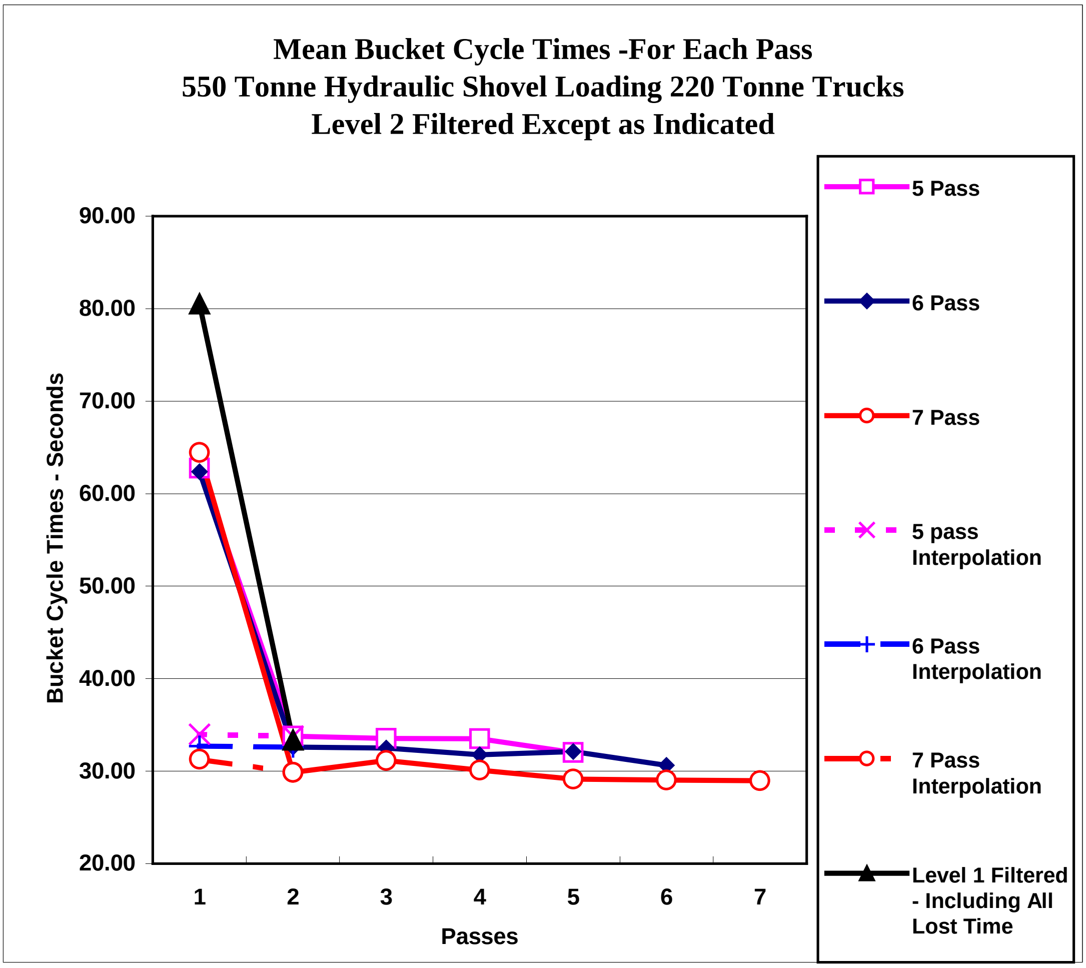


Figure 3.13 From Table 3.21

As first bucket cycles are inclusive of the truck exchange time, realistic nett first bucket cycle times were extrapolated on the basis of trend and arithmetic means for bucket cycles subsequent to the first. Including truck exchange times in the first bucket cycle time was, in this case a fortuitous practice. It provided a basis to estimate, by difference, the magnitude of truck exchange time, a time event most relevant to loading equipment productivity (Section 3.2.6 and Table 3.4); but, comparatively, not so important for truck productivity. An impact of 10 seconds of extra truck exchange time is 10 in some 200 seconds for loading equipment compared with 10 : 300 for 5-minute total truck haul cycles extending to 10 : 1800 for 30-minute total truck cycle time. Generally, when recording data to investigate truck productivity, the complete turn-and-spot time until the shovel dumps the first load is considered a fixed time to be included transparently in truck-cycle estimates. This is revisited in Sections 3.3 and 3.4.

From Figure 3.13:

1. Mean cycle time for each successive pass - for Pass 2 and above - shows a subtle downward trend.
2. First passes from Filter Level 1, before deletion of time losses non-intrinsic to equipment and operator performance, have been included in Figure 3.13 and indicate a mean of 80.5 seconds compared with some 62 to 64 seconds at Filter level 2 – so providing an indicated average of some 18 seconds (for this operation at the time data was recorded) due to those events that justified exclusion by filtering to Filter Level 2; also
3. First passes from Filter Level 2 show means of 62.7 to 64.4 seconds compared to extrapolated first cycle times estimated from means and trend of subsequent passes of 31.3 to 34 seconds, so indicating truck exchange times of 28.7 to 33.1 seconds in addition to the estimated first bucket cycle time. In terms of industry standards the indicated truck exchange time range is excessive – this issue is discussed in more detail below.

Interpretations of the observations include:

1. The subtle reducing trend of bucket cycle times is interpreted as evidence of the effect of the face-working technique where, for single side loading, the load from the furthest swing position is taken first to take advantage of the truck exchange time; and each successive bucket load is taken working towards the truck. For double-side loading the technique is often reversed. In this case, as the shovel has to swing through 180 degrees, the first bucket load is effectively 90 degrees swing wherever it is taken. To maintain an even-reach digging face, operators double-side loading generally take the first bucket load from the centre or from a proud section of the face or to facilitate bringing down an incipient overhang. The second bucket is generally from the toe of the face closest to the truck working away from the truck for successive bucket loads. These techniques are generally considered to deliver the best truck loading productivity. In practice variable digability can influence operators to focus on easier bucket-filling zones in the face. Best-practice operators avoid this temptation using the easier digging sections as later or last bucket loads to realise a truck load within the ideal designed number of passes. Collapse of overhangs is an all-too-often occurrence that is

a potential hazard for equipment and operators. Best-practice operators deal with overhangs at the incipient stage; and resist the temptation to burrow into the face where easy digging is presented. Best-practice operating is all about rhythm. An operator failing to persevere with a face-working plan best suited to the operation is counter-rhythmic. Technique-induced inefficiencies need to be investigated to determine if the cause is operator inattention, where the remedy is obvious; or there are underlying operating and material-preparation conditions that are frustrating the best intentions of the operators.

2. Time loss of 18 seconds for events non-intrinsic to the equipment and the operator – identified herein as "pit housekeeping" - represent an 8% time impost in mean truck loading time at Filter Level 1. As indicated, the data was recorded when operating conditions were relatively difficult. Physical working space is known to have been limited at the time. Material preparation by way of drilling and blasting was not realizing optimum results because of variable material competency across the stratigraphy being mined. Accordingly the indicated discount factor for the "pit housekeeping" of 92% is considered abnormally high. In practical loading operations there will always be some extraordinary time losses for the several reasons listed and other minor causes. It is considered that, for best practice, an achievable factor would be 98% with an absolutely, and unacceptable, low value of 90%. Traditionally estimations and operating budgets include this pit housekeeping inefficiency in a job management efficiency factor that includes non-utilization of available equipment. With the benefit of being able to identify and quantify such events, pit-housekeeping inefficiency should be separately identified and treated to facilitate management and control.

3. A mean truck exchange time of some 31 seconds included in the first bucket load data at Filter Level 2 is unacceptably high. As indicated, working space in the deep open pit where the data was recorded tends to be limited. Standardized safe and quick turn-and-spot techniques for exchanging trucks have to be modified because of limited span of working-face between pit walls. Turn-and-spot times were discussed in Section 3.2.6 and the effect on loading equipment efficiency illustrated by Table 3.4. For the 220 tonne mining trucks being discussed an average turn and spot time of 45 seconds is

reasonable. With an extrapolated average first bucket load of 31 to 34 seconds, theoretically the loader would be waiting 11 to 14 seconds, much less than the 29 to 33 seconds indicated by the sample of data investigated and analyzed. The additional approximate 18 seconds for first bucket load inclusive of truck exchange in the case being considered is equivalent to some 10 % reduced intrinsic efficiency in the loading equipment operation – as shown by Table 3.4. Assuming a 30 second cycle time, 6 passes and a 15 second exchange time an intrinsic efficiency of 92% is indicated by Table 3.4. If the exchange time is increased to 30 seconds the intrinsic efficiency discounts to 85%. This indicated additional productivity loss of some 7% is serious and likely unacceptable for open pit operations that are productivity focused to achieve mine development objectives. Direct-cost losses due to excessive truck exchange time of the order indicated are significant but not as serious because, during the extra manoeuvering time for each truck, costs of the loading equipment are limited to fixed costs plus operating and support labour, idling fuel or power supply and small commensurate maintenance costs plus a similar limited suite of costs for the manoeuvering truck.

Data from Filter Level 2 were separated into sub-samples of 5, 6 and 7 passes for each truck loading time with results summarized in Table 3.22, and illustrated by:

- For mean bucket cycle truck loading times by Figure 3.14; and
- For coefficients of variation (CV = σ/μ) by Figure 3.15.

The sub-samples from Filter Level 2 for 5,6 and 7 passes collectively consist 351 of the truck loading records. The remaining 17 consist 16 - 4 pass and 1 – 3 pass truck load samples considered to be anomalous pass-number data or too few in number for meaningful analysis and so have been ignored.

Table 3.22 Mean Cycle Times &Coefficients of Variation for Range of Loading Passes – Filter Level 2

Number of Passes	5	6	7
Mean Cycle Times - Seconds:			
First Pass	62.750	62.359	64.442
All Passes	39.116	36.992	34.671
Cycles After First Pass	33.208	31.918	29.709
Truck Loading Time	195.581	221.951	242.698
Coefficients of Variation:			
First Pass	0.389	0.390	0.448
All Passes	0.455	0.448	0.492
Cycles After First Pass	0.259	0.236	0.185
Truck Loading Time	0.182	0.180	0.153
Small Sample Theoretical	0.204	0.183	0.186

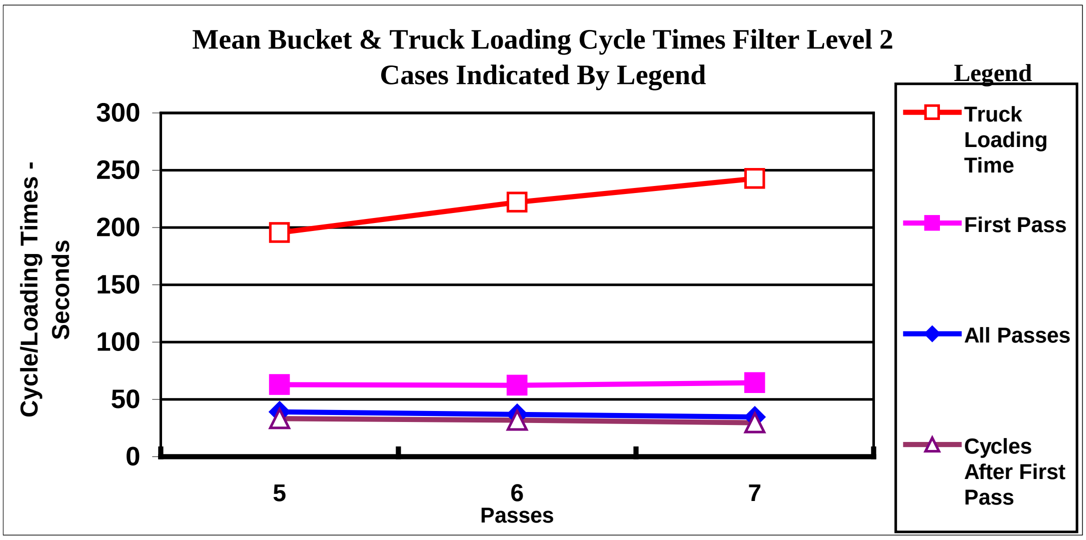


Figure 3.14 From Table 3.22

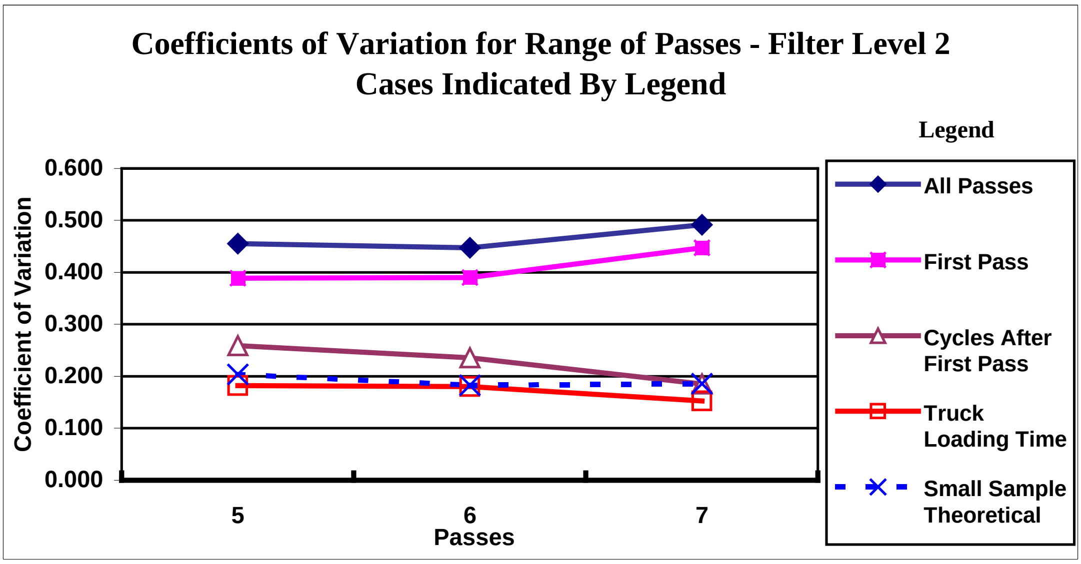


Figure 3.15 From Table 3.22

Figure 3.14 shows that mean bucket loads are consistent across the range of passes but with a subtle reduction in mean for successive passes consistent with Figure 3.13. As expected the mean truck loading times increase at a steady gradient consistent with increase of a pass in each case.

Figure 3.15 shows that the CV for all 7 Pass bucket data is higher than either 5 Pass or 6 Pass CV due to anomalously high outlying values in the first pass of the 7 Pass sub-sample. CV for data from cycles after the first pass show a tendency to reduce as passes increase as expected from Sample Theory and the Central Limit Theorem (Mathematical Principles – Notes, item MPN 3). CV for truck loading times for each of the three sub-samples show a tendency to reduce. Also included in Figure 3.15 is a Small Sample Theoretical CV for each number of passes calculated from the All Passes CV using sample theory as discussed in MPN 3, "Mathematical Principles – Notes, appended in Volume 2.

Correlation between the CV for truckloads and sample-theory projections from bucket loads is only moderate. A possible explanation is that the gradient of the truck-loading time CV is a combination of both the natural diminishing trend of each successive pass – Figure 3.13 - and centralizing effect of small samples of bucket cycles accumulating to truck cycle times. It is of practical importance that the clear reducing trend of CV for truck-loading time with increasing number of passes is sustained. This is revisited for Filter Level 3 statistics.

Further analysis of Filter Level 2 data is summarized in Table 3.23. Figure 3.16, illustrates the expected increasing trend of the measures of central tendency, average range, mode, median and mean values, for loading times over the range of passes. The relative values of these statistics are consistent with a positively skewed (to right) distribution.

Reducing the central-tendency statistics to indices enables comparison of the parameters underlying the distributions at each number of passes as illustrated by Figure 3.17. On an index basis these relationships are uniformly retained over the range of passes. This is interpreted as retention of the shape and scale parameters characteristic of the underlying distribution.

Table 3.23 - Average Range, Mode Median and Mean Values of Loading Time for a Range of Loading Passes

Number of Passes	5	6	7
Mean Etc. Cycle Times - Seconds:			
Range Average	213.500	257.000	278.000
Arithmetic Mean (Index Base)	195.581	221.951	242.698
Median	189.500	211.500	233.000
Mode	168	198	203
Mean Etc. Cycle Time Indices Mean = 1.00			
Range Average Index	1.092	1.158	1.145
Arithmetic Mean Index	1.000	1.000	1.000
Median Index	0.969	0.953	0.960
Mode Index	0.859	0.892	0.836

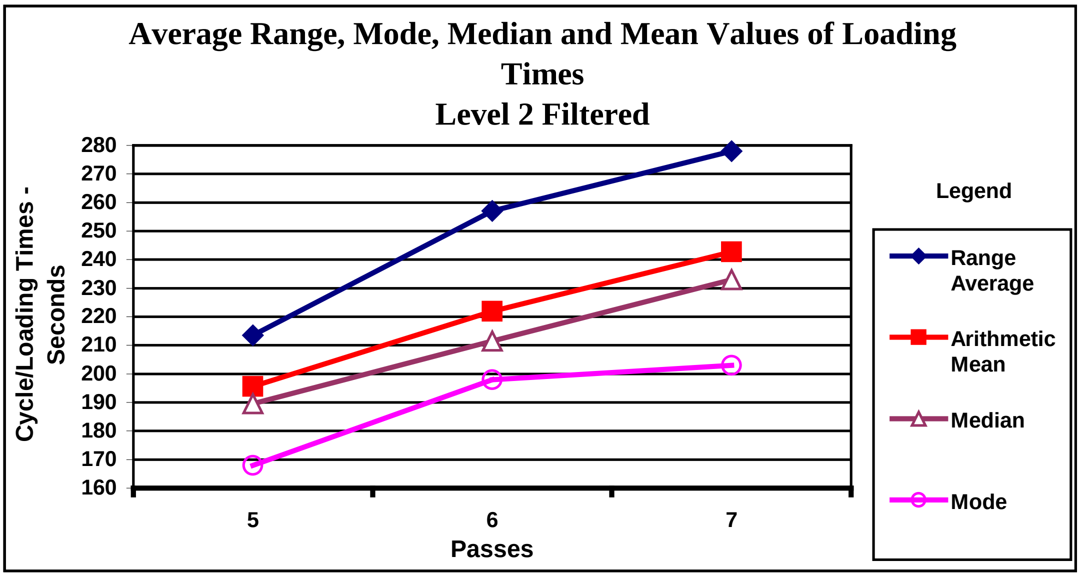


Figure 3.16 From Table 3.23

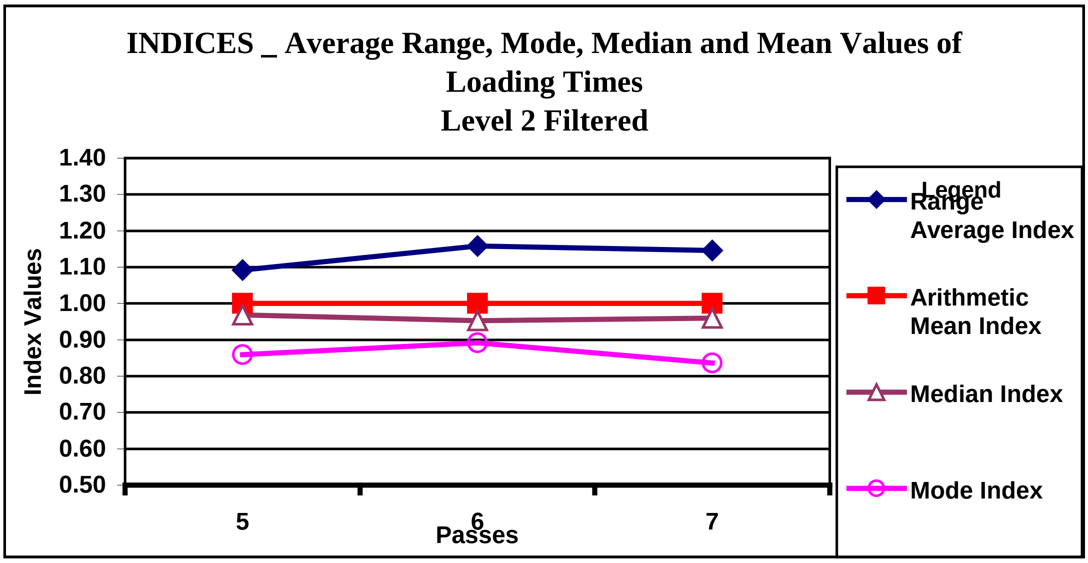


Figure 3.17 From Table 3.23

Table 3.24 summarises descriptive statistics at Filter Level 2.

Table 3.24 - Summary of Descriptive Statistics - Filter Level 2

	All Passes Loading Time	Number of Passes	5 Pass Loading Times	6 Pass Loading Times	7 Pass Loading Times	Cycles All Passes	All Cycles Less Pass 1	Cycles 5 Pass	Cycles 5 Pass Less pass 1	Cycles 6 Pass	Cycles 6 Pass Less pass 1	Cycles 7 Pass	Cycles 7 Pass Less pass 1
Number of Records	368	368	124	184	43	2093	1725	620	496	1104	920	301	258
Maximum Value	356	7	296	349	356	160	88	154	88	142	75	160	56
Minimum Value	131	4	131	165	200	15	15	16	16	15	15	18	18
Range	225	3	165	184	156	145	73	138	72	127	60	142	38
Average of Range	243.5	5.5	213.5	257	278	87.5	51.5	85	52	78.5	45	89	37
Mode	193	6	168	198	203	28	28	28	28	30	30	27	27
Mean	212.590	5.688	195.581	221.951	242.698	37.378	32.040	39.116	33.208	36.992	31.918	34.671	29.709
Median	205	6	189.5	211.5	233	30	30	31	31	30	30	29	29
Variance	1823.692	0.542	1270.506	1602.932	1371.073	290.065	60.189	317.444	74.181	274.184	56.665	290.615	30.293
Standard Deviation	42.705	0.736	35.644	40.037	37.028	17.031	7.758	17.817	8.613	16.559	7.528	17.047	5.504
Coefficient of Variation	0.201	0.129	0.182	0.180	0.153	0.456	0.242	0.455	0.259	0.448	0.236	0.492	0.185
Skewness	0.688	-0.131	0.681	1.062	1.017	1.902	1.867	2.149	2.296	1.544	1.463	1.422	1.520
	To Right	To Left	To Right	To Right	To Right	To Right	To Right	To Right	To Right	To Right	To Right	To Right	To Right
Pearsonian Skewness Coefficient	0.533	-1.273	0.512	0.783	0.786	0.947	0.789	1.367	0.769	1.267	0.765	0.822	0.387
Kurtosis	0.387	-0.248	-0.019	0.573	0.709	6.714	6.906	8.850	10.350	3.578	3.252	3.763	4.553
	Lepto-kurtic	Platy-kurtic	Platy-kurtic	Lepto-kurtic	Lepto-kurtic	Lepto-kurtic	Lepto-kurtic	Lepto-kurtic	Lepto-kurtic	Lepto-kurtic	Lepto-kurtic	Lepto-kurtic	Lepto-kurtic

K-S tests, appended in Distribution Testing in Volume 2, together with Pplots and Qplots confirmed that, notwithstanding any smoothing effect from filtering to either Level 1 or Level 2, neither bucket cycle time nor loading time data exhibit normal distribution characteristics for the total sample or sub-samples for 5, 6 or 7 passes.

Further to the K-S tests a histogram for frequency distribution for bucket cycle times nett of the first pass for Filter Level 2 was developed as illustrated by Figure 3.18. For comparison normal and gamma distributions have been overlaid. Qualitatively the bucket cycle time distribution more closely conforms to a gamma than a normal distribution. The positive skew of the frequency histogram is obvious. Figure 3.19, is a similar illustration of the loading time distribution for all truckloads at Filter Level 2 with the positive skew of the frequency data remaining manifest. But the centralizing tendency of truck loading times as small samples of bucket cycle times is also clearly indicated by comparing Figures 3.18 and 3.19.

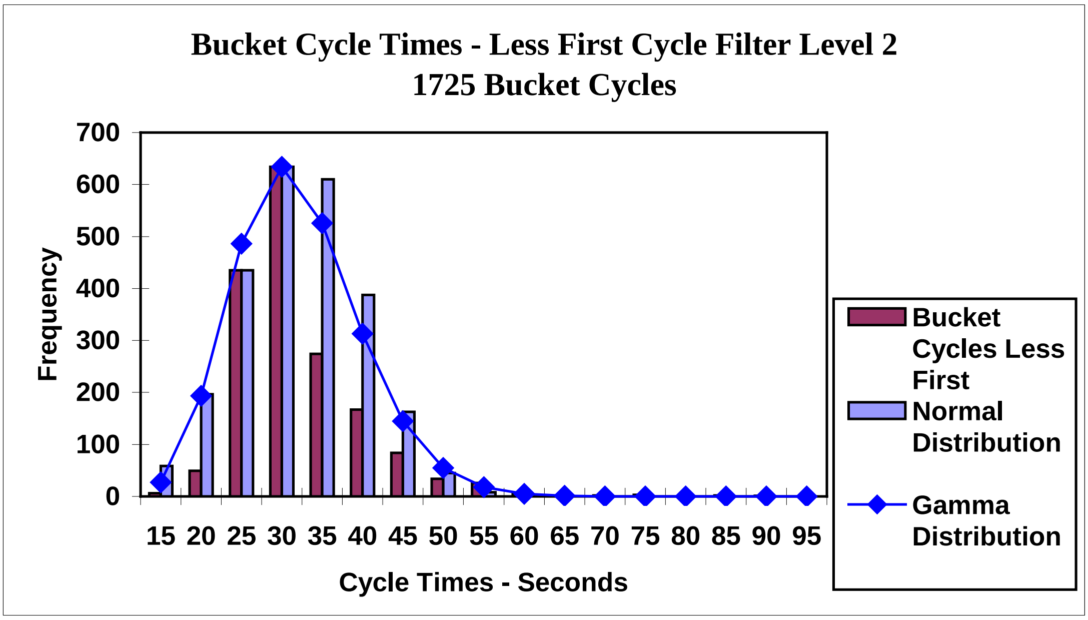


Figure 3.18 Table 3.24

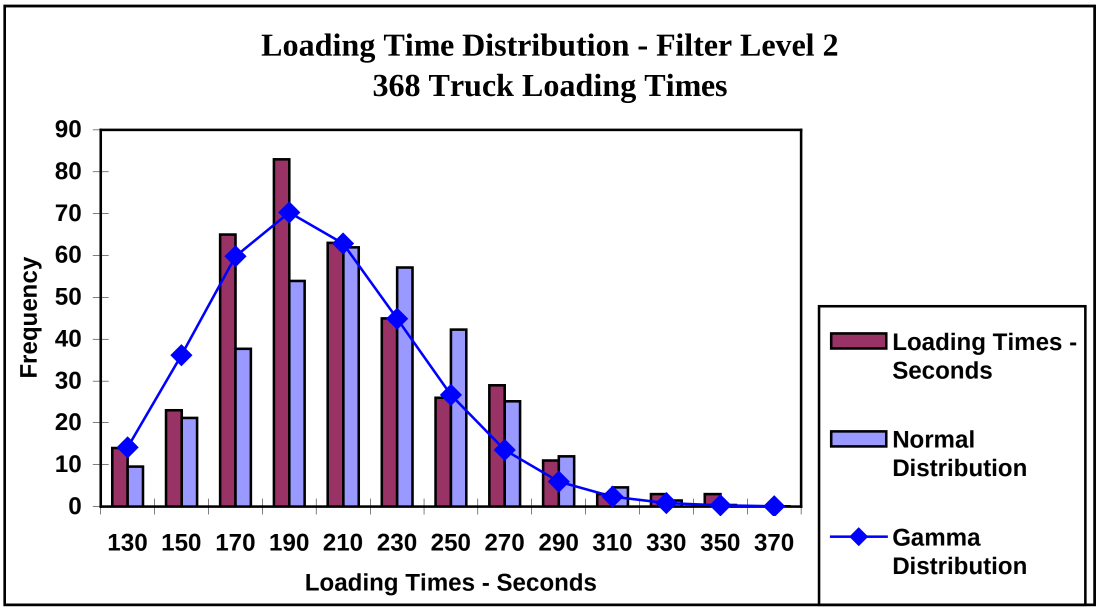


Figure 3.19 Table 3.24

Descriptive statistics at Filter Level 3 are summarized in Table 3.25.

Development of the equations and calculations of the equations and calculations of values for parameters **α** and **β** defining gamma distributions for the relevant histograms (Figures 3.18, 3.19, 3.24 and 3.25) is detailed in MPN 4, Mathematical Principles Notes. Particularly descriptive statistics adopted, mathematical process, calculated values for parameters and those values actually used are summarized in Table 3.32 appended in Volume 2.

Table 3.25 - Summary of Descriptive Statistics - Filter Level 3

	All Passes Loading Time	Number of Passes	5 Pass Loading Times	6 Pass Loading Times	7 Pass Loading Times	Cycles All Passes	All Cycles Less Pass 1	Cycles 5 Pass	Cycles 5 Pass Less pass 1	Cycles 6 Pass	Cycles 6 Pass Less pass 1	Cycles 7 Pass	Cycles 7 Pass Less pass 1
Number of Records	188	188	57	96	23	1070	882	285	228	576	480	161	138
Maximum Value	255	7	213	245	255	60	45	60	45	60	45	60	39
Minimum Value	131	4	131	165	200	18	18	21	21	18	18	18	18
Range	124	3	82	80	55	42	27	39	24	42	27	42	21
Average of Range	193	5.5	172	205	227.5	39	31.5	40.5	33	39	31.5	39	28.5
Mode	199	6	168	193	211	28	28	28	28	29	29	26	26
Mean	187.181	5.691	170.807	195.302	218.217	32.888	29.980	34.161	31.145	32.550	29.717	31.174	28.609
Median	190	6	168	193	214	30	29	33	30	30	29	29	28
Variance	660.25	0.59	386.91	280.55	212.36	66.96	23.56	71.98	31.55	65.35	21.44	56.16	13.61
Standard Deviation	25.70	0.77	19.67	16.75	14.57	8.18	4.85	8.48	5.62	8.08	4.63	7.49	3.69
Coefficient of Variation	0.137	0.135	0.115	0.086	0.067	0.249	0.162	0.248	0.180	0.248	0.156	0.240	0.129
Skewness	-0.03	-0.27	0.31	1.13	0.92	1.21	0.76	0.94	0.54	0.15	0.76	1.61	0.36
	To Left	To Left	To Right	To Right	To Right	To Right	To Right	To Right	To Right	To Right	To Right	To Right	To Right
Pearsonian Skewness Coefficient	-0.271	-1.206	0.428	0.502	0.868	1.059	0.605	0.411	0.611	0.946	0.464	0.870	0.495
Kurtosis	-0.16	-0.19	-0.65	1.05	0.28	1.07	0.48	0.49	-0.39	1.27	0.77	2.53	0.54
	Platy-kurtic	Platy-kurtic	Platy-kurtic	Lepto-kurtic	Lepto-kurtic	Lepto-kurtic	Lepto-kurtic	Lepto-kurtic	Platy-kurtic	Lepto-kurtic	Lepto-kurtic	Lepto-kurtic	Lepto-kurtic

A similar analytical process was followed for Filter Level 3 as described for Filter Level 2. Table 3.26, summarises mean values for sub-samples of 5,6 and 7 passes at Filter Level 3 with:

- Mean bucket cycle times and truck loading times – illustrated by Figure 3.20.
- CV for the first bucket cycle time, all bucket cycles, bucket cycles nett of the first cycle including truck exchange time and truck loading time across the range of sub-samples for 5,6 and 7 pass loads – including a theoretical small sample projection of truck loading time CV for truckloads based on all bucket cycle times after filtering – illustrated by Figure 3.21.

Table 3.26 - Mean Cycle Times & Coefficients of Variation for Range of Loading Passes - Filter Level 3

Number of Passes	5	6	7
Mean Cycle Times - Seconds:			
First Pass	46.23	46.72	46.57
All Passes	34.161	32.550	31.174
Cycles After First Pass	31.145	29.717	28.609
Truck Loading Time	170.807	195.302	218.217
Coefficients of Variation:			
First Passes	0.156	0.142	0.127
All Passes	0.248	0.248	0.240
Cycles After First Pass	0.180	0.156	0.129
Truck Loading Time	0.115	0.086	0.067
Small Sample Theoretical	0.111	0.101	0.091

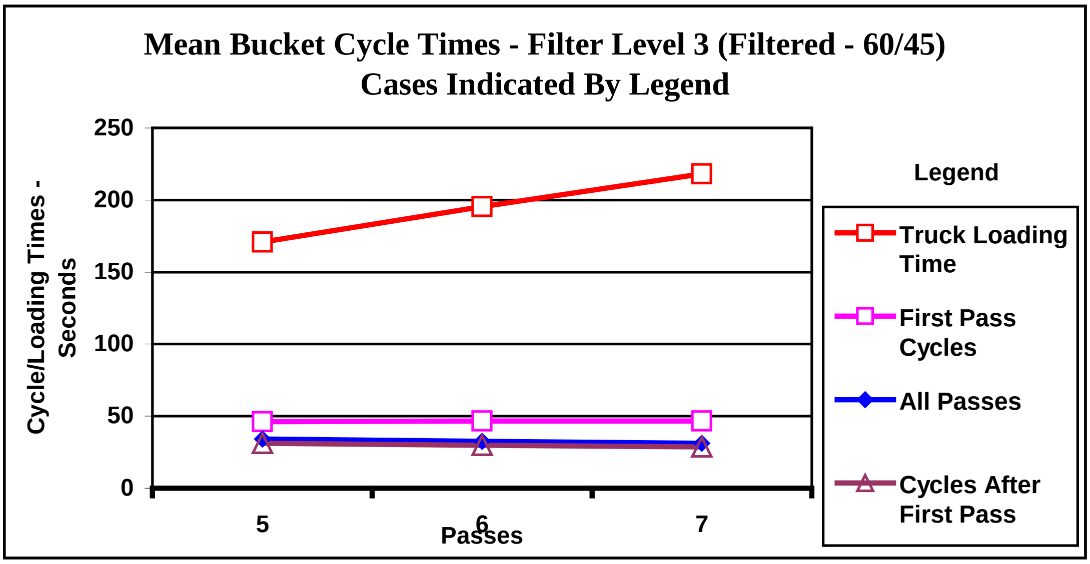


Figure 3.20 From Table 3.26

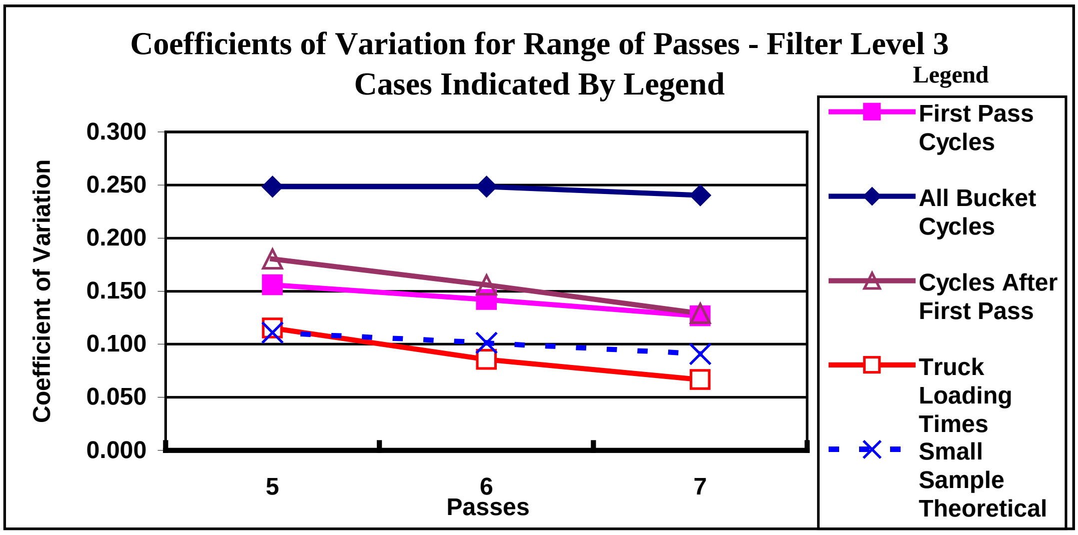


Figure 3.21 From Table 3.26

Table 3.27 summarises for sub-samples of 5, 6 and 7 pass truckloads at Filter Level 3:

- Average range, mode, median and mean values for all loading times – illustrated by Figure 3.22.
- A comparison of these same statistics reduced to indices based on an index value of 1.00 for the mean value of each sub-sample of bucket passes – illustrated by Figure 3.23.

Table 3.27 - Average Range, Mode Median and Mean Values of Loading Time for a Range of Loading Passes - Filter Level 3

Number of Passes	5	6	7
Mean Etc. Cycle Times - Seconds:			
Range Average	172.000	205.000	227.500
Arithmetic Mean Index	170.807	195.302	218.217
Median	168.000	192.500	214.000
Mode	168	193	211
Mean Etc. Cycle Time Indices Mean = 1.00			
Range Average Index	1.007	1.050	1.043
Arithmetic Mean Index	1.000	1.000	1.000
Median Index	0.984	0.986	0.981
Mode Index	0.984	0.988	0.967

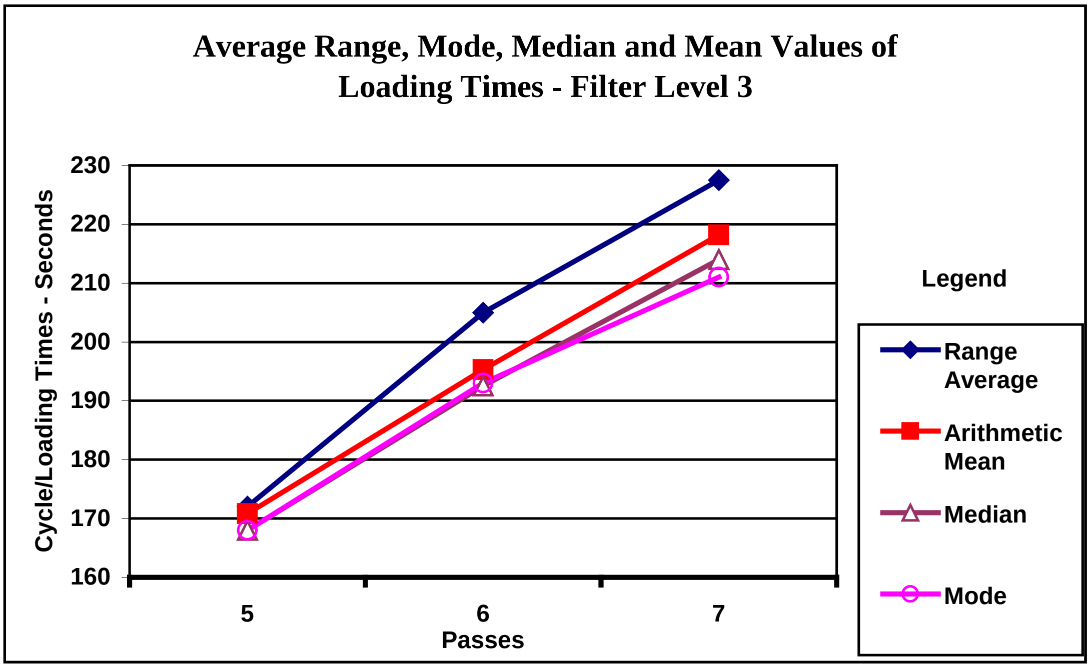


Figure 3.22 From Table 3.27

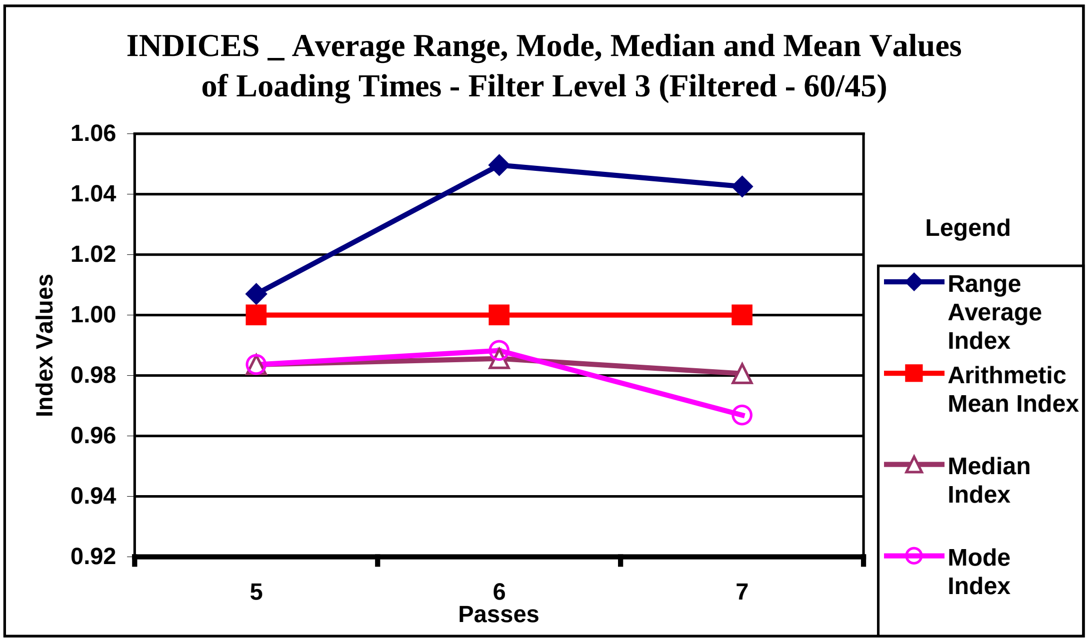


Figure 3.23 From Table 3.27

Interpretation and Implications

Bucket cycle-time data collected from actual operations is asymmetric – skewed positively (to the right). Analysis applied to the data indicates this is a robust distribution characteristic that persists into sub-populations of small samples of bucket cycle times that form truck-loading times. Filtering out anomalously high bucket cycle times has a centralizing effect tending to produce symmetric distributions of truck loading times that can be modelled by a normal distribution. Filter Level 3 described and analyzed is such a hypothetical case. This is considered a hypothetical trend that would not necessarily be replicated by operational dispersion control on actual bucket cycle data to reduce dispersion of truck loading cycle times by a continuous improvement initiative. But the central tendency resulting from reduced dispersion of bucket load data is a real characteristic that has been examined.

The several tables and figures described were reviewed and outcomes interpreted. The following illustrations of descriptive statistics over the range of 5,6 and 7 passes were compared:

- Figure 3.14 vs. 3.20 – mean bucket cycle times.
- Figure 3.15 vs. 3.21 – CV for distributions of pass groups and truckloads.

- Figure 3.16 vs. 3.22 – measures of central tendency for truck loading times.
- Figure 3.17 v. 3.23 – measures of central tendency reduced to relative indices.

The following trends and outcomes were noted:

1. Means of bucket cycle times, whatever the grouping, are generally consistent over the sub-sample range of 5,6 and 7 passes at all levels of filtering – Figures 3.14 and 3.20.
2. Filtering at Level 2 to eliminate time events non-intrinsic to the truck loading operation, by deleting non-complying complete truck loading records, reduced the mean of first passes by 16 to 18 seconds – Table 3.21. This is a measure of the effect of events non-intrinsic to truck loading operations for the operation and operating circumstances – equivalent to 7% to 8% of the actual mean loading time at Filter Level 1. This is considered towards the upper end of the range of the affect of "pit-house-keeping" activities on mean truck loading time. An appropriate range is considered to be best at 98% to a low performance of 90%.

   The outcome is that loading performance calculated from intrinsic equipment capability must provide for some "pit house keeping" duties – a factor in the range of 98% to 90% is suggested. This practical allowance should be separate and transparent. The actual effect is measurable and manageable.
3. First pass bucket cycle times at Filter Level 2 are reasonably uniform over the range of 5 to 7 passes. At Filter Level 3, with elimination of outlying high values, CV for first-pass bucket cycle times exhibit a reducing trend over the 5 to 7 pass range – Figure 3.21. This trend is believed to be anomalous, a by-product of the hypothetical filtering and reduced sample size; and is not considered as having any relevant significance.
4. Compared with Filter Level 2, filtering to Level 3 hypothetically reduced the mean for first pass bucket cycle times by some 15 seconds across the range of 5,6 and 7 pass sub-samples – Tables 3.22 and 3.26.
5. Filtering to level 3 also hypothetically reduced the mean for all bucket cycle times by a moderate 5 to 3.5 seconds over the 5 to 7 pass range of sub-

samples reflecting the combined averaging of the first pass adjustment over increasing passes for each sub-sample and the small general filtering reduction for all bucket cycle times subsequent to the first.

6. Means of truck loading times at both Filter Levels 2 and 3 reconcile with mean bucket times so confirming the arithmetic of the analysis.

7. Review of standard deviations, by applying sample theory as described in MPN 1 to MPN3, Mathematical Principles – Notes, over the range of bucket passes at both Filter Levels 2 and 3 – Tables 3.24 and 3.25 - indicate only fair correlation with standard deviations for truck loading times (see Figures 3.15 and 3.21).

8. At Filter Level 2 expected reducing trend of CV of loading times with increased number of passes is evident in Figure 3.15; and, more convincingly, evident in Figure 3.21 at Filter Level 3. Particularly at Filter Level 3, the gradient of the CV trend for truck loading times is steeper than for both passes after the first and for the small sample theoretical projection. A weighted combination of trend gradients for these two CV statistics correlates closely with the CV trend for truck loading times – likely as a result of the centralizing effect of the filtering process. So, as dispersion control is effected on bucket cycle times (by means discussed below), it appears that the small-sample down-trending of CV for truck loading times with increasing numbers of passes per load should become more effective. This down trending CV-effect should be reflected by reduced bunching inefficiency and improved load-and-haul “rhythm”.

9. It is generally concluded that increasing passes per truck load and so increasing loading time has real benefit in reducing dispersion of distribution of truck-loading times. But increasing passes is at the expense of increased truck loading times. The resulting reduced hauling productivity will, to some degree, be offset by the reduced variability in loading times that reflect in reduced bunching (queuing) inefficiency and so assist operators to retain “rhythm” in loading and hauling operations.

10. Filter Level 3 has not only centralized the distribution of truck loading times as illustrated by Figures 3.22 and 3.23 but has created bi or multi-modal

distributions of bucket load time and loading time distributions. Initial review of modal values indicated a general consistency across the range of passes reviewed and down through the filtering levels with some notable anomalies. These anomalies justified closer examination of modal values and interrelated measures of central tendency.

11. As described in detail in MPN 5, Mathematical Principles – Notes appended, Figure 3.24 - for bucket cycle times - and Figure 3.25, - for truck loading times - qualitatively illustrate the centralizing effect of level 3 filtering.

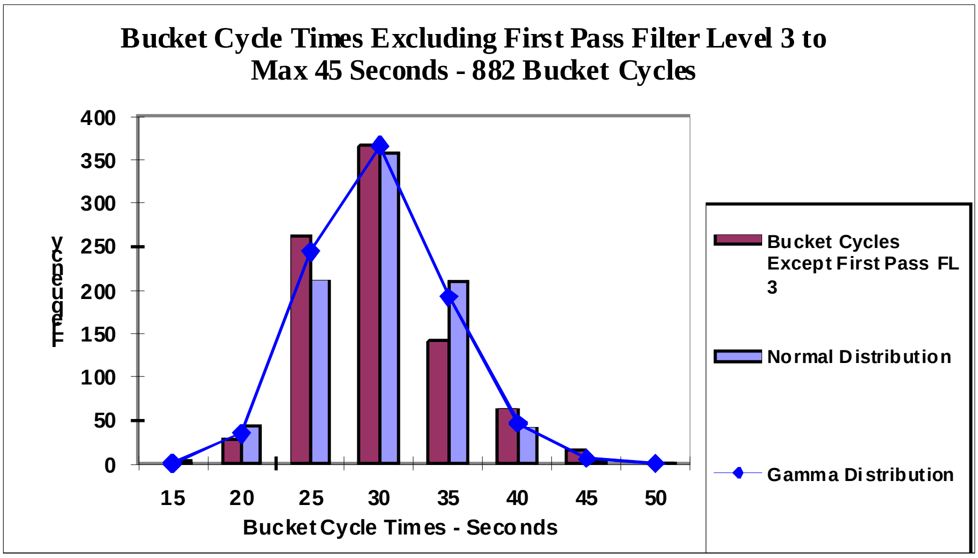


Figure 3.24 Table 3.25

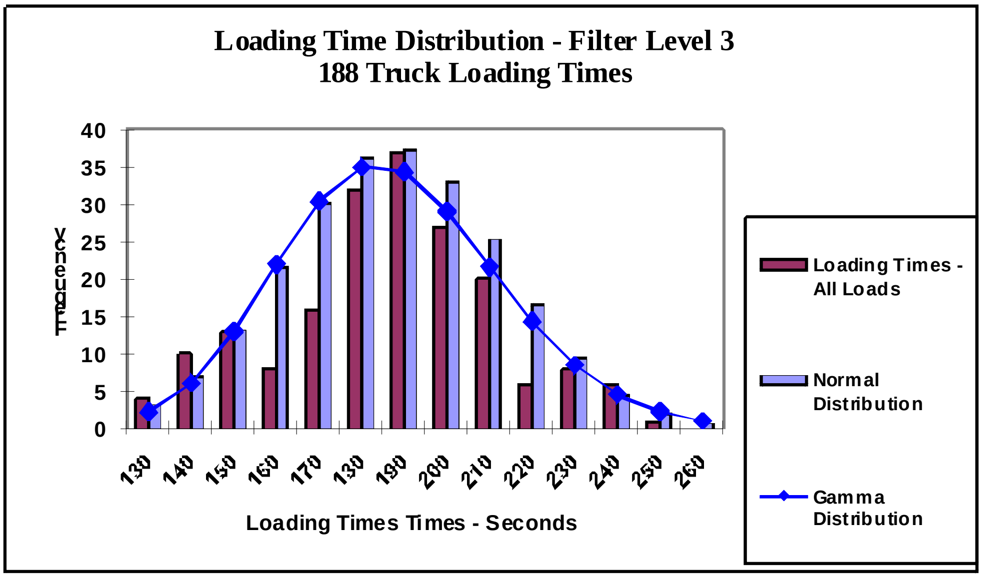


Figure 3.25 Table 3.25

“Mode” – A Suitable Measure of Central Tendency for Cycle Times?

General consistency of mode values that appear to be independent - to a significant degree - from both number of passes and the effect of outlying values, formulated the concept of adopting mode as the benchmark and control basis for bucket cycle times and truck loading times. Elimination of the influence of dispersive and mean-elevating effects of anomalous outlying values from analysis of bucket cycle times and truck loading times is obviously attractive.

Subsequent consideration and analysis of the efficacy of mode as a suitable measure of central tendency in the context of bucket cycle times and truck loading times proved to be substantial and lengthy. The considerations and analysis are recorded in MPN 5, Mathematical Principles – Notes appended.

Expected relationships between the several measures of central tendency were determined by compliance with the following empirical equation:

Mode = Mean – 3 . (Mean - Median) ……. (Chou, 1969)

Transposing: Mean – Mode = 3 . (Mean – Median)……………(3)

Discussion on mode in MPN 5, Mathematical Principles – Notes, comparative measures of central tendency and related descriptive statistics are summarized in

Table 3.28, Volume 2, Appendices. Mode values considered anomalous by comparison over the range of sub-samples were subsequently reviewed and amended to include alternative mode values and related statistics as shown in Table 3.29, Volume 2, Appendices.

The alternative values generally comply with the hierarchical order of measures of central tendency, i.e., mean > median > mode; also, values in Table 3.29 generally correlate with the empirical ratio (mean - median) **:** (mean – mode) = 1 **:** 3.

Alternative, amended, values of mode for truck loading times included in Table 3.29 are from analysis shown in Table 3.30, Volume 2, Appendices. The complete discussion, analysis, interpretation, implications and subsequent inferences are provided in MPN 5 in Mathematical Principles **–** Notes appended in Volume 2.

Small samples of truck loading times may exhibit multi-modal characteristics.

Where data demonstrates multi-modal characteristics of the distribution at any chosen frequency interval, increasing the interval of frequency – in this case from one second to five, ten and twenty seconds successively tends to interpolate a primary mode frequency range containing the unit frequency range modal value, as shown by Table 3.30.

On the basis of above discussion, and analysis described in MPN 5, Mathematical Principles **–** Notes it is concluded that, subject to some manageable limitations, mode is a reliable and, as will be shown, useful central tendency measure for bucket cycle and truck-loading times.

Skewness of Distributions of Bucket Cycle & Truck Loading Times

The validity of mode as a measure for bucket cycle time and truck loading time has been discussed. The natural positive skew of bucket cycle time and loading time distributions has also been discussed. Qualitatively from the several histograms included in Illustrations the positive skew is moderate to mild for truck loading times and generally high for bucket cycle times. (For more detail on the following discussion refer to MPN 6 and MPN 7, Mathematical Principles – Notes, appended in Volume 2). As shown by Table 3.32, Volume 2, Appendices, parameters $\boldsymbol{\alpha}$ and $\boldsymbol{\beta}$ were derived for modelling gamma distributions for comparison with actual frequency distributions - using the proposition $\boldsymbol{\mu = \alpha . \beta}$ and $\boldsymbol{\sigma^2 = \alpha . \beta^2}$ (Devore, 1999). Both $\boldsymbol{\alpha}$ and $\boldsymbol{\beta}$ proved to be relatively high numbers in order to generate a skewed distribution model tending away from the asymmetric towards a symmetric, normal distribution - particularly as the level of filtering increased. This tendency has already been discussed in empirical terms.

Mode is not generally considered a reliable measure of central tendency for highly skewed distributions because the mode is located too close to one of the extreme values to be representative of the distribution (Chou, 1969). But modelling experience as described supports the conclusion that truck-loading time distributions are not highly skewed.

A derivation of criteria for assessing degree of skewness was developed from Pearson's coefficients for skewness (Chou, 1999) as described in MPN 6, Mathematical Principles – Notes, appended. Measures of skewness were examined to test the hypothesis of using any of the several coefficients of skewness as a measure for control and improvement of bucket cycle and truck loading times.

As described in MPN 6, Mathematical Principles – Notes, appended, examination - as limited by the data sample available - of coefficient of skewness as an improvement or control measure was not absolutely conclusive; but the hypothesis that measures of skewness have potential to support or substitute for comparing the measures of central tendency is considered worthy of further research.

Forecasts - Management and Control

For samples of > 500 bucket cycle times and for samples of > 100 truck loading times, the evidence confirms stability of mode for measuring central tendency. This evidence promotes the concept of mode – the most frequently expected cycle or loading time - as a measure of minimum target or benchmark for estimating and forecasting bucket cycle times and truck loading times; also as a basis for directly related operational management and control.

Notwithstanding the analytical indications that bucket cycle times are highly skewed, as dispersion is reduced (by control of practices and techniques) centralizing tendency will tend to reduce skewness of bucket cycle times to moderate or even mild levels. This is evidenced by the descriptive statistics at Filter Level 3. Consequently mode is proposed as a potentially valid, reliable measure of central tendency in the case of bucket cycle times and truck loading times in the context of continuously improving management and control.

In addition mode can provide a target or bench mark for bucket cycle and truck loading times against which observed mean values can be compared. The difference between mean and mode values is a measure of the potential improvement except that achieving equal mean and mode values is a hypothetical, perfect outcome where distributions of bucket cycle and truck-loading times are symmetrical. This is considered a theoretical, impractical and non-achievable outcome that, modestly discounted, could be useful as a target for continuous improvement. Where the means of collecting bucket cycle and truck loading times exist, mean and mode values can be adopted as key performance indicators (KPI) and compared as a performance quotient (PQ_{BCT} for bucket cycle times or PQ_{LT} for truck loading times, generically referred to as PQ_T) with a practical target of:

Mode / Mean = PQ_T = 0.95, say (or other standard chosen)

The comparative standard corresponds to a $CV_{mean} \approx 0.16$ for bucket cycle times; and $CV_{mean} \approx 0.10$ for truck loading times consistent with a typical operation with 4 to 7 pass loading and mean passes of some 5.5. For other operations with less or more bucket passes $PQ_T = 0.95$ would likely still be valid but corresponding CV_{mean} and CV_{mode} values would change. The target statistics are consistent with an ideal operation where well-controlled dispersion of bucket cycle times and a focus on achieving best practice load-and-haul performance results in acceptable operating "rhythm".

For any of the sub-sample sets (constant bucket passes) of data from any sample of bucket cycle and truck loading times, the mean tends to diminish as management and control improvements reduce impact of outlying values on the distribution. In contrast it has been shown that mode is almost constant and so potentially more reliable and useful. So a modal coefficient of variation can be conveniently adopted as a dimensionless measure for comparing dispersion of distributions of bucket cycle time and truck loading times with differing numbers of passes.

That is: $CV_{mean} = \sigma$ / Mean

$CV_{mode} = \sigma$ / Mode

Both mean and mode coefficients of variation are compared in Table 3.31 in and Figure 3.26.

Table 3.31 – Summary - Coefficients of Variation Based On Mean and Mode

Number of Passes	5	6	7	All Passes
Filter Level 2 Mean Coefficients of Variation				
First Pass	0.389	0.390	0.448	0.395
All Passes	0.455	0.448	0.492	0.456
Cycles After First Pass	0.259	0.236	0.185	0.242
Truck Loading Time	0.182	0.180	0.153	0.201
Filter Level 2 Mode Coefficients of Variation				
First Pass	0.610	0.566	0.655	0.561
All Passes	0.636	0.552	0.631	0.608
Cycles After First Pass	0.308	0.251	0.204	0.277
Truck Loading Time	0.212	0.207	0.179	0.221
Filter Level 3 Mean Coefficients of Variation				
First Pass	0.156	0.142	0.127	0.144
All Passes	0.248	0.248	0.240	0.249
Cycles After First Pass	0.180	0.156	0.129	0.162
Truck Loading Time	0.115	0.086	0.067	0.137
Filter Level 3 Mode Coefficients of Variation				
First Pass	0.180	0.154	0.134	0.152
All Passes	0.303	0.303	0.288	0.292
Cycles After First Pass	0.201	0.160	0.142	0.173
Truck Loading Time	0.117	0.087	0.070	0.133

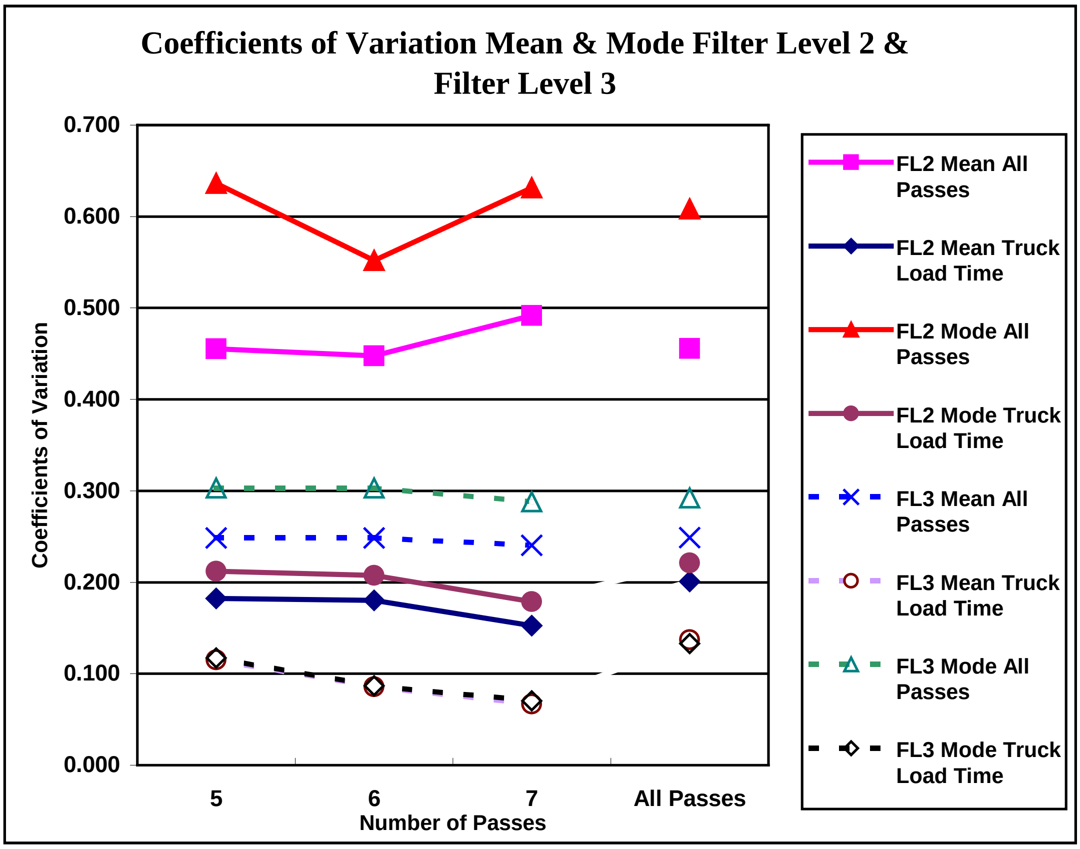


Figure 3.26 - Summary - Coefficients of Variation Based On Mean and Mode
From Table 3.31

From Table 3.31 and Figure 3.26:

For Bucket Cycle Times – 5, 6 and 7 passes:

At Filter Level 2: Range:

CV_{mode} 0.55 to 0.64

CV_{mean} 0.46 to 0.49

At Filter Level 3: Range:

CV_{mode} 0.29 to 0.30

CV_{mean} 0.24 to 0.25

Comparative "All Pass" values for bucket cycle times fall within the indicated range.

For Truck Loading Times – 5, 6 and 7 passes:

At Filter Level 2: Range:

CV_{mode} 0.21 to 0.18

CV_{mean} 0.18 to 0.15

At Filter Level 3: Range:

CV_{mode} 0.12 to 0.07

CV_{mean} 0.12 to 0.07

Comparative "All Pass" values for truck loading times are generally significantly higher than for the sub-samples of 5, 6 and 7 passes. There are two reasons for this apparent anomaly:

1. "All-Pass" truck-loading-time data having a range of values (measure of dispersion) that cover the total, cumulative, range of the 5, 6 and 7 pass sub-samples of truck loading times. The increased cumulative dispersion of the "All Pass" data indicated by overall range is reflected in the other measures of dispersion, particularly standard deviation σ. Increased σ results in increased CV_{mode} and CV_{mean} - so the higher CV values for truck loading times for "All Passes" in Table 3.31 and Figure 3.26.

2. “All-Pass” bucket cycle and truck-loading-time data includes some 4-Pass sets and one 3-Pass set of bucket cycle time data that, typically, have similar standard deviation to 5, 6 and 7 pass sub-samples but tend to lower the mean value of truck loading times – so elevating the CV of “All-Pass” truck loading time.

It will also be noted that, as discussed:

- At hypothetical Filter Level 3 the CV_{mean} and CV_{mode} values for truck loading times are practically the same.
- The reduction in both CV_{mean} and CV_{mode} for truck loading times as number of passes increase is clearly indicated at both Filter Level 2 and Filter Level 3.

In conclusion, Table 3.31 and Figure 3.26 demonstrate that, to the extent that mode is a stable and valid KPI for central tendency of distributions of bucket-cycle and truck-loading times, a CV derivative of mode is also a valid alternative for measuring comparative dispersion if coefficient of variation is the comparative descriptive statistic of choice.

Conclusions

Basic elements of bucket cycle time consist:

- Select and line up digging target in face (in final braking stage of return swing), address the face, fill bucket, break out and commence to swing.
- Swing loaded to dumping spot.
- Dump bucket load.
- Return swing.

These basic components are further addressed in Sections 3.3.11 and 4.1.4.

The following conclusions are drawn from analysis and discussion in this section and related supporting information in Mathematical Principles – Notes, Tables and Figures appended.

1. Raw bucket cycle and truck loading data records include time utilizations and losses in addition to the basic elements described that impact directly on bucket cycle times. Activities non-intrinsic to the loading operation may inflate bucket cycle and truck loading times. These time components have been discussed. On the basis of limited operating data available and

knowledge of the actual operating conditions, a discount factor in the range 0.92 to 0.98 is indicated as justified.

2. Time is utilized for necessary activities that enable operations to progress including face housekeeping necessarily performed by loading equipment, maneuvering in a limited operating area, dealing with oversize particles in the face and following the face as excavation progresses. This category of time is treated by a propel factor in Section 3.2.4 – recommended at 0.85 by Atkinson (Atkinson T, 1992). In the absence of empirical determination of a suitable average or allowance range, factor application is the alternative option. This time utilization category needs to be transparently assessed, allowed for and, if possible, separately recorded in operations to enable examination of nett purely-intrinsic bucket cycle time and truck loading time data for control and management.
3. Truck Exchange Time is the difference between "turn-and-spot" time for mining trucks and time taken for the first bucket cycle from last dump and truck dispatch to the position of "dump-ready" spotting position for the operator of the next truck to be loaded. The general issues and magnitude of truck exchange time have been discussed in some detail in Section 3.2.6 – specifically the influence on intrinsic productive efficiency of loading equipment. It is paradoxical that the smaller the number of passes taken to load a mining truck the lower the intrinsic loading efficiency; but the higher the intrinsic hauling efficiency.
4. On the evidence of the data sample available (2,508 bucket cycle times and 436 truck loading times) bucket cycle times are highly positively skewed. This is consistent with experienced, intuitive expectations. Bucket cycle time data is expected to be truncated at a minimum of some 15 seconds with mean values nett of all non-intrinsic time losses and inefficiencies generally in the range 20 to 45 seconds depending on the type, size of loading equipment and operating conditions; but with maximum values that can include high outliers resulting in exaggerated positive skewness.
5. Compared with bucket cycle times, truck-loading times, as expected, exhibit central tendency. Degree of skewness and dispersion is generally lower. Accordingly CV values for truck loading times are likely in the range 0.2 or

less with reducing CV values as the number of passes increases; but, naturally, increased mean truck loading time.

6. Reduced dispersion of truck loading times tends to reduce the degree of bunching inefficiency – as discussed in Section 3.4. Reducing dispersion of truck loading times is a desirable management initiative in the interests of improved “rhythm”, increased productivity and cost benefit improvements.
7. Analysis of actual bucket cycle and truck loading time data provides empirical evidence that reduction of dispersion of bucket cycle times will have a centralizing effect on the distribution of truck loading times; i.e., an effective reduction in the dispersion of truck loading times and improved “rhythm”.
8. Mean truck loading times decrease with reduced number of bucket passes, theoretically increasing the productivity of hauling; but, as expected, the CV for truck loading times increases implying that short truck-loading times will tend to exhibit increased dispersion that manifests as increased bunching inefficiency and less “rhythm” – discussed in Section 3.4.
9. As operational improvements are effected to substantially eliminate time utilisation and time losses non-intrinsic to the loading operation, consequent dispersion reduction of bucket cycle time distributions will be reflected in significant reduced dispersion of truck loading times – in addition to any dispersion reduction from increased number of bucket passes.
10. Reducing truck loading time dispersion to realise benefits by way of reduced bunching and improved hauling “rhythm” can be achieved by any of the following individually or combination:
    - Ensuring that the material to be loaded is in condition suitable to realise optimum (best practically possible) bucket filling times within each bucket cycle time – this benefit is complementary to the significant benefit realised by higher bucket fill factors, reduced bucket load dispersion, flowing on to improved control of truck payload dispersion and containing truck payloads within an acceptable confidence interval.
    - Identifying and treating any extraordinary time utilization and losses non-intrinsic to loading operations (suggested to reduce intrinsic loading time to 92% to 98% for the individual operation where data analysed in this

section was sourced) to reduce impact on productivity to the degree practically possible.

- Minimizing any time utilization and losses integral with loading operations so reducing dispersion of bucket cycle times to best-practice levels that, in turn, will result in truck loading times with relatively limited dispersion realizing the benefits of reduced bunching inefficiency and optimal operational "rhythm".

11. To facilitate control and improvement management, valuable measures of central tendency mean and mode will indicate current average loading time and the potential hypothetical best outcome.
12. A contrived descriptive statistic with potential benefit as a management aid: –

    Productivity Quotient = PQ_T = Mode/Mean

    - with target value of 0.95 as a starting point is proposed as a valuable KPI for efficacy of time management – lower values of PQ_T indicate under-performance.
13. To facilitate comparisons of measures of dispersion not totally dependent on mean values, a coefficient of variation based on mode is proposed in the course of the analysis and discussion.
14. Comparative skewness of bucket cycle and truck loading time distributions also provide alternative descriptive statistics of potential value for measuring management control of dispersion of distribution of bucket cycle and truck loading times. More detailed discussion is provided in MPN 6, Mathematical Principles – Notes, appended in Volume 2.

It is necessary to consider the limited dispersion-reducing effect of increasing the number of loading passes, particularly for payload distributions; but also for loading times. The following comments are relevant:

- In Section 3.2.8, and this Section 3.2.9, it has been implied that increased number of loading passes results in reduced dispersion of bucket loads and payloads - evidenced by theoretical analysis and empirically by CV statistics.
- The implication applies for bucket loads and payloads where distributions tend to be robustly normal, i.e., symmetrical.
- For bucket cycle times and loading times, as discussed in this Section 3.2.9, when practical skewed distributions are hypothetically stripped of outliers

non-intrinsic to actual loading equipment cycling, the resulting hypothetical distributions exhibit centralizing tendencies to which reduced dispersion – lower CV – applies; and increased number of passes does have a beneficial effect by reducing dispersion, but at the expense of increased loading time.

- Between the practical limits of three passes and eight passes, increasing the number of bucket passes tends to have reducing dispersion-reduction benefit.
- Table 3.33 provides the relative effect on CV of increasing passes by assigning an index of 1.00 to the CV for a single pass with increased number of passes aligning with reducing CV index. The percentage change for each digital addition to number of passes is also provided in Table 3.33.
- Figure 3.27 illustrates trends of the data in Table 3.33 – particularly the diminishing benefit of CV reduction as number of passes increase.
- The gradient of the curves in Figure 3.27 are steepest for the range three to eight passes. For nine passes or more the trends tend to be asymptotic – i.e., increased number of passes above eight delivers substantially diminished returns in dispersion reduction.
- So practical benefit in CV reduction, indicative of distribution dispersion reduction, is limited to the range three to eight passes with the greatest potential benefits realised from 3 to 4 (87%), 4 to 5 (89%), 3 to 5 (77%) and 4 to 6 (82%).

Table 3.33 - Coefficient of Variation Index v. Number of Bucket Passes Truck Payloads and Truck Loading Times

Number of Passes	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**
CV Index	***1.000***	***0.707***	***0.577***	***0.500***	***0.447***	***0.408***	***0.378***	***0.354***	***0.333***	***0.316***	***0.302***	***0.289***
Change %	**0%**	**70.7%**	**81.6%**	**86.6%**	**89.4%**	**91.3%**	**92.6%**	**93.5%**	**94.3%**	**94.9%**	**95.3%**	**95.7%**

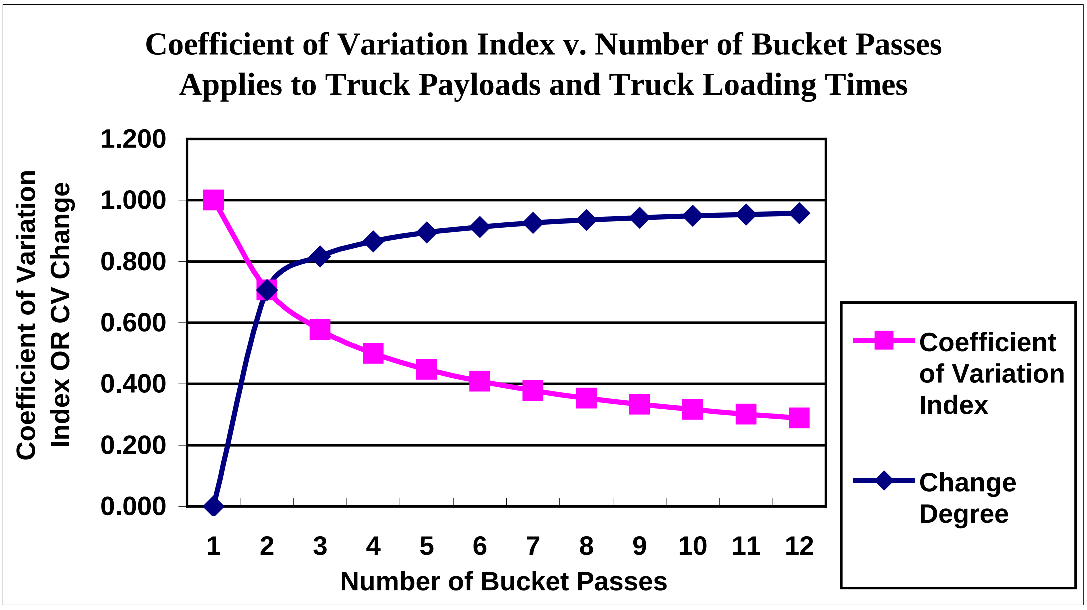


Figure 3.27 From Table 3.33

For the largest mining trucks, where loading equipment options to genuinely four-pass load are limited, it may both benefit productivity and reduce costs to opt for five or six pass loading. When all factors are taken into account, benefits from the larger number of passes will tend to offset, possibly totally or exceed, the expected benefits from short loading time realised from four-pass loading of large mining trucks. This is further discussed in following sections.

3.2.10 Bucket Dumping

Preliminary

Differing dumping characteristics of buckets through the range loading equipment types is a necessary, if minor, consideration in the selection of suitable loading equipment for any open-pit mining application.

The effect on mining trucks, particularly impact of material discharge from buckets, needs to be understood both qualitatively and quantitatively.

Bucket Dumping Styles and Truck Body Issues

In ascending order of impact on trucks each dumping style and action is briefly described below:

1. Bottom opening or clamshell buckets on hydraulic shovels are opened and closed by a dedicated bucket hydraulic cylinder with modulated control. This

arrangement enables dumping with low clearance between bucket and truck, controlled discharge; and if necessary partial load dumping. Clamshell buckets on hydraulic shovels can also be rotated forwards with bucket tilt cylinder(s) to dump a large particle so avoiding any possibility of the particle bridging and jamming in the bucket – altogether the most versatile bucket style and facility over the range of loading equipment.

2. Hydraulic excavators, with buckets of cutting-edge width decreasing through the range mass excavation (mining applications), normal and trenching, dig with a backhoe action. The excavator works from the top of the face and the bucket tilt cylinders rotate the bucket away from the face to break out when bucket is full. When filled the crown of the load is under the stick. With excessively high bucket fills in hard rock the load can damage the underside of the stick. When discharging, the bucket tilt cylinder(s) provides modulation control. Dumping height can be at low clearance – sufficient for bucket teeth to clear the truck floor or payload when the bucket is reverse rotating in an “un-curling-like” action. Impact on trucks can be controlled and comparatively “soft” - similar to clamshell buckets.

3. Front-end wheel-loader (FEWL) buckets have significantly larger capacity than a hydraulic or rope shovel of equal operating weight. Dimensions of FEWL buckets reflect the special purpose shape. Typically bucket width **:** height and bucket width **:** depth ratios are large. The bucket must have sufficient width to protect the front tyres when the loader is crowding the face to fill the bucket. Generally available dumping height is more limited with FEWL than with shovels and excavators. Wheel loader operators tram forward with the bucket at dumping height with the bucket held in the carry position until the hinge pins (bottom bucket hook up and centre of rotation for bucket dumping) clear the side rail of the truck. The FEWL is braked to stationery before front tyres collide with the truck. Simultaneously the bucket is rotated forward by the bucket cylinder(s) to dump the load. Bucket cylinder(s) provide modulation during dumping to control discharge rate to minimize impact on the truck – particularly the first load is controlled to establish a protective layer on the floor of the truck body. Later bucket loads are dumped with more speed. The last pass is carefully placed over the load

centre of gravity on the truck body if provided and, in default, over the bottom trunnion of the body lift cylinder as a guide. FEWL operators do have some facility to centralize each bucket load on the centerline of the truck body by crowding forward using the bucket as a bulldozer to move material across the truck. Best-practice operators develop a technique of flicking the tailings of each bucket load off bucket floor and cutting edge and point adaptors using the fast, but modulated, bucket cylinder control. This technique minimizes carry-back in the bucket.

4. Rope shovel buckets have digging-geometry limitations in that the bucket pitch relative to the stick is adjustable only with pitch braces - short turnbuckle struts - between bucket and stick. But the pitch braces can only be adjusted when the shovel is not operating. Unlike hydraulic shovels, excavators and FEWL, digging geometry of rope shovel buckets cannot be adjusted on the fly whilst operating. Activating the latch securing the trapdoor bottom of rope shovel buckets dumps total bucket loads. The load discharges under gravity pushing the trapdoor clear. Snubbers on the hinges control swing of the trapdoor. Snubbers are travel dampers (traditionally) using Beleville spring washers, elastomer pads or hydraulics that positively damp and limit trapdoor swing to avoid damage by impact between the bucket trapdoor and the truck body. The dumping system of rope shovels precludes any modulation or control of bucket dumping. It should be noted that the pitch angle of the bucket generally prevents free fall of the total bucket load into the truck body. Typically bucket loads slide down the angled front face of the bucket so reducing velocity of discharge.

Generally:

- Rope shovels are time-proven well established technology with only moderate evolutionary change and improvement to bucket design and appointments. Setting up rope shovel geometry for optimum digging is an art producing a configuration that is fixed until loading operations are suspended to make adjustments.
- Hydraulic shovels and mass excavators have the advantage of higher versatility compared with rope shovels. Partial discharge is possible but rarely

practiced. Operators appear to prefer taking a reduced bucket load to top up a truck payload. Adjustments to digging geometry of hydraulic loading equipment can be; and as a facet of digging technique are, made at any time without interrupting loading operations. If necessary, top up of truck payload by less than a full bucket load can only be effected by taking a partial last bucket load.

- FEWL bucket technology is well established with evolutionary improvements limited to bucket cross-section profile to facilitate bucket filling and detailed design for reduced wear, improved reliability and extended bucket life. Partial discharge is convenient if necessary. Adjustments to digging geometry are convenient at any time and part of normal operating technique. FEWL buckets can serve as a bulldozer for face and dump housekeeping with the penalty of accelerated underside wear of cutting edge, points and adaptors and sacrificial wear plates on the underside of the bucket.

The apparent advantage of hydraulic loading equipment compared with rope shovels in this limited comparison must be considered along with all other comparative characteristics and more importantly:

- Optimum choice for the operation.
- Productivity.
- Owning and Operating unit cost of production.

The two volumetric standards for bucket capacity are:

- *Struck* – essentially water line volume taking a line through the back wall top edge to the base of the bucket points (teeth) - adopted by rope shovel OEM.
- *SAE 2:1 Heaped* – in addition to the volume to the water line a pyramidal topping projected with slopes of 2 horizontal to 1 vertical is superimposed - adopted by OEM offering hydraulic shovels, excavators or FEWL.

Bucket capacity of rope shovels is conservative in comparison with other loading equipment standards. This is a consideration in pre- selection investigation of loading equipment. But a more important consideration, for the loading equipment options under consideration, is the bucket load factor and the actual bucket load per pass, both volumetrically and in terms of weight that will determine the target mean number of passes to meet the target truck payload (mean) and to match the volumetric capacity of the truck body.

Truck Body Capacity

Of the several standards for volumetric capacity of mining truck bodies the most commonly used is the SAE 2:1 rating. The truck body is assumed completely filled to the side rails with a 1:1 projected slope from the rear discharge edge to side-rail height. A pyramidal super-load above the body side rails is projected at a slope of 2 horizontal **:** 1 vertical. Although comparative this protocol is not realised in practice as some parts of the load, particularly the four corners within the body and above the side rails, adopt a naturally-reposed, conical shape. Also the sharp pyramidal apex is, in practice, an artificial concept.

There has been substantial recent attention to this issue. Third party truck-body design-and-suppliers market on the basis of designs for truck-body volume capacity that are more realistic than the SAE 2 **:** 1 standard. This issue is revisited in more detail in Section 3.3.5.

3.3 MINING TRUCKS

3.3.1 Selection Process

The introductory discussion on loading equipment in Section 3.2.1 is equally applicable to selection of mining trucks for a specific application and production plan.

The issue of priority of selection – loading equipment or transport equipment – was briefly introduced in Section 3.2.1. The concept of capacity hierarchy was also introduced and discussed in Section 3.2.1. It was noted that capacity of loading equipment generally tends to exceed total mining production requirements; also tends to exceed the intrinsic capacity of the truck fleet. Evidence of "over-shoveling" from bench marking studies was discussed in Section 3.2.4. This implied loading inefficiency is complemented by a benchmark truck utilisation in Australian metalliferous mines of 50%. That is 50% of annual total hours of 8,760 (Tasman, 1997) with even lower benchmark utilizations in Australian open pit coalmines.

It is believed these benchmark utilizations are ultra-conservative outcomes of open pit mines mostly producing feedstock for steel mills and power generation. These comparatively low utilizations are believed to be a symptom of mature producers on long-term contracts requiring high reliability of supply and with commodity prices largely cost-driven rather than subject to robust market-place competition. Security of supply and consistency of product specification are both important factors, the influence of which tends to shape mining equipment asset management and operations management strategy in these specific sectors of the mining industry that constitute a large proportion of open pit mining commercial activity in Australia.

Open pit mines producing precious metals and stones; also high-value base metal producers, where an open international competitive market largely controls prices (albeit clouded to some degree by the affect of forward sales and futures trading), necessarily adopt different equipment inventory strategies. In a highly competitive, commercial environment, investment in mining equipment, particularly loading and hauling equipment, tends to be more closely tailored to firm productivity and cost budgets to realise competitive cost of production – below the median and preferably in the lower quartile of production costs. This cost-competitive environment can be expected to yield best-practice outcomes. These commercial characteristics are the backdrop and basis for considerations to follow with overall truck utilisation assumed to be in the range 65% to 75%.

The risk accepted by selecting loading equipment in isolation was discussed in Section 3.2.1. A similar risk is attached to selecting mining trucks without simultaneous consideration of matching loading equipment. Because haulage in open pit mines contributes a relatively large proportion of total mining costs, 3 to 4 times the unit cost of loading, it is logical that equipment selection should focus on the mining-truck options to meet production programmes and then to select loading equipment from available options to suit the trucks. This is the general policy recommended and adopted for the purposes of the research described in this thesis. But mining trucks and loading equipment are so interrelated by cross effects on mining productivity and costs that some early consideration needs to be given to suitable loading equipment, - type, size and indicative numbers in concert with determining type, size and indicative numbers of mining trucks. This will ensure that when mining truck selection is finalized that loading equipment selection is essentially confirmatory, without the necessity for major scale and specification changes.

Truck selection should be the principle focus but with all necessary consideration of interrelated loading equipment selection issues; and, to a lesser degree, support equipment, i.e., pit house keeping mobile equipment including dozers. Graders water trucks and specialised road construction equipment.

3.3.2 Cyclic versus Continuous Haulage

As indicated in Section 3.2.2, cyclic, conventional loading equipment serviced by cyclic mining trucks – specifically conventional rear-dump, off-highway trucks are the focus of this thesis. This type of haulage unit is very flexible and imposes fewer constraints on open pit mine design compared with alternative truck types described in the following section.

3.3.3 Truck Types and Application

Mining Truck Types

Off-highway trucks can be conveniently classified as three types:

1. Conventional rear-dump.
2. Tractor-trailer - bottom or side dump.

3. Integral bottom and side dump.

The three types are illustrated by Figure 3.3.28 at right.

Advantages and Disadvantages

Ronald Hays tabulates advantages and disadvantages for the three identified types (Hays, 1990 Table 1 p673).

Amongst the advantages for conventional rear dump trucks Hays lists: “Provides maximum flexibility” and “Versatility, can haul a variety of materials”. It is these two attributes, flexibility and versatility, that make conventional rear dump trucks the more generally applicable and utilised truck type.

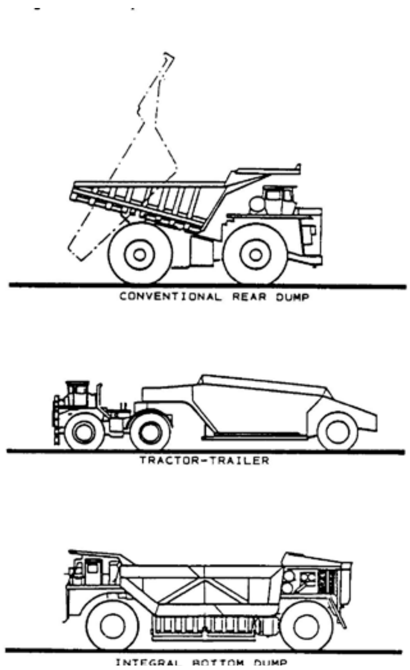


Figure 3.28
Mining Truck Types
(Hays, 1990)

The term “mining truck” is used throughout this thesis for conventional off-highway rear dump trucks.

It must be noted that trucks used in mining include all of the three types illustrated by Figure 3.28; also articulated trucks in two axle and three axle configurations, particularly smaller rear dump trucks have been adapted to underground application along with development of special- rear dump trucks of smaller capacity combining some of the features of tractor-trailer and conventional rear dump trucks. Highly maneuverable articulated trucks using two axles have been developed for short hauls in difficult operating conditions with poor, if any, roads in open pits for mineral sands and other shallow weathered-zone deposits that generally experience dewatering problems. Some “mining trucks”, particularly the specialised, articulated designs, are used on confined construction sites where roads are temporary and generally poor standard. Conventional mining trucks are also generally applied to large-volume medium haul bulk earthworks projects such as earth-and-rock fill dams and large road and railway formation works.

Payload (PL)-to-Gross Machine Weight (GMW) ratio is a most significant initial basis for comparing mining trucks. Conventional rear dump trucks are disadvantaged by a generally lower payload to GMW compared to other types illustrated in Figure 3.28 above. See Table 34 below – (Hays, 1990, p673, Table1).

Table 3.34 – Truck Payload v. GMW Ratios

Truck Type	Truck Payload/GMW	Comments
Conventional Rear Dump	0.55 to 0.60	Relatively Disadvantaged
Tractor-Trailer	0.60 to 0.65	Significant Advantage
Integral Bottom (or Side) Dump	0.60 to 0.70	Substantial Advantage

Truck types exhibiting the more favourable ratios indicated above, particularly the integral bottom or side dump truck, have encouraged OEM to adopt some of the design benefits of these trucks for the larger (150 tonne payload plus) conventional rear dump trucks – obviously tending towards the unconventional.

Two major categories of design initiatives by OEM and third party suppliers:

1. In an attempt to convert body weight to payload third party body design-and-construct suppliers saw a market opportunity to supply light weight bodies with weight saving features such as fabrication from hardened and tempered alloy wear steel and wear plates limited to high impact areas rather than mild steel plate with full liner plates. OEM responded to this competition with site/application-specific designs, e.g., Caterpillar's "Mine Specific Design" – "MSD" generic body designs for larger mining trucks.
2. Creative designs that reduce frame weight and upgrading body design to a stressed quasi-monocoque design suspended between rear trunnions and a forward bearing support for an overall tare saving, e.g., Liebherr's Ti 272 – at present experiencing a protracted testing period. Liebherr offers the Ti 272 but to date trucks have been placed on trial. It is understood that Liebherr intend to work towards industry acceptance of the Ti 272 concept with any necessary future modifications indicated by early field experience (Kloverkorn H, 2004).

It is the author's opinion that potential upside for optimum rear dump configuration for larger mining trucks has not been fully exploited at this stage. Improved payload to GMW ratio appears to have the best potential, and may be the only practical potential, for improved productivity and unit owning and operating costs for large mining trucks It could be reasonably expected that all OEM are pursuing all potential improvements in payload to GMW ratio for improving the current, and developing the next generation of large mining trucks.

Conventional rear dump trucks generally have a higher power-to-weight ratio than other truck types. Together with loaded GMW distribution of 33% on the steering wheels and 67% on the dual driving wheels, high power-to-weight realizes better gradeability for deep pit operations.

Larger conventional rear dump trucks generally have higher average wheel loadings that are more demanding on tyres leading to speed-haul distance limitations compared with other types of trucks. Wheel loadings and tyre performance for the various truck types will be considered in more detail in 3.3.7 below.

3.3.4 Intrinsic Truck Performance

Performance of all off-highway trucks, including mining trucks of the conventional rear-dump type, is a function of the following:

Componentry – An Historical Review

Drive Line Components – typical simplified lists:

Mechanical Trucks: Engine, torque converter, transmission, differential and final drives.

Electric Drive Trucks: Engine, alternator, converter, DC wheel motors with integral reduction gear train and mechanical final drives; alternatively, from converter, inverter, AC drive motors with integral reduction gear train and mechanical final drives.

Historically OEM shared many of the above components.

OEM commonly engineered trucks around available components from three major engine suppliers, two major torque converter/transmission third party suppliers, and three electrical driveline component suppliers.

More recently two OEM have emerged from the industry rationalization as suppliers of the full range of mechanical drive developing their own engines, transmissions and complete drivelines. Obviously some of the componentry is outsourced but supply is to purchasers' specification and standards and, often, designs.

The field of OEM manufacturing electric-drive trucks has been rationalized from four majors and as many second tier manufacturers to four enjoying significant market share. OEM continue to supply large mining trucks with designs engineered around electric-drive components from three major third-party suppliers. Engines are sourced from two major suppliers and offered to customers to make a choice

With substantial commonality in designs, specification and even componentry, differences in intrinsic performance of mining trucks offered by any OEM tend to be more subtle than substantial.

Early development of off-highway trucks was essentially an extrapolation of configurations and designs of conventional highway vehicles. Engine, clutch, transmission and driveline to the rear wheels was typical in the earliest off-highway trucks. Power-shift transmissions (first seen in early Ford Model T vehicles some 100 years ago) and hydraulic torque converters were to follow with the next advance to automatic-shift transmissions.

Development of mechanical drive mining trucks stalled at 50 tonne capacity in the 1950's encouraging the adoption of DC electric traction drives from railway locomotives to increase truck capacity successively through 65, 75, 85 100, 120 to 150 and ultimately 200 tons (of 2,000 lbs). Mechanical drive trucks eventually caught up increasing in capacity through 85, 100 to another plateau at 120 ton trucks due to non-availability of transmissions rated at 900kW or more. Over the most recent 20 years OEM have developed and offer trucks up to 360 tons with a choice of mechanical or AC electric drive with DC drive losing favour for the larger capacity mining trucks.

The efficiency advantage from engine flywheel to road of mechanical drives compared with DC electrics has encouraged development of AC wheel motors that closely compete with mechanical drive efficiency for the largest mining trucks. The new AC systems have subsequently been retroactively developed down to 120-ton trucks.

Steering and braking performance must comply with recognized international standards from SAE, ISO or standards endorsed in the country where trucks will operate. All OEM can be expected to ensure that steering and braking systems have substantial excess performance above the minimum standard to ensure compliance after some moderate deterioration consistent with reasonable wear and tear before mandatory maintenance and to allow for varying operator efficiency. But steering and braking systems are still cost-driven designs (to a low degree of acceptable residual risk) that can be expected:

- To have comparable rather than significantly different performance over the full range of truck capacities and for all OEM.
- To have warranty limited to a maximum GMW or payload – Caterpillar's policy is a maximum over-target payload of 120% (some 11% of GMW).

Similar design-and-supply philosophy generally applies through all componentry for large mining trucks.

Engines for large mining trucks have traversed several phases of development. All engines for mining trucks use Diesel's cycle with relatively recent historical trading off of extended torque curves in favour of engine speed – "over-square" engines. Even more recently there has been a reversal of this trend with increasing engine stroke delivering the advantage of lower BMEP (brake-mean-effective-pressure), longer engine life and, importantly at current prices, improved fuel economy.

The study of engines for large mining trucks, the design issues, future opportunities including lubricant disposal through supplementation of fuel is a substantial research subject in its own right.

Debate continues between operators, OEM and suppliers of large mining trucks on the relative merits of mechanical compared with electric-wheel, particularly AC, drive.

Mechanical drive componentry modifies the conventional umbrella-shaped speed-torque curve of diesel engines to a flatter torque curve in saw-tooth form by changing transmission ratios so providing an extended torque range many times the range of the useful torque range of engines.

DC drives have been disadvantaged in that the speed-torque curve with its spectacular low speed rise to peak torque tends to fall away quickly so limiting truck acceleration to a short initial period.

Torque-speed characteristics of AC drive have a much flatter and extended torque-speed curve that more closely follows the mechanical drive attributes.

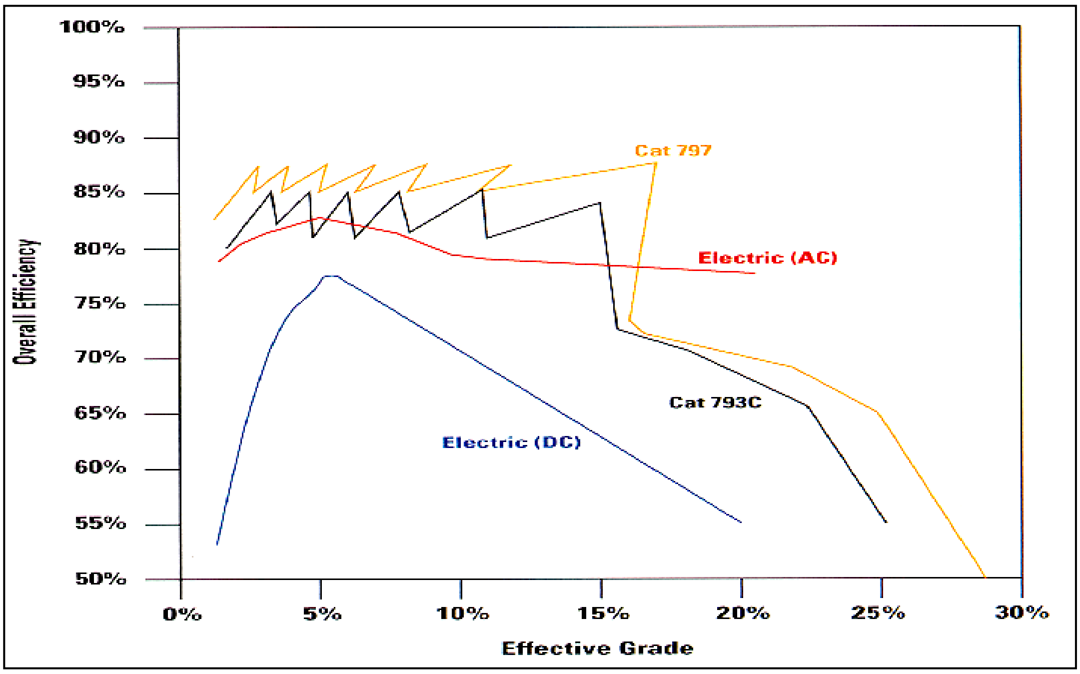


Figure 3.29 - Mechanical v. AC v. DC Drive Efficiencies (Caterpillar Australia)

Figure 3.28 (Caterpillar, Australia), above indicates average relative driveline efficiencies over the useful working range - engine flywheel to road - as follows:

Driveline Type:	Maximum %:	Minimum %:
Mechanical	83*	83*
Electric – AC	81.5	79
Electric – DC	77.5	70

* - Ignoring the Cat 797 torque-speed data as being atypically high; but the 797 does appear to have some advantage in comparison with like-capacity AC drive large mining trucks.

The above values are for normal working range up to 15% effective adverse grade.

The efficiency advantage currently enjoyed by mechanical drives is obvious – as is the closing in of electric wheel drives with the change to AC. There is potentially more upside potential in newer AC technology than in the more-dated mechanical drive technology. The future competition between OEM in this area will be fascinating and likely will deliver significant benefit for open pit mining operators.

Fuel burn rates are directly related to the above-indicated efficiencies for trucks of equal GMW. So fuel economy has been the principal focus in comparing alternative truck drive systems. Recent fuel price increases seem to be persistent and likely to have long-term effect making the comparison focus even more acute.

Characteristics of drive system options:

Mechanical Drive: Driving response and travelling characteristics closest to on-highway vehicle, fuel-efficient, very mature technology development, with comparatively less potential for improvements. Starting torque modulation control is an advantage on wet or soft benches.

AC Drive: Driving response has specific characteristics, starting torque can be very high so heavy foot = wheel spin with increased risk of tyre cutting on wet benches, electric resistance braking response is different requiring different skill in application, close to mechanical drive fuel efficiency, improved electrical efficiency compared with DC, more recent technology with significant improvement potential.

DC Drive: As the forerunner, exhibits driving characteristics similar to AC drives. Dated technology that is disadvantaged in terms of efficiency, performance and cost effectiveness compared with mechanical and AC drives. DC appears to be phasing out in favour of AC drives, introduced for the largest mining trucks and in process of retroactively replacing DC drive systems.

Which drive system is best? There is no clear favoured system. The decision to opt for either mechanical or AC drive may rest on the comfort the owner/operator has with electric drive technology, physical conditions in the open pit including haul road profile, wet and soft running conditions in loading areas with potential for tyre damage and road construction and maintenance standards.

In the broad view, differences, although measurable in intrinsic performance terms, in the ultimate productivity and cost differences tend to be subtle. Selection needs to focus on greater differences such as product support, how the selected truck fits into local skills demography, pit geometry and environmental issues. High top speed is of little use if the opportunity to use it is limited or denied by operating circumstances or other limitations such as tyres, overheating in high ambient temperature.

3.3.5 Body Selection and Truck Payload

Body Development and Experiments

Evolution of body designs has created a classifiable range that is described by Ronald Hays (Hays, 1990, p675) as follows:

"Rear dump bodies are available in several basic styles with the following most common:

> Transverse V-shaped, flat bottom floor;
>
> Transverse V-shaped, longitudinal V-shaped floor;
>
> Horizontal, flat bottom floor;
>
> Horizontal, longitudinal V-shaped floor."

"Transverse V-shaped" describes the side profile of the truck body – alternatively termed "wedge" design - currently the industry standard profile. Current body designs are tending to favour flat bottom floors – particularly favoured by special weight-saving designs, such as Caterpillar's MSD (Mine Specific Design) body options, offered for larger mining trucks.

Since 1990 evolution of mining trucks has marched on with several notable advances to enhance performance and contain costs:

- Initiatives in modular design to facilitate maintenance described in more detail in Section 3.3.8 below.
- Reduction of tare, or "nett machine weight" (NMW), as it is more often termed in the industry, to realise increased payload for a constant GMW. As indicated above, recent body design philosophy, initiated by third party body suppliers and counter-response by OEM, aims to reduce tare in favour of increased truck payload. So OEM and distributors occasionally have to supply with, or ready for, a third party designed and supplied body.
- Gravitating to wedge shaped bodies with flat floors as an emerging design standard relatively simple to fabricate and refurbish.
- "Duck tails" – Hays' "upward sloped tail chute" (Hays, 1990, p675) are in the past. Current practice for load retention is to rely on forward slope of the flat floor supplemented by liners in the form of rock-box grillages that retain fine

material so protecting the body floor in the high-wear area. Protection options for high wear areas inside bodies including the "duck tail" are illustrated by Figures 3.30 to 3.33,

- There has been some interest in "limited-life" bodies where cost of early replacement is traded off against reduced on site repairs and increased payload. A brief initiative by OEM and third party body suppliers to produce Ultra Light Weight (ULW) bodies was rewarded with only limited success. This concept of bodies designed for regular total replacement has contracted to applications with relatively low-impact and low-wear materials – where sufficient body life is realised to justify total replacement cost
- ULW bodies for large mining trucks have been tested in iron ore, oil/tar sands and coal overburden applications with little success. Applications for lightweight, highly fragmented and low-abrasion materials have enjoyed some success. Hard-rock, moderate-to-severe, applications have moved on to more robust designs. There is significant local (Western Australia) history of ULW trials with some successes; but also substantial failures – particularly in more severe hardrock applications. At present the "limited-life" philosophy does not appears to have significant or increasing following. It is not clear if the apparent lack of interest is due to misapplication, resistance to change within maintenance management or "great-idea-poor-design-and-execution" syndrome or a combination of these. But there is manifest ground swell of mood for change/reversion to more robust designs.
- Bodies fabricated from aluminum alloy; also rubber-lined with cable-suspended rubber floors have also been field tested, particularly where loading noise from steel bodies is perceived as a problem. These experiments enjoyed only limited success being confined to operations where material hauled is low impact, usually for long hauls and where cost of roads to highway (unsealed) standards (to reduce site-severity vibration effects on light weight bodies) can be justified.

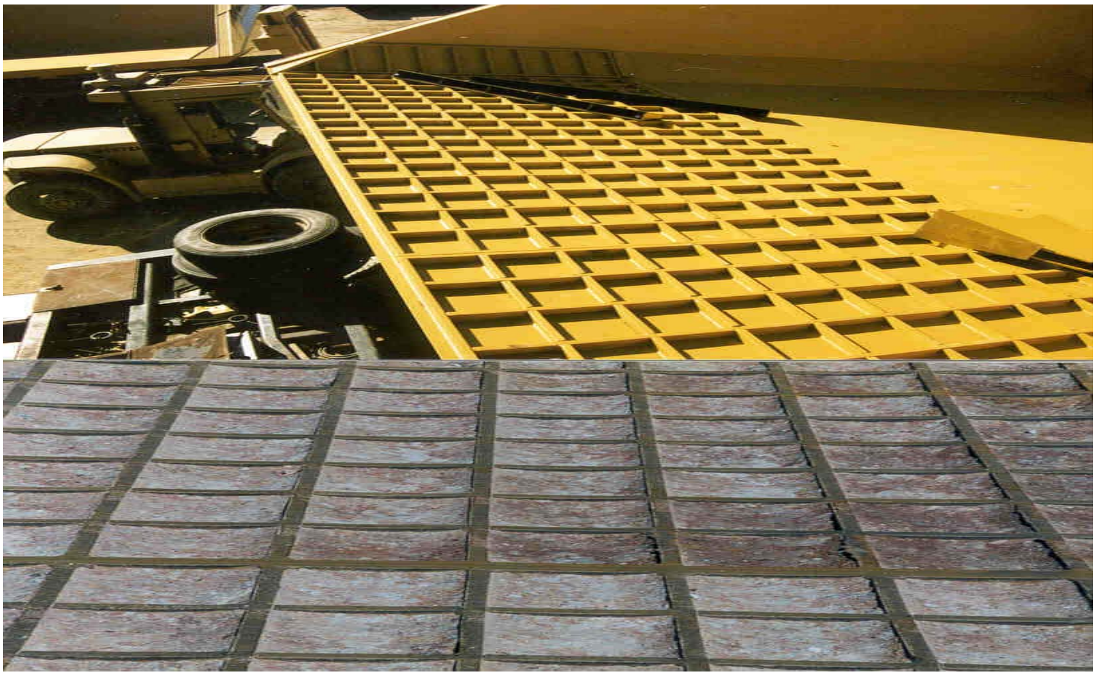
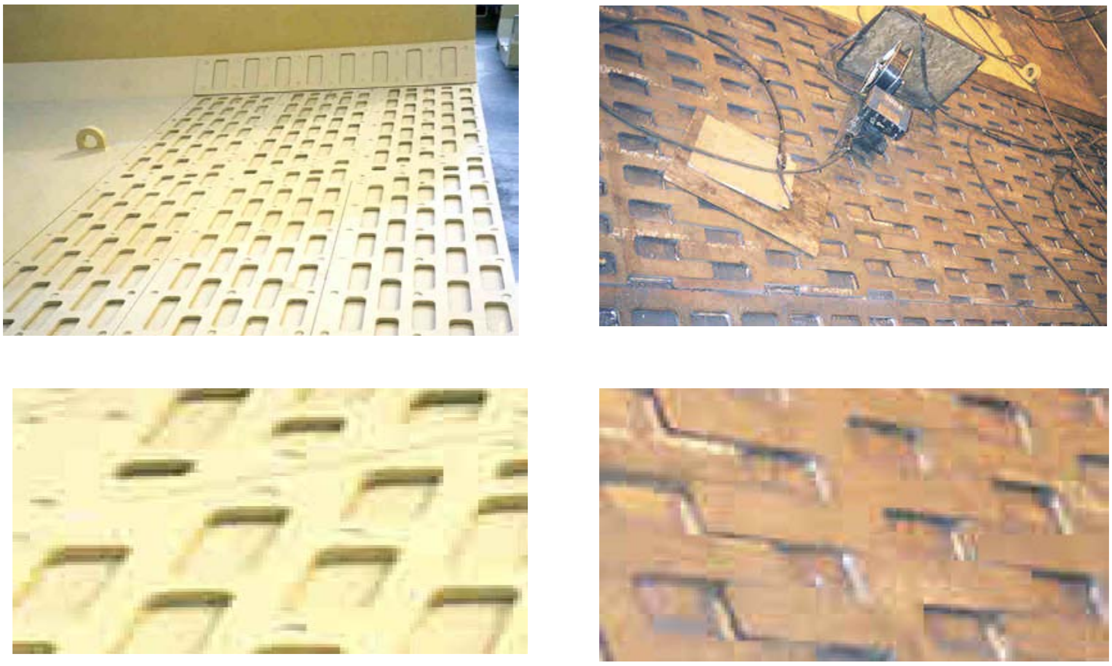


Figure 3.30 Liner Options – Rock Box (Caterpillar Mining Forum, 2003)



Grid Liner **Interlocking Grid Liner**

Figure 3.31 Liner Options – Grid Liners (Caterpillar Mining Forum, 2003)

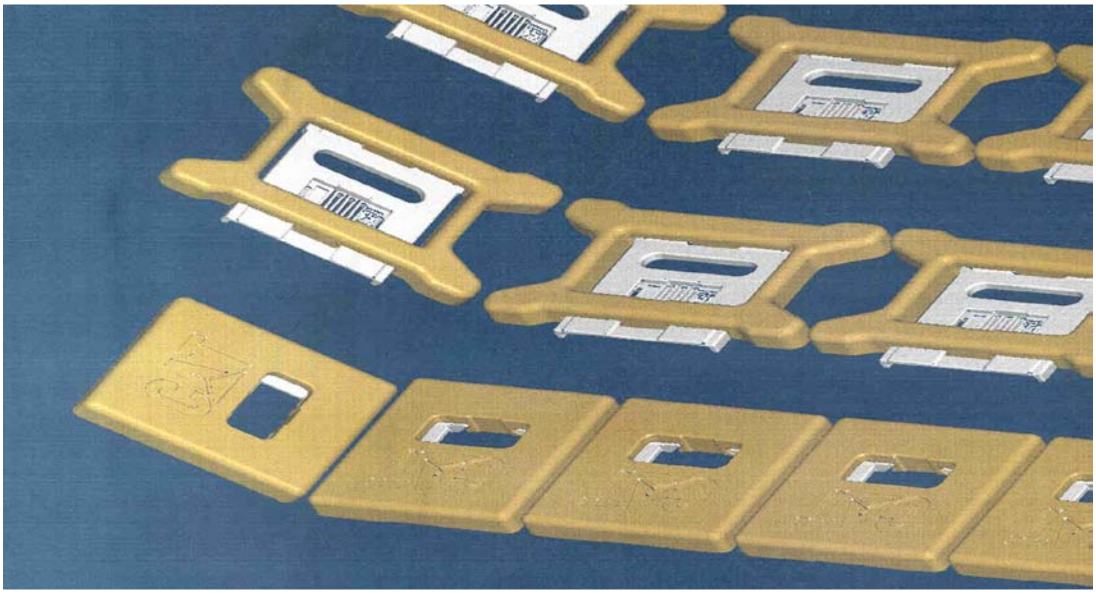


Figure 3.32

Liner Options – Skeletal Mechanically Attached Wear Plate System (MAWPS)

(Caterpillar Mining Forum, 2003)

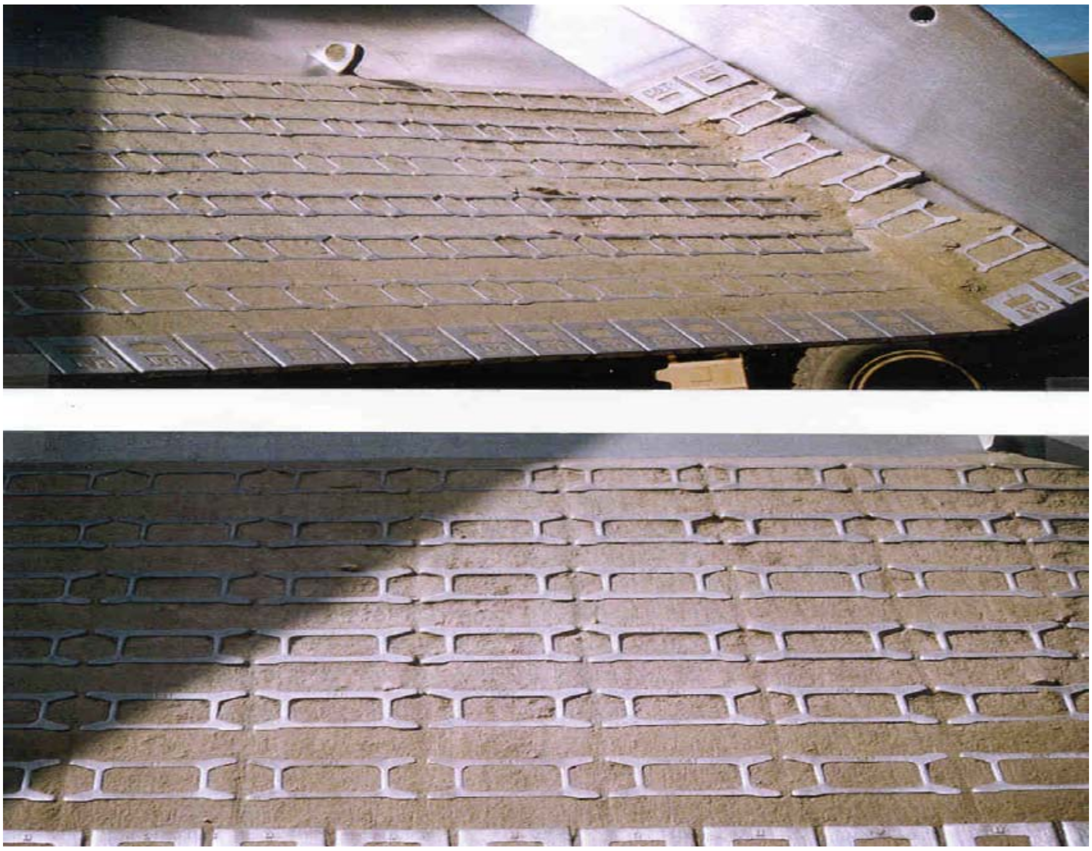


Figure 3.33 Liner Options – MAWPS Installed & Operational

(Caterpillar Mining Forum, 2003)

In the process of truck body selection, issues to be considered to deliver expected performance include:

- Body weight is a relatively small proportion of total tare (typically 25% or more) and an even smaller proportion of GMW – about 10% to 13%. Ratio of body weight to payload is also small – about 20%. Refer to Table 3.35, Volume 2, Appendices, p 438 appended in Volume 2.
- Savings in bodyweight from any of the available initiatives tend to be relatively small compared to the payload benefit realised.
- A change in body weight of some 5% will allow an increase in payload of 1% - using a Cat 793C 240 ton truck as an example – refer to Table 3.35. Total dual slope body weight, including liner kit, is 43.2 tonnes. A reduction of 10% will provide for an increase in target payload from 218.1 tonnes to 222.4 tonnes – an increase of some 2%.
- Accordingly the benefits of reduced bodyweight in favour of increased truck payload will tend to be subtle rather than startling. The additional payload is virtually free carried without additional direct operating cost, so realizes a substantial benefit relative to the margin of earnings less costs – and this is the justification for pursuing reduced bodyweight.
- But apparent productivity and unit cost benefit need to be discounted by any additional downtime and maintenance cost due to shorter body life and early major rebuild or replacement. Nett production and cost benefits can be difficult to quantify at the time of deciding to take advantage of any reduced body weight initiative.

Truck bodies specifically designed for some applications in Western Australia have been observed by the author to attract substantial maintenance after a relatively short life. Reducing body weight by reducing plate thickness and using high strength wear steel as a structural material that is more difficult to weld; and adding wear plate kits for severe service significantly reduces the potential weight saving. Add to this the reduced life that some operations have experienced and the attraction of the concept tends to wane.

In the end result, enhancing payload by reducing truck tare through reduced body weight is likely cost advantageous providing that all maintenance risks are identified, assessed and, as necessary, treated before finalizing the financial justification to proceed.

Body Volume Capacity and Standards

Volumetric capacity of truck bodies has traditionally been based on SAE standards; both struck and heaped 2:1 capacities (Hays, 1990)

OEM specifications for mining trucks generally provide both struck and heaped 2:1 volumes; but, at least, the heaped 2:1 capacity will be provided. The pyramidal projection and sharp peak of this standard model for truck body capacity is artificial and is not realised in practice and can only be considered as an indicative guide. A particular impractical perception is that the rear free face of struck volume is projected back from the discharge edge at 1:1 – an obviously optimistic, impractical model of angle of repose of most mined materials in a truck. The heaped 2:1 model is essentially a pyramidal shaped projection superimposed on the struck volume.

Experience shows that the general angle of repose of most materials mined is steeper than 2:1; but certainly not as steep as 1:1. Actual corners of truck payloads are not sharp but tend to be conical sectors with a rounded truncation of the peak.

Ronald Hays advises: “Experience indicates that actual capacities are about 105% to 130% of SAE (standard) struck volumes with higher percentages valid for largest trucks with wide bodies and top (side rail) extensions.” (Hays, 1990).

Table 3.35, appended in Volume 2, provides a list of Caterpillar mining trucks showing that ratio of OEM-specified 2:1 heaped volume to struck volume varies from 127% to 143% with modal ratios in order of 133%. High values in this range of trucks are for a 90 tonne payload truck that has a relatively wide body compared with both wheelbase and body target length. Low values are for the largest truck currently available with a relatively short target length and close-to-square body target. Mining trucks from other OEM can be expected to exhibit similar relationships between struck and heaped 2:1 rated capacity; also for the other specification items in Table 3.35.

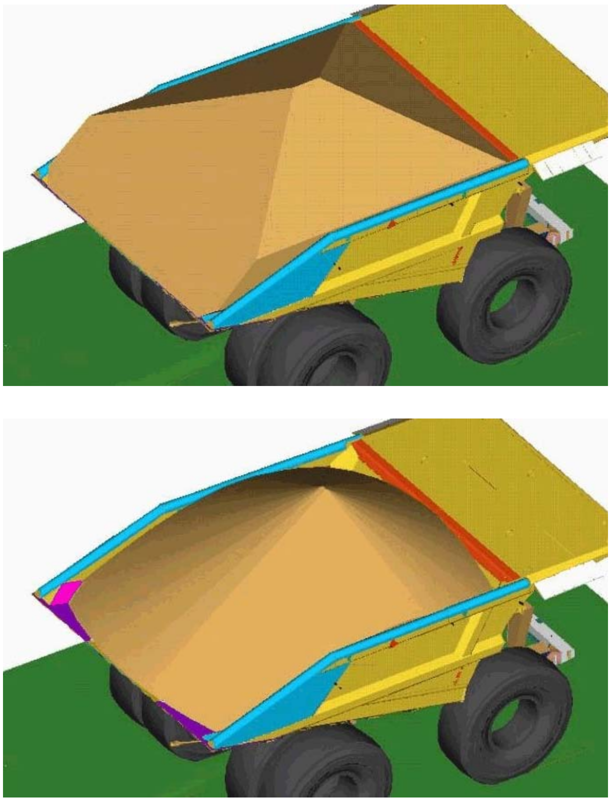
The actual volume accommodated by a truck body does not conform to the hypothetical solid geometrical configuration of SAE heaped 2:1. Natural angle of repose of material as mined in the moist condition is generally in the range 30 to 38 degrees (1.7:1 to 1.3:1). Highly weathered materials dry out and slump to flatter angles with rain, wind and general atmospheric exposure. Loaded materials as dumped in trucks will initially assume natural angles of repose and may slump slightly as the truck travels depending on the vibration from running surfaces, significant for suspension and chassis life; but largely damped by tyres and suspension so that the load generally remains static as placed.

Spillage from trucks is a continuing operational management issue. Main causes are overloaded trucks, presence of large fragments in material to be loaded that sit on the load rather than encapsulated within. Sharp changes in road direction without transition curves or appropriate super elevation, undulations in roads due to uneven compaction (usually a combination of incompetent road formation material and inadequate working and compaction) or surges from transmission shifts may dislodge free particles so they roll off the load. Spillage from loaded mining trucks in transit can be substantially reduced by sound roadwork preparation, and construction; also operating protocols such as prohibiting transmission shifts on ramps.

Development Directions

Body volume design for mining trucks is essentially a compromise. Body capacity determination has to accommodate:

1. Variability in insitu density across the stratigraphy to be mined.
2. Variability in voids ratio typical for geophysical characteristics of each stratigraphic horizon.
3. Variability in voids ratio of material prepared for loading onto trucks within each stratigraphic horizon due to localized material differences and variable drilling and blasting (or other preparation method).
4. Variation in slope angle (of repose) of material in the truck body from the SAE standard 2:1 - Figure 3.35; also for a specific horizon, and further within that horizon.
5. Misplacement of bucket loads in the body resulting in payload variability.
6. Solid geometrical form of the load tending to conical rather than pyramidal form of the SAE 2:1 standard – Figure 3.34.



Diagrams are based on 1.7:1 slopes. Conical payload volume - bottom diagram - is estimated at 87% of SAE 2:1 volume at left.

Figure 3.34 SAE 2:1 Volume v. Conical Volume (Caterpillar Australia)

Indicates that **35 degrees is more realistic slope** for the particular load of material.

Figure 3.35 Illustration of Realistic Slopes -Based on Angle of Repose

(Caterpillar Australia)

Determination of body capacity of mining trucks is based on:

- Insitu density provided by the purchaser/future operator (from a number of apparent rock density determinations).

The following are best performed cooperatively between purchaser and supplier:

- Adoption of a load factor that is a convenient ratio of loose to insitu density that, in application, accommodates all of the factors such as swell, voids ratio, and their variability (Caterpillar, PHB 35).
- A factor allowing for the expected variation in slope angle and solid geometrical form of the payload relative to SAE 2:1 rating – 87% is one estimate – Figure 3.34.
- Mining truck OEM and third party body suppliers are designing bodies using capacity criteria modelled on actual observations of load form. In detail the form of the load model is considered to be conical above the side rail so causing voids at front and rear corners rather than pyramidal as assumed by the SAE heaped 2:1 standard. There is some compensation for the corner voids with adoption of angles of repose steeper than 2:1. Figure 3.34 and Figure 3.35 above, from Caterpillar, illustrate the concept of conical load profile and slopes steeper than 2:1 – 1.7:1 in the concept of Figure 3.34.

Although truck specifications from all OEM include the SAE 2:1 capacity, for axle-load distribution specifications; and for body design, more realistic load profiles have been adopted. Particularly for the recently developed Cat MSDII body; and equivalents from other OEM and third party body suppliers, initiatives have been based on more realistic load modelling.

Bodies for larger trucks now tend to be designed for the mining conditions to be experienced. Caterpillar’s MSD bodies are an example. The type of material to be transported, fresh rock, transition or highly weathered, including abrasion potential are assessed and bodies designed to the specification including expected material density. Bodies so designed are fabricated from hardened and tempered weldable alloy steel. Liner kits are limited to light impact inserts across the front bulkhead and body floor with wear protection at discharge – rock boxes or one of a number of grid liners on offer as illustrated by Figures 3.30 to 3.33.

Reference to Table 3.35, appended Volume 2, indicates that body weight for Caterpillar trucks (including liners) only constitutes 8% to 14% of GMW – and 15% to 20% of truck payload. So a reduction of 10% of bodyweight will realize 1.5% to 2.5% addition to payload.

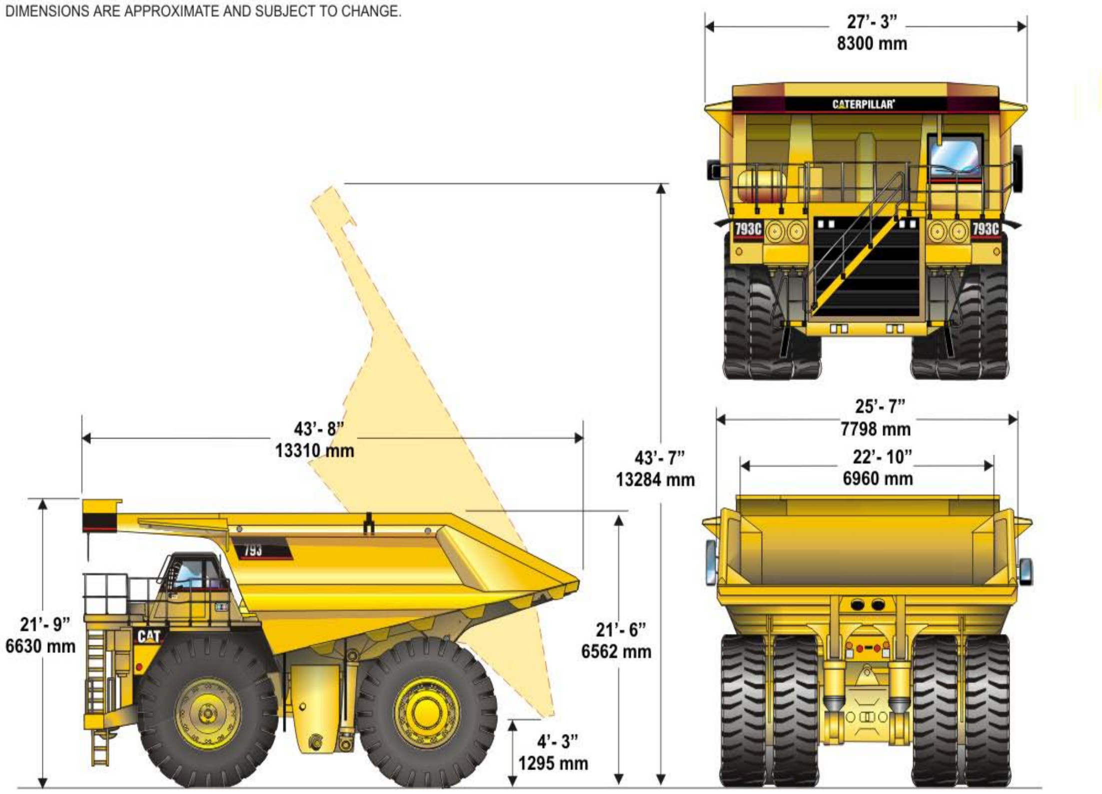
A typical Caterpillar MSD design for a 793C mining truck (nominal 240 tons) is illustrated by Figures 3.3.5.7A and B appended. It will be noted that:

- Compared with the flat floor body weight in Table 3.35, the 793C MSD body realizes weight saving of 8 tonnes i.e. some 20% of body weight, allowing some 3.5% increase in target payload.
- Heaped 2:1 volume rating has increased from 147.7 to 175.8 cubic metres to accommodate relatively low density material at 1.6 tonnes per loose cubic metre.
- Corner fillets (standard for Caterpillar MSD bodies) provided in the particular design and application are intended to inhibit saturated sandstone and mudstone sediments from the partings between multiple coal seams compacting in reentrant corners as “carryback”: (the corner fillets also function structurally to stiffen the corners and avoid high stress concentrations at 90 degree reentrant corners).
- The comparatively light liner plates of 450 Brinell Hardness Number (BHN) weldable steel on the rear floor and sides plus floor impact plates are supplementary wear treatment on a body generally designed to be wear resistant, being fabricated from weldable 400 BHN hardened and tempered alloy steel plate.
- The typical MSD cutaways at the rear of body sides are shown in detail in Figures 3.3.5.7A and 3.3.5.7B. This saves weight of unnecessary body side extensions and discourages overfilling the body to spill over the discharge edge with the risk of further spillage whilst hauling.

CATERPILLAR
MSD
Mine Specific Design

793C MSD

DIMENSIONS ARE APPROXIMATE AND SUBJECT TO CHANGE.



CAPACITY & WEIGHTS

GMW	846,000 Lbs	383,739 Kg
Approximate Chassis	263,500 Lbs	119,524 Kg
Approximate Body	66,630 Lbs	30,223 Kg
Target Payload	258 Tons	234 Tonne
SAE 2:1	230YDS ^{3}	175.85 M ^{3}
Density	2,698 Lbs/LCY	1.6 Tonne/LCM

- **Standard CAT warranty and MSD warranty apply.**
- **Specifications valid until 02-01-03**

- **Empty chassis weights may vary depending upon configuration.**
- **Payload must be adjusted for actual empty chassis and body weight in your application**
- **Because material densities vary widely throughout a mine site it is the responsibility of the mining company to manage the payloads in accordance with Caterpillar 10/10/20 payload policy.**

Figure 3.36 Typical MSD Design 793C – General Arrangement

(Caterpillar Australia**)**

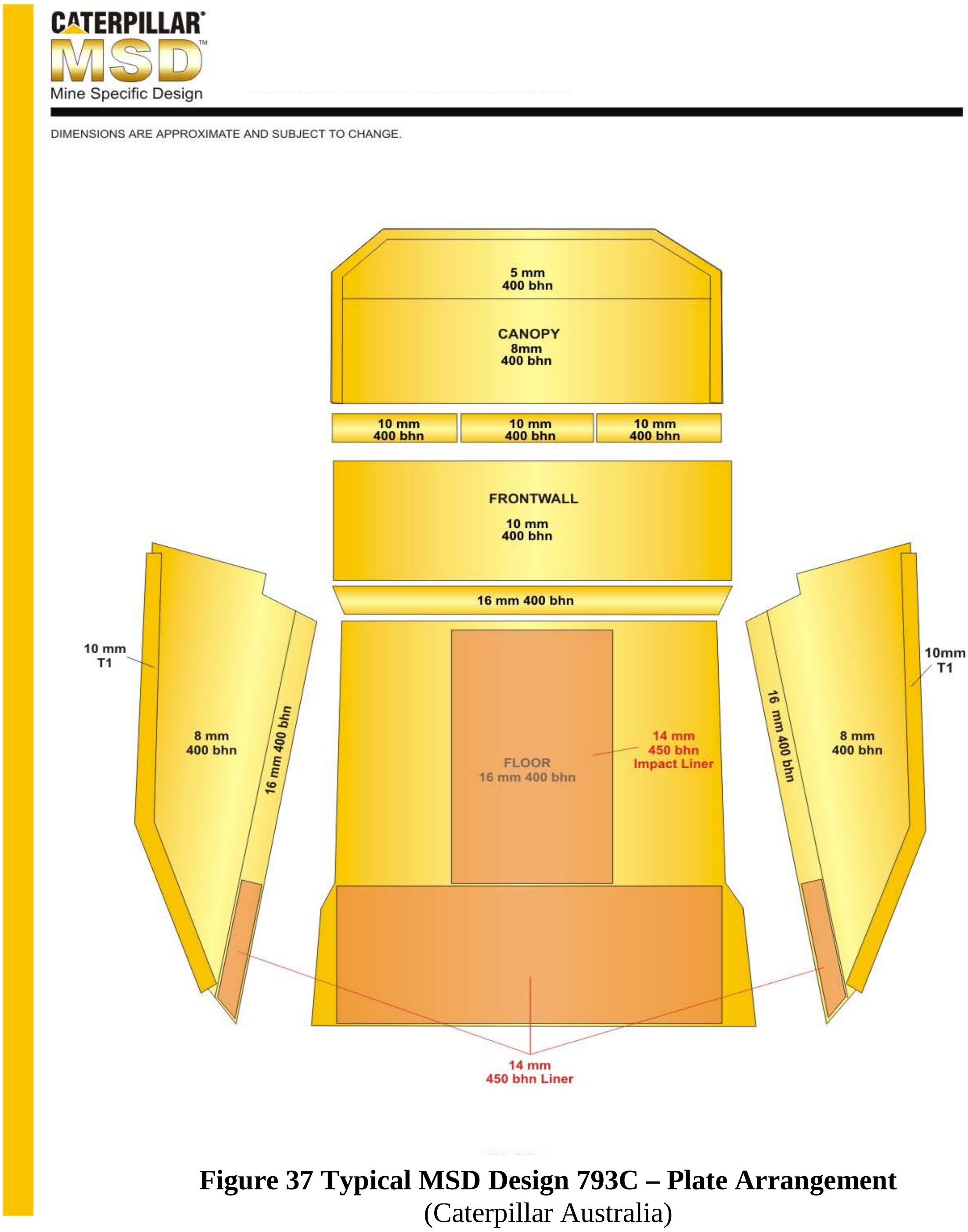


Figure 37 Typical MSD Design 793C – Plate Arrangement
(Caterpillar Australia)

The reduction in MSD body weight of some 8 tonnes effectively enhances target payload by 3.5%, an obviously welcome improvement. But as noted above, the risk of additional costs due to shortened body life and maintenance needs to be considered.

The example described is for moderate to light duty, accommodating low impact materials that are abrasive having high silica content. The reduction of 20% illustrated can be considered as a typical practical limit.

An improvement in the second generation of development in MSD II bodies is a flare-out from front to rear to reduce wear on body sides and bottom corners. For a nominal 220 tonne truck the taper is some 300 mm front to rear. Technically this increases the operating width of the truck; but the overall increase is small – some 150 mm – compared with the nominal specified width of 7.41 metres.

An important consideration in hauling efficiency is carryback of material in the body generally compacted into reentrant corners where sides meet the floor - more importantly in the front corners. Corners fillets can be seen in Figures 3.36 and 3.37 as a treatment to reduce carryback. Early designs of truck bodies included rolled or pressed plate circular quadrant fillets to all body corners. This detail seems to have been abandoned in the interests of construction simplicity and cost. The 450 BHN hardened and tempered plates are likely not amenable to pressing or rolling. But it would possible to make up corner quadrants in mild steel plate and provide wear-resistant liner-plate protection.

Truck Bodies For A Range of Material Density

Open pit developers and mining operators often have to cope with hauling of materials with a range of density. Some typical examples are discussed below.

A. An open pit coal mine of moderate scale where a laterite capping overlays tertiary sediments – transported granite detritals – that overlay sub-bituminous coal measures with sandstone-mudstone partings. Typical density profile is:

	Insitu ARD – tonnes/bank cubic metre
Laterite Capping	Estimated 2.4 - small quantity
Tertiary Sediments	2.0
Shale Partings	1.8
Coal	1.34

B. A deep open pit gold mine with shallow, transported, lateritic gravelly clays ranging in depth from zero to as deep as 30 metres across the pit width overlaying a weathered zone down to 60 metres depth (ranging from 2 metres to 60 metres or more) with a sharp oxidation cutoff, typical of the Achaean greenstone stratigraphy of the Eastern Goldfields of WA. Typical density profile:

	Insitu ARD – tonnes/bank cubic metre
Gravel Clays	1.8 (author's estimate)
Weathered Zone	2.4 (author's estimate)
Fresh Rock Zone	2.8

C. A deep base metal open pit with laterite cover down to some 3 metres over leached out clay weathered zone as deep as 30 metres, followed by a transition zone of some 20 to 30 metres exhibiting the complete range of incompetent clay to fresh rock to an oxidation cutoff above fresh rock. Typical insitu density profile is:

	Insitu ARD – tonnes/bank cubic metre
Laterite	2.4
Weathered Zone	2.2
Transition Zone	2.4
Fresh Rock Zone	2.6

Example A:

Rope shovels load a mixed fleet of mining trucks currently in process of rolling over to a 220 tonne mechanical drive fleet. Coal is hauled in the same trucks used for overburden and interburden, i.e., partings between the seams. Body size is selected for overburden and interburden. Full-volume payloads of coal under-load the trucks. The coal-haul distance from pit exit to crusher is some 5 km of fast, flat road. So the lightly loaded trucks reduce the TKPH (tonnes-kilometres-per-hour) demand on tyres. The operator opts for the flexibility of this arrangement in preference to a dedicated fleet of smaller mining trucks with bodies sized for coal.

It is understood that new mining trucks are sized for the sandstone/mudstone partings that are of similar loose density to the tertiary sands.

Example B:

Current open pit depth is more than 300 metres with an ultimate depth exceeding 500 metres. As the pit deepens, the proportion of tertiary detritals and weathered zone becomes less as the pit walls are pushed back.

In this example truck bodies are selected for the estimated loose density derived from estimated swell (voids ratio) and an insitu density of 2.88. Any inefficiency due to transporting lighter materials that are relatively small quantities is absorbed in mining costs – assumed to be a studied compromise.

It should be noted that historically detritals and weathered zone were a more significant proportion of total production. This was a period when the current open pit was worked as several individual relatively shallow open pits along a strike of several kilometres. As the pits have coalesced into one large pit, the proportion of lower-density material has reduced significantly.

Example C:

Current pit depth is some 200 metres with a final design depth of some 400 metres. A compromise similar to Example B above has been made in this case.

In this particular case the open pit was commenced with a substantial pre-strip of weathered and transition zones to the base of transition and into the fresh rock zone hosting the ore targets to expose and commence mining of first ore. As the open pit has developed, the proportions of lower density weathered and transition materials have diminished – as with Example B. For this operation truck body capacity has been selected for the fresh rock zone with a density of 2.6 tonnes per BCM and a loose density assessed as 1.7 tonnes per LCM.

In each of the above cases truck body capacity is essentially a compromise. The options are:

1. Select bodies to carry optimum truck payload with the material of expected lowest loose density and control potential truck overload by monitoring indicated truck payload with a load measuring system such as Caterpillar's VIMS or equivalent with indicator light or digital readout – and, in default, by limiting the number of bucket loads in each payload.
2. Select bodies to accommodate optimum truck payload of material density corresponding to the maximum quantity in the suite of differing density materials to be hauled. For higher density materials truck payloads can be controlled as described in option 1 above. For lower density materials, any inefficiency caused by under loading trucks is a compromise to be accepted and absorbed; but there is always the option to fit side rail extensions

(“hungry boards”) temporarily or permanently provided the quantity of lower density material justifies the cost of body-capacity enhancement.

Each case of multiple material densities needs to be considered on its merits; and comparative unit costs compiled for loading and hauling in each case. As with many cost comparisons in the process of equipment selection; or operating open pit mines, particularly unit cost differences tend to be subtle. But for this particular case, the difference between comparative costs can be considered a valid basis for decision if those costs are prepared diligently and with the necessary precision to suit any comparison that may be made.

It should be noted that truck fleet additions and replacements are an opportunity to re-visit truck-body capacity and body design in general. This is a typical due-diligence opportunity that should not be overlooked in the process of acquiring new haulage equipment.

As a last point:

It is manifestly important to ensure that truck-body capacity should be comfortably sufficient to accommodate optimum mean truck payload. It is a far more comfortable management priority to control mean truck payloads down to target payload than to suffer the inefficiency and cost of under loaded trucks with the fallback remedy of adding extensions (“hungry boards”) to body side rails to increase truck body capacity.

Interpretation and Comments

Recent efforts by OEM and third party body designer-suppliers to increase payloads and so improve payload **:** GMW ratio by refined body designs have been successful but necessarily limited in effect. Selecting body designs resulting from these initiatives necessarily has to be justified on the basis of productivity and cost analysis. Any initiatives by OEM and third-party suppliers to improve body designs, reduce body weights and improve payloads are obviously potentially beneficial in terms of productivity. But the outcomes can be expected to the useful rather than spectacular. Although possibly of moderate rather than spectacular value, Caterpillar’s MSD truck bodies or equivalents from other OEM are likely justifiable, even attractive to open pit mining operators, on the basis that small contributions accumulate to significant improvement – a fundamental continuous improvement philosophy.

Compared with refined body design, there is greater potential for productivity and cost improvements by moving on from the time-honoured 2-4 wheel mining truck configuration with payload **:** GMW ratios in the range 0.55 to 0.60. Recent industrial advances in this direction were briefly discussed in Section 3.3.3. It can be expected that initiatives will surface in the future to substantially improve payload **:** GMW ratio for mining trucks by creative designs and non-traditional configurations.

3.3.6 Truck Payload Measuring Initiatives

Preliminary Discussion – History and Current Status

Discussion in Section 3.3.5 on body selection design describes questionably practical body capacity specification standards and recent efforts to develop bodies with capacity based on more realistic volumetric models.

Payload weight measuring developments have been a continuing focus of OEM, tyre suppliers and equipment operators. Intrinsic truck performance is a function of GMW (tare plus payload weight) and NMW - tare weight alone. Early payload measuring consisted recording a relatively small sample of payloads effected by weighing trucks unloaded (tare/NMW) and loaded (GMW). Mining trucks were (and, for calibration purposes, still are) weighed wheel-by-wheel using a transportable platform weighbridge temporarily installed at road level. Payload is derived by difference between GMW and NMW.

Payload control has historically relied on operators sustaining consistent, repeatable volumetric loading practice visually with occasional weighbridge verification of payload actually achieved.

On-board Payload Measuring

More recent payload measuring developments utilise sensing of differences in suspension cylinder pressures for loaded and unloaded trucks to determine NMW and GMW. Initially systems were prone to significant error due to derivations based on single, static readings of suspension pressures from stationary trucks. Caterpillar’s Truck Payload Measuring System (TPMS) is an example still offered; but in much improved form, for smaller trucks where the complete VIMS system is not justified. Similar systems are offered by all truck OEM.

More recent improvements include Caterpillar encapsulating TPMS within the VIMS system. Still based on sensing suspension cylinder pressures, the updated system samples a number of pressure readings and derives mean values to improve accuracy. Static readings are taken of incremental payload additions by bucket load after the truck becomes stationary. Readings and resulting payload calculations are subject to errors as described in Section 3.2.8 and as further discussed in Section 3.3.7.

On departure fully loaded, VIMS senses when the truck shifts to second range on the level pit floor, samples suspension cylinder pressures and derives mean GMW. Payload is determined by difference using a pre-calibrated NMW. Limited self-calibration facility and other calibration issues were described in Section 3.2.8. Caterpillar claims accuracy of +/- 5% for 95% of loads measured. Systems provided by other OEM can be expected to provide similar facilities with equivalent accuracy.

Partial payload measuring facility and convenient data transfer capability were both described in Section 3.2.8.

Historical payload measurement efforts have not considered measurement of loose volume of each payload. Influence of material fragmentation on loose density and consequent effect of variability on loading equipment performance were introduced in Sections 3.2.4 and 3.2.8. Actual bank volume per truck payload can be derived from open pit volume surveys and truckload numbers from production records.

Volumetric Measurement of Payloads

Transcale, in collaboration with CSIRO, have developed a system, *Trayscan,* to measure truck payload volume on the fly. The system consists of two scanning lasers mounted on a boom extending from the platform of a scissors-lift vehicle that can be moved around the site and set up alongside a haul road. As trucks pass under the lasers a 3D profile is built up from processed data.

The output provides:

- Volume of truck payload by difference and estimates of body utilisation.
- Load distribution, enabling determination of CG of the payload.
- Angle of repose of materials in trucks.

- Fragmentation can be characterized from texture of the 3D surface and calibrated with other particle size measuring systems to reconcile with drilling and blasting practice. (Valmadre A, Pers. Comm., 2005)

Transcale have also developed a mining truck weighbridge system with capacity up to 700 tonnes (equivalent to 400 tonnes of payload). Using *Trayscan* and *Transcale* facilities in combination will facilitate determination of loose density of payload in the truck. Currently accuracy of field weighbridges is understood by the author to be in the order of +/- 1% or less.

Accuracy of *Trayscan* indicated in promotional material to be from field tests is +1.4% of published calculated struck volume. A test of *Trayscan*-determined actual, fully-loaded average volume indicated 1.6% above published struck volume and only –0.4% less than published heaped 2:1 capacity for the 91 cubic metre bodies of the nominal 160 tonne mining trucks measured (Valmadre A, Pers. Comm., 2005).

This last volume comparison is interesting in that OEM and third party body manufacturers promote the concept that the 2:1 heaped standard is artificial and not representative of the actual form and volume of a full body payload. From the results claimed by *Trayscan* it seems that, notwithstanding the conservative 2:1 slopes assumed by the heaped SAE 2:1 standard, the actual steeper slopes in the range 1.7:1 to 1.5:1 tend to compensate for the actual proportion of the payload over struck being conical rather than the pyramidal shape assumed by the standard. There is room for further research of this issue.

Efficiency of loading operators in filling trucks also needs to be considered.

Loose Density –Discussion

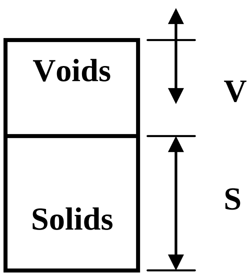
Estimation of loose density in material piles or trucks has generally been based on the ratio of BCM **:** truck payload volume of standard 2:1 heaped capacity – selected load factor multiplied by insitu density and adjusted on the basis of experience. Loose densities so derived are of questionable accuracy but, historically, have been accepted as reasonably comparative.

Mean value for loose density, with reasonable degree of accuracy, can only be measured indirectly. Volumes are much more convenient to measure using surveying techniques that have become sophisticated and very responsive so operators generally measure production and report in volume units. As shown below, loose density that translates as payload in truck bodies is directly related to voids ratio and insitu density of material in the face prior to drilling and blasting or other pre-loading preparation. Voids ratio can be derived from before-and-after volume measurements of material insitu and prepared for loading. Insitu density can be determined from samples of stratigraphy to be mined in the course of exploration drilling and sampling extended to geotechnical testing. Subsequent derivation of loose density of material prepared for loading is a simple calculation. Loose density in the truck body can be derived from volume assessment in the truck body or application of Trayscan as described above; and onboard payload weighing or by weighbridge determination.

Parameter Definitions and Relationships

Voids Ratio (VR):- is the ratio between voids volume and solids volume

VR = V/S as per sketch below (Harr, 1977, p23)



Swell (SW):- is the increase in volume relative to insitu volume – usually expressed as % (Cat PHB 35, 2004, p22-2) – initially discussed in Section 3.2.4..

The following analysis is based on a nominal 220 tonne capacity mining truck. But the outcomes are generic. The analysis also assumes that for practical purposes truck bodies are filled to design levels in modified conical form – subject to a fill efficiency factor. The fill factor allows for departures from assumed standard pyramidal form of truck payloads and natural repose slope. Application of the fill factor needs to be carefully considered to avoid double covering of departures of loose density and voids ratio from expected values.

Interpretations, inferences and conclusions for dispersion of payload distribution can be confidently applied to any mining truck subject to acceptance, or verification, of the basic assumptions of insitu density, expected values for, and distribution characteristics of, voids ratio. Insitu Density considerations are for solid rock that requires preparation for loading and hauling by drilling and blasting or ripping and pushing. Variability of Insitu Density of solid rock is generally small with CV in the range 2% to 4% (Harr, 1977). Lower values in the CV range are applicable for higher density primary and metamorphosed rock types generally unaffected by weathering. For the purposes of the following analysis mean values have been adopted for insitu density. Variability of insitu density and affect on the analytical outcomes has been taken as small and ignored.

1 Mean Load Factor (LF_{MEAN}) is the ratio between mean loose and mean insitu density (Cat PHB 35, 2004, p22-2).

$$\mathbf{LF_{MEAN} = 100\%/(100\% + SW\%)}$$

LF_{MEAN} is a convenient parameter for estimating purposes. It is derived from mean voids ratio as shown in Item 3 below.

The range of load factors assumed for analysis ranges for granite is 0.61 (Cat PHB 35, 2004) increasing through 0.67 to 0.75 as efficacy of fragmentation and volume increase by drill and blast or other means falls away from optimum.

2 Mean Loose Density (LD_{MEAN}) = Insitu Density (ID) x LF.

$$\mathbf{LD_{MEAN} = LF \times ID}$$

ID ≡ Specific Gravity (SG) for practical purposes of the current analysis – see Item 4 below.

3 Mean Voids Ratio – deduced from Load Factor as follows:

SW ≡ VR in same units – proportional or %.

$\mathbf{LF = 1/(1 + SW)}$ (Caterpillar PHB 35, 2004)

$\mathbf{= 1/(1 + VR_{MEAN})}$ the fundamental determination of Load Factor.

$$\mathbf{VR_{MEAN} = (1/LF) - 1}$$

4 Voids Ratio Coefficient of Variation (CV_{VR}) – assumed range based on:

Insitu Density (ID ≡ Specific Gravity [SG] for practical purposes) of low plasticity (primary or metamorphosed fresh rock) rocks has a CV in order of 0.02 (Harr, 1977, p369)

Review of CV_{VR} determined for a selection of particulate materials by others (Table 10-1, Harr, 1977, p369) and intuitive understanding for hard rock prepared for hauling that a practical CV_{VR} range is 0.05 to 0.15 with hypothetical CV_{VR} values, 0.0 and 0.20, added to extend the trend.

5 Distribution of Voids Ratio assumed to be normal. Where expected (mean) value of the VR distribution is 1, $CV_{VR} = \sigma$. Z for the model normal probability distribution = 3.08 for a one-tailed probability of 0.999. That is, any empirical VR $\leq (1 + 3.08CV_{VR})$. That is:

$$\mathbf{VR_{MAX} = (1 + 3.08CV_{VR}) \times VR_{MEAN}}$$

Devore provides probabilities for given Z values (Devore J, 1999, Table A3 inside cover)

6 Normal distribution symmetry is assumed to apply. Accordingly, for a one-tailed probability of 0.999 any empirical VR $\geq (1 - 3.08CV_{VR})$. That is:

$$\mathbf{VR_{MIN} = (1 - 3.08CV_{VR}) \times VR_{MEAN}}$$

7 Estimated Maximum Loose Density (LD_{MAX}) derived from VR_{MIN}, and Insitu Density (ID):

$$\mathbf{LD_{MAX} = ID \times (1/(1 + VR_{MIN}))}$$

8 Expected Value (mean) of Loose Density LD_{MEAN} for Selected Load Factor

$\mathbf{LD_{MEAN} = LF \times ID}$ Repeat of (2) above.

9 Estimated Minimum Loose Density (LD) derived from VR_{MAX}, and Insitu Density (ID):

$$LD_{MIN} = ID \times (1/(1 + VR_{MAX}))$$

10 Theoretical Body Capacity (BC_T) At Selected Fill Efficiency – derived from:

Selected Target Payload (PL_{MEAN})

Selected Fill Efficiency ≤ 1 (η_F)

Expected (Mean) Loose Density LD_{MEAN}

$$BC_T = PL_{MEAN}/(LD_{MEAN} \times \eta_F)$$

11 Estimated SAE 2:1 Body Capacity (BC_{SAE}) – At Selected Body Factor:

Selected Body Form Factor (BFF) to revert from conical solid load and realistic repose slope to pyramidal SAE form and 2:1 slopes BFF ≥ 1

$$BC_{SAE} = BC_T \times BFF$$

12 Theoretical Payload (Maximum) at Minimum of Voids Ratio Range – For Target Body Capacity at Selected Load Factor, Insitu Density and Selected Fill Factor – For the Range of Voids Ratio CV.

$$PL_{MAX} = BC_T \times LD_{MAX} \times \eta_F$$

13 Mean Payload at Mean Voids Ratio – For Target Body Capacity at Selected Load Factor, Insitu Density, and Selected Fill Factor – For the Range of Voids Ratio CV.

$$PL_{MEAN} = BC_T \times LD_{MEAN} \times \eta_F$$

14 Theoretical Payload (Minimum) at Maximum of Voids Ratio Range– For Target Body Capacity at Selected Load Factor, Insitu Density and Selected Fill Factor – For the Range of Voids Ratio CV.

$\mathbf{PL_{MIN} = BC_T \times LD_{MIN} \times \eta_F}$

15 Maximum Loose Density Proportion of Mean %

A percentage index applicable to any mining truck subject to analysis and premises stated above.

$\mathbf{LD_{MAX}\% = LD_{MAX} \times 100/LD_{MEAN}}$

As body capacity is a designed constant, the above equation is equivalent to the ratio of payload maximum to payload mean expressed as %.

16 Theoretical % Overload (OL_T) at Voids Ratio Minimum – At Selected Fill Factor

$\mathbf{OL_T\% = (PL_{MAX} \times 100/ PL_{MEAN}) -100}$

17 Minimum Loose Density Proportion of Mean %

A percentage index applicable to any mining truck subject to analysis and premises stated above.

$\mathbf{LD_{MIN}\% = LD_{MIN} \times 100/LD_{MEAN}}$

18 Derived Truck Payload Coefficient of Variation (CV_{PL}) is, within the constraints of this analysis, directly related to variability of Loose Density of material in the truck body. Assuming a probability of 0.999 and the Z statistic = 3.08 – as described in Item 5 – a reasonable approximation for the truck payload CV_{PL} can be derived as follows:

$\mathbf{CV_{PL} = (PL_{MAX} - PL_{MEAN})/(PL_{MEAN} \times 3.08)}$

PL_{MAX} and PL_{MEAN} are derived as shown in Items 12 and 13. Manifestly CV_{PL} is independent of truck body capacity or truck fill factor.

19 CV_{PL} as derived in Item 18 are for the selected range of CV_{VR}. Limiting values of CV_{PL} to comply with Caterpillar's "10/10/20 Policy" are identified by notation with applicable probability. The 10/10/20 Policy is taken as two Rules for purposes of analysis, viz., "10:10" and "20" Rules. Justification for this separation is discussed in Comments at the end of Section 3.2.8.

Voids Ratio, Loose Density and Truck Body Capacity

Paragraph Item numbers 1 to 19 above cross-reference to numbered columns in Tables 3.36, 3.37 and 3.38 appended in Volume 2.

The relationships in Items 1 to 19 are embodied in the three tables. Table 3.36, Volume 2, Appendices, works through the relationships from load factor through voids ratio to loose density to truck body capacity and payload distribution. Loose Density versus Voids Ratio Coefficient of Variation (CV_{VR}) is illustrated by Figure 3.38 below.

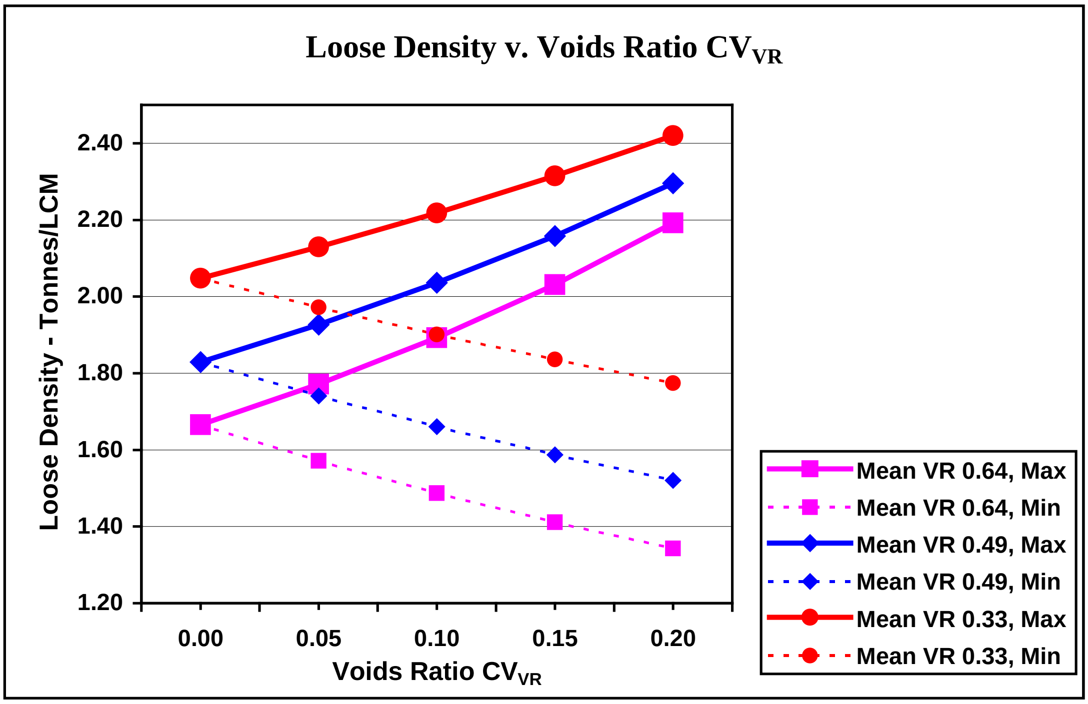


Figure 3.38 From Table 3.36

Selection of the range of voids ratio CV_{VR} for illustration of trends is discussed in Item 4. On the basis of reference evidence, (Harr, 1977), CV_{VR} for particulate rock materials, such as sands, can be as high as 20%. For the purposes of discussion it is supposed that CV_{VR} reduces from a practical upper limit for blasted rock materials of 0.15 tending towards 0.05 for the higher load factor with an ultimate lower limit of zero for solid-rock stratigraphy.

Loose density ranges for the three VR_{MEAN} consistent with the selected range of LF as described in Item 1 – illustrated by Figure 3.38, are theoretical. Practical truncations result from applying the supposed practical limits of CV_{VR} for each VR_{MEAN} related to the selected LF range. The result of these truncations, shown in Figure 3.39 below, reflect reducing CV_{VR} as VR_{MEAN} decreases towards a theoretical limit of zero for solid material. The truncations are indicated in all figures to follow as "dashed" trend lines.

Shaded rows in Tables 3.36, 3.37 and 3.38 reflect the supposed practical truncations of CV_{VR} as discussed above.

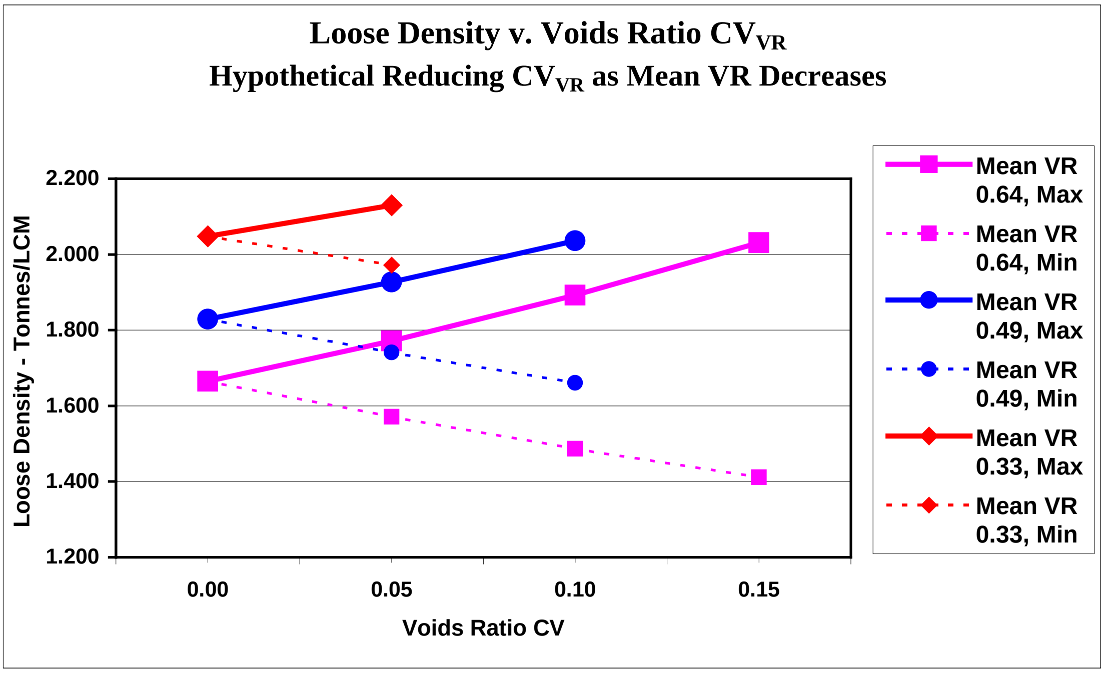


Figure 3.39 From Table 3.36

Figure 3.39 implies that, as efficacy of drill and blast preparation of material reduces, mean voids ratio will decrease and loose density will increase. The range of applicable CV_{VR}, in accordance with the relevant supposition above, is also expected to decrease so limiting the loose density range.

Table 3.36, appended in Volume 2, and Figure 3.39, previous page, show that:

- For higher VR_{MEAN} the wider practical range of CV_{VR} results in relatively wide variation in LD.
- The wide variation in LD flows on as a substantial variation in truck payload.
- As VR_{MEAN} increases, LF (applicable to either bank volume or tonnes) naturally decreases – truck payload tends to decrease so justifying an increase in truck body capacity – as shown in Tables 3.36 and 3.37.
- Increasing trend in load factor/loose density resulting from reduced VR_{MEAN} might be seen as a convenient remedy for the undesirable tendency of increased VR_{MEAN} to reduce truck payload as indicated; but, as discussed below, decreased VR_{MEAN} has significant negative effect on loading productivity.
- Experience evidences that reduced VR_{MEAN} results in reduced digability; so loading equipment will experience slow and inefficient bucket filling and consequent reduced productivity.

Digability is discussed in context with VR and productivity in some detail below.

Payload CV v. Voids Ratio CV

Table 3.37, Volume 2, Appendices is an extract from Table 3.36 to facilitate Figure 3.40. Overlaid on Figure 3.40 and figures to follow are practical limits to each of the variables compared with CV_{VR} for compliance with Caterpillar's "10:10:20 Policy".

For convenience of the analysis the "10/10/20 Policy" is considered as two separate protocols. Each of these protocols – "10:10 Rule" and "20 Rule" – have different functions. The "10:10 Rule" relates to adverse outcomes for component life and operating cost resulting from breach of overload tolerance. The "20 Rule" relates to absolute loading limit for preservation of steering and braking safety. The two "Rules" also relate to different levels of truck payload coefficient of variation (CV_{PL}), viz.,

- "10:10 Rule" compliance that not more than 10% of payloads exceed 10% over target (probability 0.90 that payloads are≤ 10% over target) is achieved by limiting CV_{PL} to 0.078.
- "20 Rule" compliance that no single load should exceed 20% over target payload (assumed probability 0.999 that all truck payloads≤ 20% over target) is achieved by limiting CV_{PL} to 0.065.

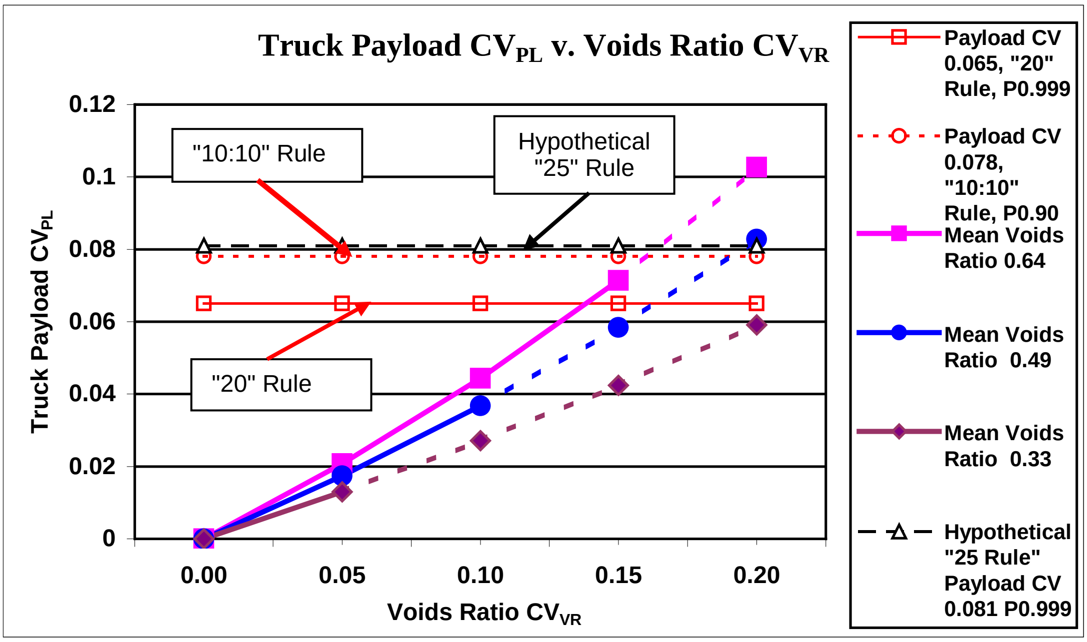


Figure 3.40 From Table 3.37

Figure 3.40, facilitated by Table 3.37, compares truck variability in terms of CV_{PL} with CV_{VR}. The relationship depicted by Figure 3.40 is considered typical rather than specific as the trend lines reflect suppositions and assumptions basic to the analysis. The actual relationship may differ and needs to be determined by empirical sampling of relevant data to provide descriptive statistics, including CV, for each variable. Also, CV_{PL} may reflect other influences such as:

- Bucket (fill) factor that is a function of:
  - Digability (see discussion below) that in turn is a function of:
  - VR_{MEAN} and CV_{VR}; and
  - Operator inefficiency and skill deficiency.
- Number of bucket passes as discussed in Sections 3.2.8.and 3.2.9 – a combined effect of bucket capacity selection and bucket fill factor.

From Figure 3.40, theoretically, as VR_{MEAN} decreases, LD_{MEAN} increases. As a result of the supposed truncation limits, CV_{VR} reduces. As a further result CV_{PL} reduces significantly so that, in the circumstances, compliance with the 10/10/20 Policy appears to be a non-issue. But, as a reality check, the above discussion on digability and VR_{MEAN} should be noted.

At optimally high VR_{MEAN}, where digability is acceptable, the inference of Figure 3.40 is that CV_{VR} increases. In turn CV_{PL} increases so compliance with the “10/10/20 Policy” does become an issue. From Figure 3.40, and subject to all underlying assumptions and suppositions, it appears that, for VR_{MEAN} 0.64, compliance with the “10:10 Rule” only becomes a problem when $CV_{VR} \geq 0.16$; but non-compliance with the “20 Rule” will be likely for $CV_{VR} \geq$. 0.14.

The hypothetical concept of a “25 Rule” is also introduced as an overlay on Figure 3.40. The “25 Rule” must remain hypothetical until such time as the mining truck market and OEM can justify consideration of, and, if necessary, design for, higher standards of steering and braking capability.

Clearly the “25 Rule” is a hypothetical concept of the author only based on the mathematical relationships of payload distributions. It has no connection with the Caterpillar 10:10:20 Policy described herein. Neither has it been discussed with Caterpillar or any other OEM. The 10:10:20 Policy addresses for Caterpillar trucks maintaining steering and brake certification to SAE and ISO standards and British Columbia Codes. Understood by the author to be the first of its kind, the “Policy” has served, and continues to serve, the industry well in maintaining safe load and haul operations.

OEM other than Caterpillar can be expected to adopt the same design standards for equipment manufacture; and, therefore, should require similar limits for payload and GMW.

The hypothetical “25 Rule” – where, with a probability of 0.999, no single payload should be greater than 25% overload - corresponds to a CV_{PL} of 0.081. In such case the “10:10 Rule” and “25 Rule” would be reasonably consistent standards of limiting CV_{PL}. Given a “25 Rule” the “10:10 Rule” would control the CV_{PL} upper limit to 0.078. In such circumstances compliance with the “10:10 Rule”, at the chosen confidence level of 1:1,000, would ensure compliance with the “25 Rule”. This is in contrast with the current situation where compliance with the "20 Rule" at $CV_{PL} \leq 0.065$ ensures compliance with the "10:10 Rule".

Compliance with the "10:10 Rule" is generally tested by compiling truck payloads, i.e., empirical data from VIMS, or an equivalent, and/or truck weighing. Payloads > 110% of target payload are counted and expressed as % of total payloads. Collecting empirical truck payload data as an accumulative running sample size of, say, 1,000 payloads and deriving descriptive statistics to test $CV_{PL} \leq 0.078$ can achieve the same result.

From Tables 3.36.and 3.37, each of the three selected VR_{MEAN} (as a derivative of LF_{MEAN}) the calculations yielded a specific body capacity. This is a somewhat artificial concept. It will also be noted that calculations from VR_{MEAN} through to LD_{MEAN} are independent of body capacity. So Figures 3.38 and 3.39 are valid for any designed body capacity.

Comparison of CV_{PL} and CV_{VR} in Figure 3.40 is also independent of body capacity and truck filling efficiency factors as noted in Item 18.

Voids Ratio CV v. Maximum Truck Payload

To facilitate Figures 3.41 and 3.4, Table 3.38, Volume 2, Appendices, was compiled by modifying Tables 3.36 and 3.37 both appended in Volume 2, to a common truck body capacity based on the highest VR_{MEAN} in the range adopted for analysis. Basing analysis on maximum truck payloads and potential truck overload on a common truck body capacity provides more realistic comparative results.

Figure 3.41 relates PL_{MAX} to CV_{VR} over the selected VR_{MEAN} range. This relationship is a simple extension of the CV_{PL} v. CV_{VR} relationship of Figure 3.40. In earlier Section 3.2.8 the connection between truck payload and CV_{PL} was discussed in some detail.

It will be noted that:

- The equivalent potential limit of excess over target payload, the "10:10 Rule", is overlaid as part of the "Rules suite"; but has no significant relevance in the context of Figure 3.41.
- The "20 Rule" and hypothetical "25 Rule" are superimposed to indicate absolute compliance limits.

- Target Payload is superimposed as a bench mark for comparison.
- As VR_{MEAN} decreases Theoretical Maximum Truck Payload (TMTPL) can be expected to increase at any CV_{VR}; but the practical range of CV_{VR} tends to decrease (in accordance with the truncation supposition introduced above).

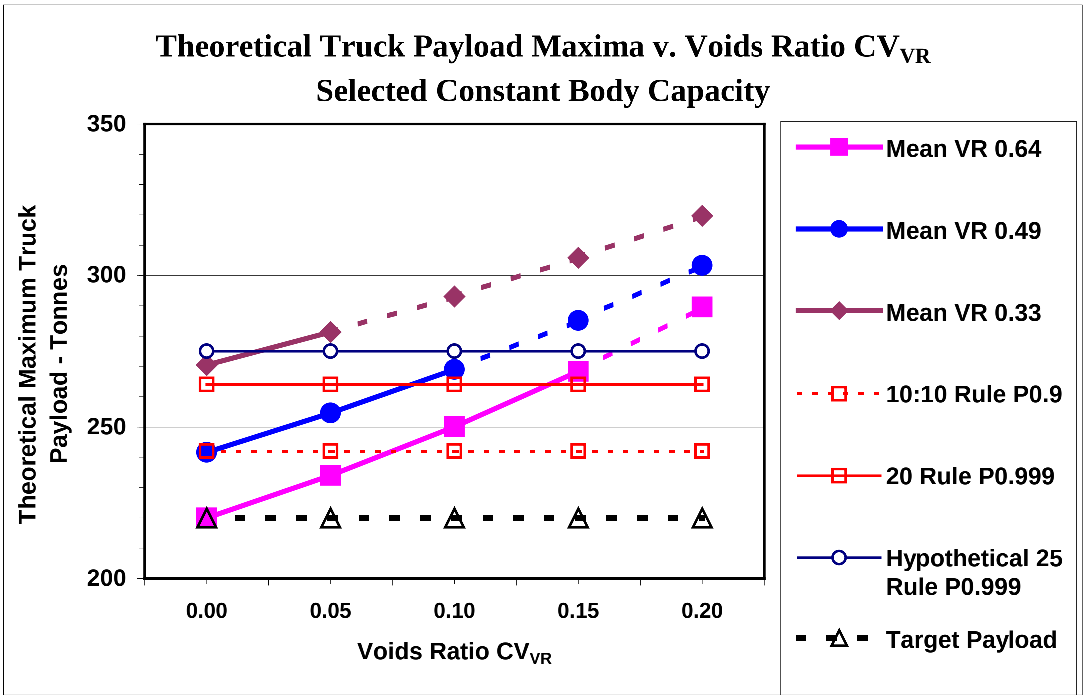


Figure 3.41 From Table 3.38

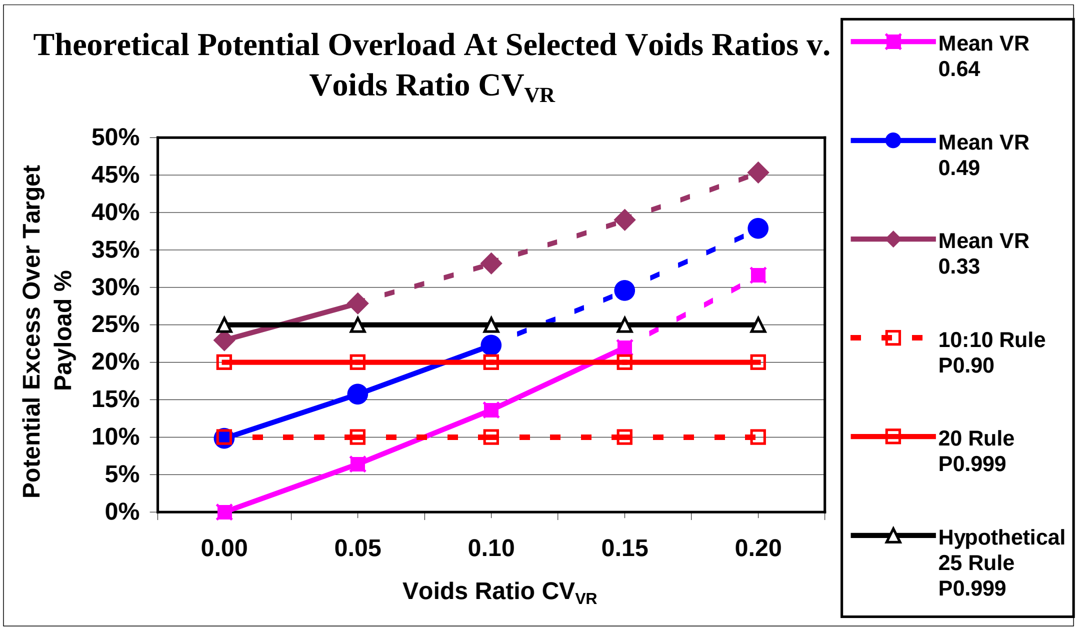


Figure 3.42 From Table 3.38

As shown by Figure 3.41 the TMTPL below the practical truncations of CV_{VR} are within reach of compliance with the "20 Rule" (and would generally comply with the hypothetical "25 Rule"); but compliance with the "10:10 Rule" tends to become more difficult, if not impossible.

The conclusion implies that, for any given truck body capacity, the potential to overload increases with reducing VR – as expected intuitively. Fortunately offsetting this to a partial, if not total, effect is the expected reduction in CV_{VR} as VR_{MEAN} decreases. Compliance with the "20 Rule" is likely possible with some operational fine-tuning such as:

1. Curing the underlying cause by improving fragmentation – particle size, grading and shape – in preparing material for loading.

Or, if inhibited by adverse insitu material conditions, the cure is prohibitively costly or unacceptable operationally:

2. Treating the symptoms by controlling truck payloads with a system such as VIMS or equivalent that, if, on reweigh, a truck payload is indicated in excess of 20% overload, triggers an override system such as Caterpillar's MAX Payload Speed Manager. When Reweigh identifies a payload in excess of 120% of target this system within VIMS automatically restricts the transmission to second range and the engine to 1,750 RPM. Speed of large mining trucks on flat roads with typical in-pit rolling resistance is limited to some 14 kph. With engine power so limited climbing ramps in deep pits, if at all possible, will be limited to first range converter drive. So, practically, the unacceptably high payload has to be dumped at the face or elsewhere within the open pit.

Obviously the first of the above treatments is preferred if conditions and economics are not prohibitive. Disturbance to the "rhythm of operations by interfering with hauling routine, a side effect of the second treatment, generally reduces efficiency and affects productivity so is to be avoided. But cost considerations or adverse operating conditions may compel adoption of the second treatment.

Figure 3.42, previous page, comparing Potential % Payload In Excess of Target and CV_{VR} is essentially an alternative expression of Figure 3.41 with Potential % Overload replacing Theoretical Maximum Truck Payload. This illustration clearly shows how truck PL's of reduced VR material potentially have maxima that totally exceed 10 % over target payload. So achieving a maximum of 10% of payloads $\leq$ 10% over target payload clearly becomes difficult as VR moves down from the specified optimum.

Digability

For the purposes of the research described by this thesis "Digability" is defined as the measurable volumetric productivity of loading equipment. Digability is usually expressed as BCM or tonnes per hour. Although neither "digable" nor "digability" is a recognized word in common English usage they are acceptable in colloquial mining terminology in this country for describing performance of loading activities.

Intrinsic (hypothetical) loading performance was defined in Section 3.2.4. Allowing for reasonable changeover time, but for no other time losses by loading equipment other than inching tramming to follow the working face, the resulting reduced production rate is, for the purposes of this analysis, "digability". It assumes that loading equipment does not experience any time waiting for trucks other than reasonable changeover time and that no time is lost for any interruptions such as maintenance, pit house keeping, dust suppression, face problems relocation tramming or operator changeover/inefficiency. Digability is generally measured over short periods of time that comply with the above-described premises; or by accounting for all extraneous time losses that result in reduced productivity.

The premises outlined above ensure that "digability" for an open pit mining operating system so assessed is comparable internally; and also externally with other operating systems.

Essentially "digability" is a productivity attribute that describes the state of the loading activity within an open pit operating system.

Digability is variable, dependent on:

- Voids ratio of the material prepared for loading.
- Particle size, grading and shape.
- Equipment constraints such as ratio of bucket least dimension to maximum dimension of largest particles.

Interpretation and Conclusions

Control and management measuring tools for load and haul productivity, such as described above (VIMS, TPMS and equivalents and Trayscan), facilitate after-the-event accounting for payload volumetric and weight efficiency; and any consequent management action is necessarily reactive rather than proactive.

Effective process for procurement of mining trucks is based on reliable design criteria including specifying body style and capacity to accommodate the volume of loose material that will realise the target mean payload weight. It is necessary to have confidence that, by filling truck bodies to a consistent volumetric level and shape mean target payloads by weight, constrained within a mandatory distribution range, will be achieved consistently and reliably.

Onboard payload measuring is a useful management aid that can provide some guidance for monitoring control of actual payload within a reasonable tolerance or target payload protocol such as Caterpillar's 10/10/20 Policy. Verification of payload volume as provided by systems such as *Trayscan* is also a desirable management aid. But neither of these essentially monitoring systems impose any absolute constraint on the actual payload placed in the truck body. A correctly sized body and predictable, repeatable loose density of material to be hauled will enable competent operators to produce distributions of truck payloads that are confined within acceptable dispersion limits.

Equipment selection and procurement processes and related focus of the research described in this thesis were outlined in Section 2.2.

Fundamental to achieving these objectives is the necessity to have available reliable methods of estimating loose density of materials to be loaded. The current state of knowledge of loose density determination is based on empirical observations of swell – increase in insitu volume in preparing the material for loading and hauling – interpreted as load factor criteria and listed in industrial references such as Cat PHB, 2004 as described above.

The adoption of such reference material is reasonable for general, deterministic, estimating purposes. In the absence of actual operating evidence, criteria selected from references that are empirically based is the best available. But, for an on-going open pit operation, a more scientific and duly diligent approach is possible.

Empirical data is available to determine design criteria. Such criteria include loose density treated stochastically for final, definitive body capacity determination and design detail in larger mining trucks, the performance and operating costs of which are so sensitive to any payloading inefficiency. This stochastic-based analysis and highlighting of the relationship between variability of VR of material prepared for loading onto mining trucks; and the consequent, variability in truck payloads provides useful guidelines.

The value of a more reliable determination of loose density of materials is manifest. Loading bucket and truck body capacities are dependent on a reliable estimate of loose density in the truck body. Voids Ratio (VR) expresses the relationship between loose volume and insitu volume. VR_{MEAN} provides a measurable parameter that can be related to Digability (D), i.e., loading performance; also providing a relationship through fragmentation to drilling and blasting design parameters. Insitu density is a variable but with small variability as described in Section 3.2.7. As loose density is directly related to VR so the significant variability of VR will be reflected by loose density variability. Payloads are directly dependent on loose density of material in a truck body with practically fixed volume. So variability of VR will be reflected by variability of truck payloads.

As VR variability is an outcome of drill and blast efficacy so truck payload variability is dependent on drill and blast (or other pre-loading process) fragmentation and particle grading outcomes.

Section 3.2.7 described how bucket factors are dependent on insitu density, swell (voids ratio) and operator efficiency. Truck filling efficiency will therefore have similar dependencies. Insitu density variability is relatively small. On the basis of experience it is hypothesized that operator skills are predictably high so variability due to operator inefficiency will tend to be relatively small. This leaves swell, i.e., VR variability as the dominant contributor to truck payload variability.

The principal driver of truck payload dispersion is voids ratio dispersion that is, in turn, directly related to drill and blast efficacy.

Fundamental to selection of loading equipment and mining trucks, more specifically loading equipment bucket and truck body capacities, is VR of material to be loaded and hauled. So this measurable parameter, VR, should be tied to selected equipment as a KPI for operators to be able to compare with empirical data from actual operations.

The interrelationship between digability and drilling and blasting is further discussed in Section 6.1.2. Load and haul productivity and cost relationships are further considered in Section 5.3.

The analysis above examined the relationship between CV_{PL} and CV_{VR} and PL_{MEAN} and VR_{MEAN}. Taking a broader view it must be recognized that CV_{PL} and PL_{MEAN} are, as described above, also affected by:

- Variability in insitu density, fortunately small as evidenced above, that manifests cumulatively in CV_{VR}.
- Bucket fill factor that is a function of digability (including VR_{MEAN} and CV_{VR} parameters), and small-effect operator skill and efficiency.
- Number of bucket passes per truck payload (also interrelated with bucket fill factor, digability and operator skill and efficiency) – the effects of which were discussed in some detail in 3.2.8 and 3.2.9 above.

In 3.5 below productivity effects of bunching and queuing are discussed. Consistency, i.e., low dispersion, of truck payloads and loading times is a prerequisite for achieving an acceptable standard of “rhythm” that always characterizes optimum productivity and acceptable operating cost outcomes.

Finally, achieving optimum high VR is essential to preempt problems with digability, bucket filling, truck target payload and compliance with safety and design-permitted protocols for truck operations.

For consistency it is also important to limit VR variability, indicated by CV_{VR}. The outcome from achieving expected values of VR_{MEAN} and CV_{VR} will be forecast productivity and operating costs accompanied by acceptable operating “rhythm”.

VR_{MEAN} and CV_{VR} have been shown, by analysis, to be most useful predictive indicators of the state of the load and haul system and to be a principal attribute for predicting productivity outcomes from load and haul activities. VR values for each blast can be reasonably and simply derived from in-pit surveys. VR_{MEAN} and CV_{VR} are convenient parameters derived directly from VR data samples.

In-truck payload measurement systems, volume or weight, are essentially monitoring facilities, most useful as indicators of productivity and performance activities. Such systems do not solve problems – they only identify that a problem exists and indicate magnitude of the problem.

Ubiquitous management systems such as Caterpillar's TPMS embedded in VIMS; and similar systems from other OEM, integrated with supplementary control systems are now familiar facilities for monitoring truck payload weights and providing management information. Recent technological developments, such as Trayscan, for measuring in-truck volume indicate efficacy of volumetric truck payload. When combined with payload weight sensing, loose density in the truck body is simply calculated – so VR_{MEAN} in the truck body can be derived. Further, VR_{MEAN} in truck bodies can be correlated with VR_{MEAN} in the face. As a means of realizing these performance criteria, payload measuring systems, weight or volume, provide valuable accounting information on operating performance.

The starting point for ensuring efficacy and expected outcomes in loading and hauling activities is quality-assured drilling and blasting practices. Drilling and blasting outcomes are directly related to performance and efficiency of downstream load and haul activities. A convenient and fundamental control parameter through the activity process from blast design to load and haul productivity, and acceptable unit costs, is Voids Ratio and its variability parameters.

As a final thought on compliance with Caterpillars 10/10/20 Policy:

- It is shown above that compliance with the "10/10" rule is likely with a payload CV of ≤ 0.078.
- Compliance with the "20" rule is likely at the more stringent CV of ≤ 0.065.

It is concluded that, for compliance with the "20" rule, CV ≤ 0.065, implying that not more than 10% of payloads should exceed something less than 110% of mean target payload or vice versa. The options for "20" rule compliance appear to be:

- Not more than 6.2% of payloads should exceed 110% of mean target; or
- Not more than 10% of payloads should exceed 108.3% of target mean.

In practice compliance with the "10/10" rule as an operating protocol and rejecting +120% payloads is likely necessarily practical, particularly for Caterpillar trucks.

3.3.7 Truck Payload Centre of Gravity (CG)

This section examines distribution of GMW, NMW (tare) and payload to truck axles and wheels of naturally occurring, but abnormal, operating conditions during hauling operations, Particularly consideration is given to axle load distribution on ramps and the effect of rear wheels running up the toe of the face, generally referred to as "chocking" of truck wheels. The effect of moving the payload CG on distribution of wheel loads is examined; and payload measuring using suspension cylinder (ride strut) pressure differentials, and expected accuracy of this technique, is also considered.

The general thrust of investigation in this section is to develop some understanding of the magnitude of load-transferring effects between axles due to payload misplacement; effect of ramps and virtual load increases or decreases due to braking of stationary trucks. Variability of tyre loads, particularly overload, is significant operationally, particularly as truck size increases.

The GMW of a mining truck, determined on level ground with the truck stationary and brakes off, or at constant velocity consists:

- Suspended weight that includes the chassis, body driveline components:
    - For mechanical drive trucks - forward of the main drive shaft (that can oscillate via two constant velocity {universal} couplings).
    - For electric drive trucks – all electrical components except the wheel motors.
- All services, onboard storage of fuel, lubricants, hydraulic fluids, coolant and payload.
- Unsuspended weight that includes the pistons and lower sections of the ride struts, rear wheel group including any driveline componentry attached to it (mechanical transmissions or electric wheel motors and final drive reduction gearing) and the wheels and tyres.

The following significant issues need to be considered:

- Variation of GMW distribution to front and rear axles – particularly tyre loads in excess of designed expectancy on which forecasts of tyre performance and operating costs are based.
- Operational circumstances that generate excessive loads on truck structural componentry including chassis, suspension, A-frame trunnion, four bar linkage or other arrangements that transfer driving and braking forces through the mechanical systems of the truck with the potential to reduce component life and increase maintenance costs.
- How accuracy of on-board load-sensing management systems based on ride strut pressure variations might be reduced by variation of CG location due to abnormal off-design payload configurations; also
- Operational circumstances that result in inaccuracy and unreliability of payload management systems.

Limited consideration is also given to results of loading off the fore and aft centerline of trucks.

Parts A to F in this section consider three basic problems:

1. Consideration of axle and wheel loads as payload is hypothetically misplaced in truck bodies under theoretical conditions where simple statics can validly analyze the force systems applying and provide understanding as discussed in Parts A to D and the complementary analysis in MPN 8, Parts A to D in Mathematical Principles – Notes.
2. Considerations of situations where braking and drive torque are necessarily applied and further complicate mechanical analysis as shown in Parts E and F and the related detailed analysis in Parts E and F in MPN 8, Mathematical Principles – Notes.
3. As a natural extension of load distribution considerations, to review accuracy of load measuring systems based on ride strut pressure sensing in practical applications.

The following generally summarises analysis provided in detail in MPN 8, Mathematical Principles – Notes appended in Volume 2.

Part A. Solution for Payload Centre of Gravity Position

OEM normally specify distribution of NMW and GMW on front and rear axles of mining trucks. Using a Caterpillar 793C nominal 218 tonne truck as a typical example:

Table 3.39 Typical Mining Truck Axle Load Distributions

Distribution Item	**Front Axle %**	**Rear Axle %**	**Total %**	**Total Tonnes**
GMW Tonnes	33.3	66.7	100	383.7
NMW Tonnes	43.5	56.5	100	160.8*
Target Payload Tonnes	**?**	**?**	100	222.9

* *Total Tonnes* for NMW depends on customer specified configuration as described in more detail in Section 4.1.5.

The above data are for nominal Caterpillar specifications (Caterpillar PHB 35, 2004).

Simple statics analysis of wheel loads determines that:

1. Nominal truck payload is distributed:

Front %	Rear %
26.0	74.0

2. CG of truck payload is some 1.53 metres from the rear wheel centre i.e., 26% of the truck wheelbase – as indicated by the axle distributions of truck payload.

In Section 3.3.9, increasing sensitivity to overload of front tyres as trucks increase in size is discussed in some detail. For the purposes of this current discussion it is sufficient to recognize the potential problem. As an aid for controlling placement of payloads in mining trucks OEM are providing markers on body rails as illustrated by the typical example in Figure 43. As mining trucks have increased in size, tyre loads have become more critical as discussed in Section 3.3.9. Correct placement of the load to control wheel load distributions, both longitudinally and transversely within the body has become essential for large mining trucks – so the provision of markers. It should be noted that the marker is for bucket load placement. The marker is not intended to, and does not necessarily; indicate the position of the payload peak or the CG of the payload.

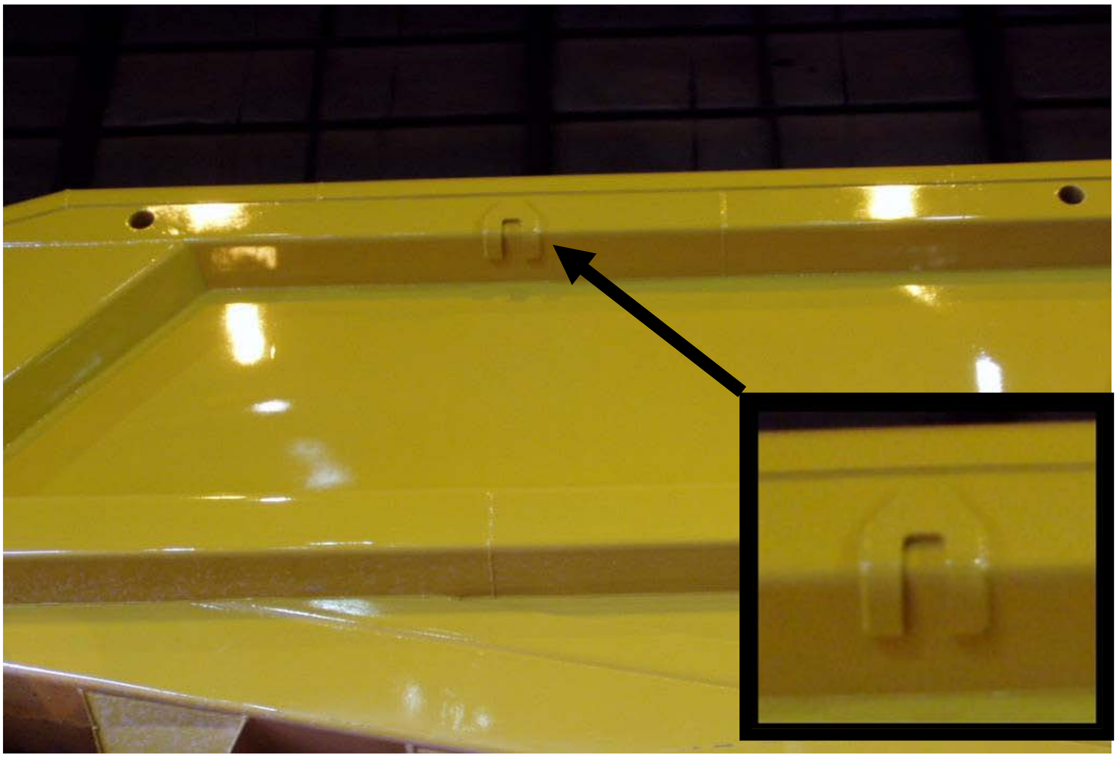


Figure 3.43 Load Centre Marker for MSD Bodies (Cat 793C Mining Truck as example)

It should be noted the marker indicates where bucket loads should be placed – not the Centre of Gravity of Payload see Section 3.3.7 for full discussion.

Particularly for three and four pass loading, it is recommended by OEM that each bucket load be placed as indicated by the marker. For five or more passes the early bucket loads should be spread evenly over the length and width of the body with the final two or three bucket loads placed at the marker position.

Part B. Proportion of Payload Subject to Misplacement

Before considering the affect on wheel loading and suspension struts of misplacement of payload, the proportion of payload that can actually be misplaced needs to be considered. Using a Caterpillar 793C truck as an example, Standard Capacity Specifications are provided in Table 3.40.

Table 3.40 Standard Capacity Specifications for a Caterpillar 793C Mining Truck

Body Type,	Heaped 2:1 Cubic Metres,	Struck Cubic Metres,	Heaped 2:1/Struck,
Flat Floor,	147.6	110	1.34,
Dual Slope	129	96	1,34

For the purposes of the following discussion it will be assumed that some 34% of the struck volume is superimposed above the side rails of the body (equivalent to 0.34/1.34 = 25.4% of total payload).

The form and shape of the payload proportion superimposed above the struck volume has been discussed in Section 3.3.5. Current industry direction is to assume a conical form with slopes steeper than 2:1 – in the order of 1.7:1 – although slopes as steep as 1.5:1 are observed in actual practice. For the following discussion, payload slope of 1.5:1 has been adopted.

As shown in MPN 8, Part B, Mathematical Principles – Notes, amending the shape of the superimposed load from a pyramidal to a conical form and changing the slope from 2:1 to 1.5:1 indicatively increases the **proportion of superimposed load relative to struck volume to 0.40**.

So the ratio of **superimposed volume to total payload = 0.40/1.40 = 28.6%**

Providing for a small freeboard allowance below the side rails **≈ 30%**

It is hypothesized that the struck proportion of the payload is practically confined, so the CG of the contained portion, the struck volume, remains practically fixed. The "superimposed" load is assumed to have an essentially symmetrical form with a centre of mass at the sectional centroid in all dimensions. This proportion of the payload is considered to be misplaced to varying degrees to determine effects on the suspension and wheel loads. That is, only the superimposed load can be misplaced from the designed position.

For the nominal payload of 222.9 tonnes:

Proportion of payload fixed in position = 0.7 . 222.9 = 156.0tonnes

Proportion of payload that can be misplaced = 0.3 . 222.9 = 66.9tonnes

For convenience of calculations it is assumed that the CG locations for the superimposed load and the contained load (struck volume) and the combined payload are all on the same vertical alignment. Considering the wedge shape of the body in elevation, the natural repose slope at rear and steeper containment at the front of the body the base target for the superimposed load is approximately square. There is no certainty that the component and combined CG do align. But the actual misalignments will be small and are considered to not invalidate general outcomes of calculations and interpretations.

It should also be noted that, for the hypothetical cases considered, the superimposed load only is misplaced forwards. Theoretically the rear slope could move forward, also leaving a void at the rear of the contained (struck) volume and potentially under-loading the truck. No provision has been made for this rear slope void and potential under-loading for two reasons:

1. It is considered that the effect on the distribution calculations would be small.
2. A void on the rear slope will be obvious to the loading equipment operator; and skilled operators will place bucket loads to remedy any such obvious voids.

Part C. Relationship Between Misplacement and Distribution of Superimposed Payload To Wheels

Five cases were considered:

- A hypothetical case where the superimposed payload proportion only moves 0.5 metres to the rear to increase rear suspension and wheel loads – questionable practically as the open and uncontained rear slope has little accommodation for rearward misplacement of the superimposed load before spillage occurs.
- Generally practical cases of misplacement of the superimposed proportion of payload forward by 0.5, 1.0, 1.5 and 2.0 metres; of these 1.5 and 2.0 metres, are also likely impractical because the payload would manifestly spill across the canopy and be obvious to the operator and supervision.

By apportioning the components of GMW - in accordance with relationships of simple statics – distribution of GMW on front and rear wheels is derived. Results are relative to the design CG position that realizes a distribution of 33.3% to front and 66.7% to rear wheels when the truck is on a horizontal surface.

Table 3.41 summarises the spreadsheet analysis for axle load proportions for a range of payload displacements in MPN 8, Part C, Mathematical Principles – Notes. Figure 3.44 illustrates the results of analysis.

Table 3.41 - Variation of Axle Load Distributions For Superimposed Payload Displacement

	Positive Values = Forward Distance from Designed Position						
Displacement of Superimposed Payload	**Metres**	**-0.50**	**0.00**	**0.50**	**1.00**	**1.50**	**2.00**
Proportion Of Wheelbase	**%**	**-8.47%**	**0.00%**	**8.47%**	**16.95%**	**25.42%**	**33.90%**
Distribution Variation							
Front Wheels	**%**	**31.85**	**33.33**	**34.81**	**36.28**	**37.76**	**39.24**
Rear Wheels	**%**	**68.15**	**66.67**	**65.19**	**63.72**	**62.24**	**60.76**

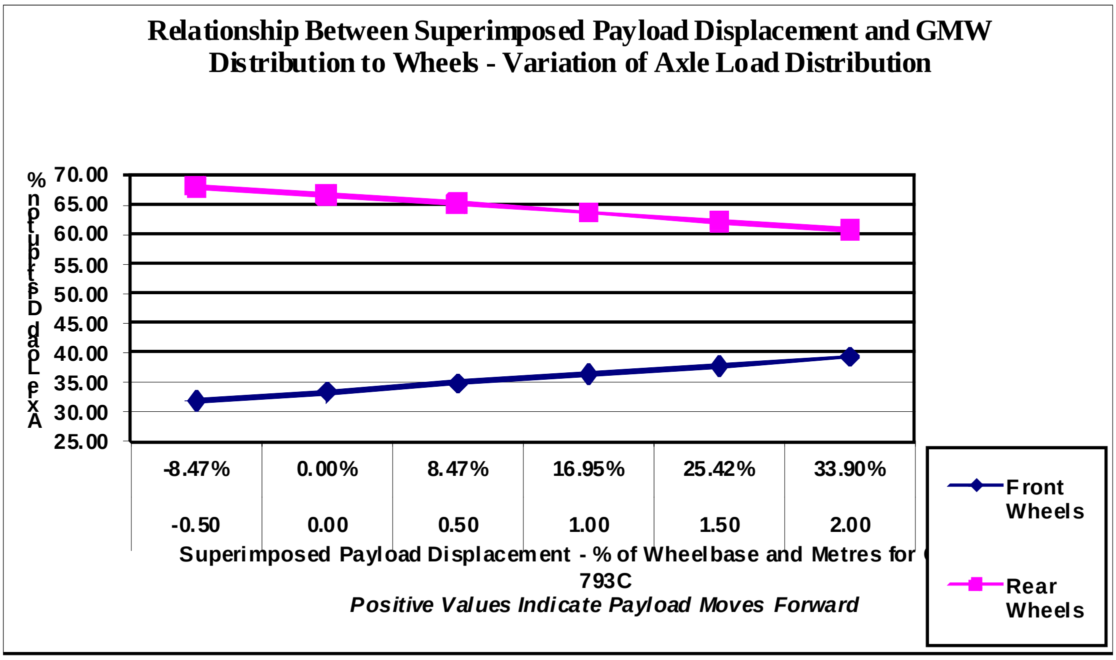


Figure 3.44 From Table 3.41

Table 3.42 summarises analysis of individual wheel loads; also showing the relative load change with displacement, i.e., the load on a single tyre for the same range of payload displacements as adopted in Table 3.41. The results in Table 3.42 are illustrated by Figure 44.

Table 3.42 Variation In Wheel Loads For Payload Centre of Gravity Displacement

	Positive Values = Forward Distance from Designed Position						
Displacement of Superimposed Payload	Metres	-0.50	0.00	0.50	1.00	1.50	2.00
Proportion Of Wheelbase	%	-8.47%	0.00%	8.47%	16.95%	25.42%	33.90%
Distribution							
Front Wheels	%	15.927%	16.67%	17.403%	18.142%	18.880%	19.619%
Rear Wheels	%	17.037%	16.67%	16.298%	15.929%	15.560%	15.191%
Percentage Increase							
Front Wheels	%	-4.431%	0.000%	4.431%	8.862%	13.294%	17.725%
Rear Wheels	%	2.215%	0.000%	-2.215%	-4.431%	-6.646%	-8.861%

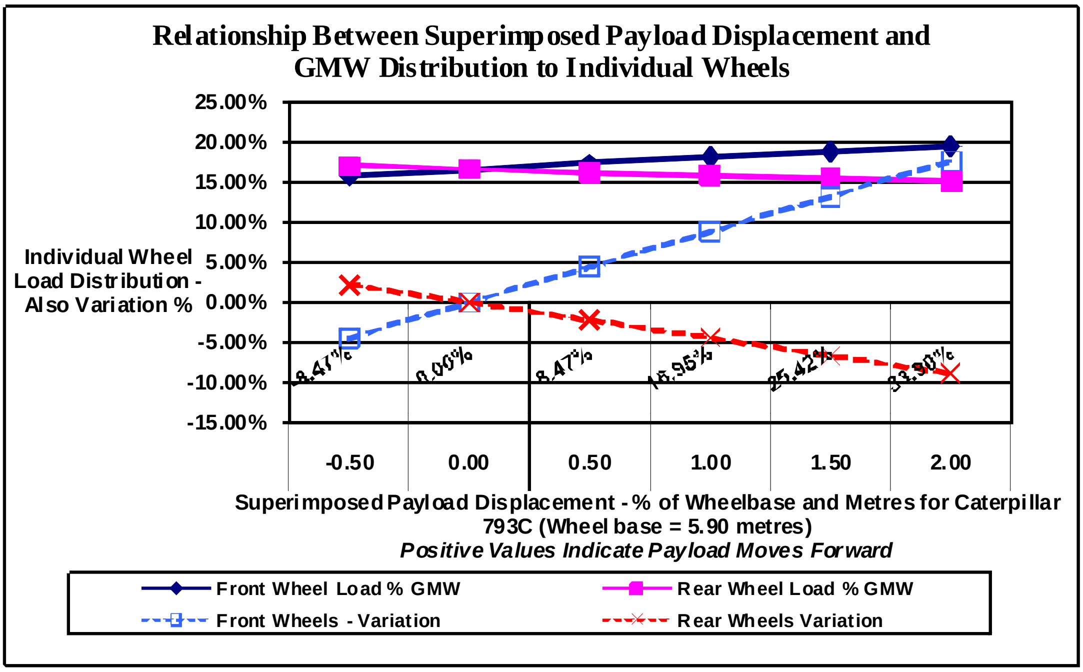


Figure 3.45 From Table 3.42

Part D. Relationships Between Payload Transfer Systems

Following is a general description of the componentry that collectively constitute the load transfer systems of mining trucks.

- Firstly, payload is enclosed and practically constrained by the truck body.
- The combined weight of payload and body is transferred to the truck chassis or frame through damping support systems (generally elastomer blocks) locating the body along and across the frame.

- Tipping trunnion pins are not designed to be under load except during body tipping – any load on trunnion pins with the body down is due to collapse of the damped body elastomer support system or misalignment/warping of body structure.
- The chassis and all load components above it are supported by the suspension system generally consisting of inert gas over oil suspension cylinders commonly termed “ride struts” or simply “struts”.
- Oil in suspension cylinders acts as a gas seal and oil seal lubricant – oil seals are generally a group of elastomer o-rings with the outside ring designed to act as a wiper seal to prevent build up of contaminated oil on the cylinder shaft and consequent contamination of contained oil.
- Other suspension mediums, including stacked elastomer blocks acting as compression elements, have also been used with only limited success – such alternatives have not carried forward into current designs for large mining trucks.
- Generally front ride struts have longer travel than the rear struts; but across the range of large mining trucks available there are exceptions where front and rear struts have practically equal travel and at least one truck where rear-strut travel exceeds travel for front struts.
- Rear struts carry some double the load of a front strut; so rear struts are necessarily stiffer, i.e., have a higher spring rate (load over travel) than front struts.
- Front ride struts are generally in line with the front wheel centres for all mining trucks – in some designs, particularly Caterpillar and Komatsu, the front suspension cylinder acts as the steering king post with a stub axle mounted onto the strut shaft; other designs have trailing links positioning integral stub axles or double wishbone configurations supporting a separate king post/stub axle assembly, similar to light highway vehicles. These alternative front suspension systems that position the front axle have suspension struts mounted between front wheel support componentry and the chassis frame with universal pin and bush mountings designed to accommodate loads only in line with the strut.

- Rear struts are mounted between chassis frame and rear axle group either behind the centerline of the rear axle and wheels or directly above the rear axle/wheel group on the axle centerline. Mountings are generally universal so that, for practical purposes, struts only experience in-line loads.
- Where the rear ride struts are mounted behind the rear axle centre, a secondary support system consisting of an A frame or four-bar linkage transfers suspension torque to the chassis as horizontal or vertical forces – this secondary support system also acts as thrust or draw bar componentry that accommodates drive and braking torque and transmits rimpull reaction to the truck chassis to drive the truck forward or in reverse.
- There is no commonality of suspension system designs amongst the five major OEM supplying mining trucks (Caterpillar, Komatsu, Liebherr, Terex and Hitachi) with each vendor marketing the claimed special benefits of each system – all of which are functional; and all of which exhibit only small variations in geometrical displacement as a natural consequence of limited suspension movement under operational conditions.

The following descriptions and analysis apply generally to the Caterpillar range of mining trucks – adopting the Cat 793C nominal 240 ton (218 tonne) truck as a model for analysis. Similar analysis, differing only in detail could be applied to any mining truck from any OEM.

Special features of the generic Caterpillar truck configuration include:

- Mechanical drive system for the complete range of mining trucks available at time of writing this thesis – a situation that may change in the future.
- Torque converter and lock up componentry close mounted to the engine flywheel housing and transmission close mounted to the rear wheel assembly connected by a relatively short drive shaft including constant velocity joints allowing independent rear axle group movement.
- An A-frame provides rear axle location with the forward trunnion (a ball joint) allowing universal movement of the rear axle group – a common feature of the range of Caterpillar mining trucks up to the Cat 793C nominal 218 tonne truck – that has been changed to a 4-bar linkage for the top-of-

range Cat 797B truck, likely because of longitudinal space limitations due to the extra length 24 cylinder engine assembly currently used in Caterpillar's largest mining truck.

- The combined rear axle group constitutes a large unsuspended mass of componentry with a substantial overall longitudinal dimension compared to the relatively short wheelbase.

The listed special features are illustrated by Figure 3.46.

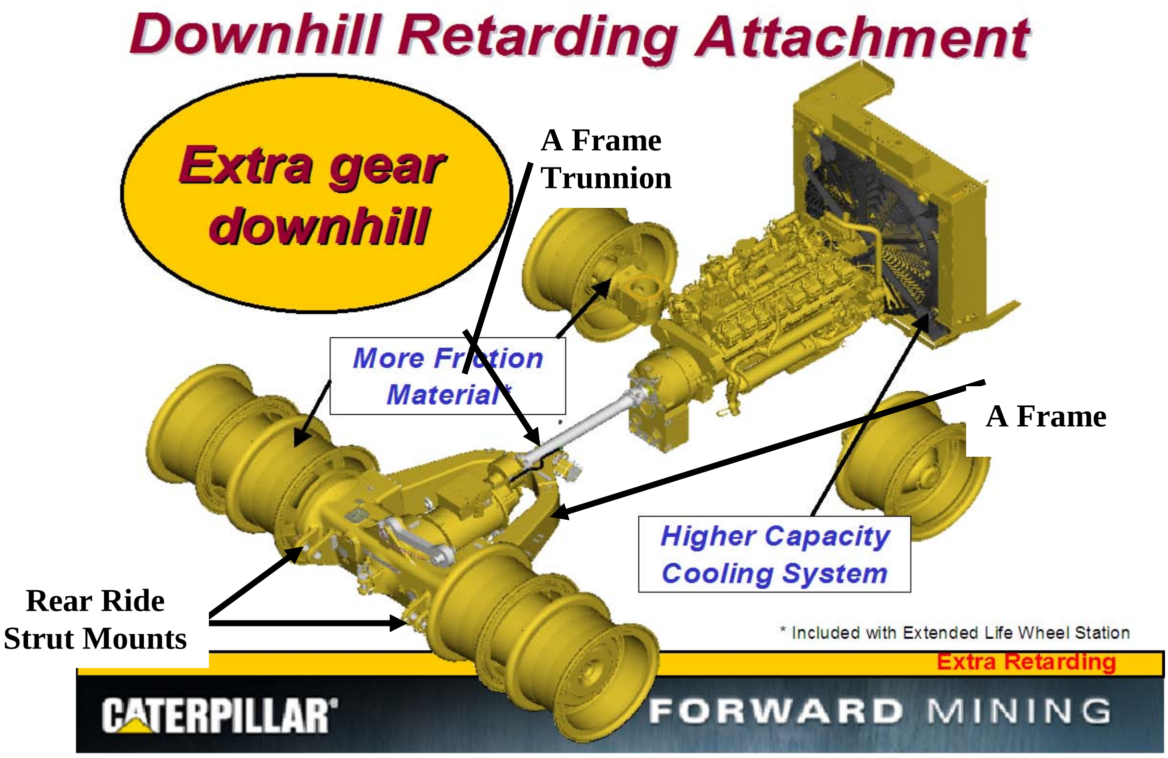


Figure 3.46 Axle Group Arrangement (Cat 793D as example)

Load and haul operations on open pit benches are often relatively restricted. To retain reasonable maneuverability, turning circles of mining trucks must be limited. So necessary reduced ratio of wheel base to wheel track dimensions is a significant design accommodation for extrapolation of generic mining truck designs to the largest trucks available at this time. This limits the longitudinal space for the driveline componentry between front and rear wheels that:

- As indicated above, is already influencing truck componentry design as truck scale increases.

- Is a constraint on extrapolation of current configurations to even larger mining trucks than are offered by OEM at this time.

Analysis in terms of simple statics of the load transference systems, using a Cat 793C truck as a model, are provided in a series of equations in MPN 8, Part D, Mathematical Principles – Notes. These theoretical, simple statics relationships provide a basis for general understanding but are dependent on some assumptions that are not necessarily absolutely true in practice. These assumptions include:

- Trucks are on level ground and remain horizontal during loading.
- Trucks are stationary and remain so without application of brake or drive torques unless specifically stated otherwise.
- Geometrical relationship between the various load transfer systems is retained both intrinsically and in space.
- All external forces applied to ride struts are represented without loss (due to friction) by the pressure differentials in the ride struts.
- There are no transverse force components that reduce the force applied to ride struts.

The above list is not represented as exhaustive. In practice the load transfer systems and load sensing facilities of large mining trucks cannot be exactly modelled by simple theoretical concepts. The geometry of the suspension and load transfer systems does not remain constant. Small, but significant, changes to strut alignment due to difference in front and rear strut travel under incrementally changing load, appear to play a significant role in determining accuracy of load measuring systems based on suspension pressure differentials. Also, friction in ride strut seals affects response to discrete load variations – with reduced degree of friction effect where strut loads are varied dynamically, i.e., when the truck is travelling.

Part E. Rear Wheels On Toe of Face Or Otherwise Chocking Wheels

Effects on rear wheel load and suspension resulting from chocking the wheels whilst spotting the truck for loading are analyzed in MPN 8, Part E, Mathematical Principles – Notes appended in Volume 2. Practical causes of wheel chocking include reversing a pair of rear wheels onto the toe of the face, or wheels contacting debris – tailings - from loading operations and the like.

A general practical case where only one rear group of dual wheels is chocked is adopted for analysis. It is possible for a single rear wheel to experience chocking effects. But the analysis is based on each wheel of a dual pair sharing forces from chocking effects equally. Specific outcomes of the case of chocking a single wheel are generally discussed below. The equations developed in MPN 8, Part E, Mathematical Principles - Notes will generally be valid for the static forces applying in the hypothesized circumstances.

Incipient dynamic forces, resulting from braking torque to restrain the chocked truck in position, are distributed through the chassis to manifest as forces on ride struts that are, in turn, distributed to front and rear axles. Even if one rear dual wheel group or single wheel only is chocked, ride struts for a specific axle, front or rear, will generally share any virtual increase in suspended load equally across the truck – i.e., incremental virtual load on left and right side struts should be practically equal.

Diagrams illustrate both static and incipient dynamic sets of loads and relevant equations are provided in the table in Part E, (MPN 8, Mathematical Principles – Notes:

1. Static loads on a rear wheel group of a pair of dual wheels due to the load offsetting effect of chocking are resolved vertically and horizontally only affecting tyre and wheel loads.
2. Incipient dynamic loading resulting from braking torque to retain the truck in position (equilibrium) is resolved into components that affect ride-strut loads.

Using a Cat 793C with 40.00 R57 tyres as a model, Table 3.43 indicates:

- Static load increase in tyre-support reaction on a wheel group of a pair of wheels/tyres for a range of offsets caused by chocking.
- That, if chocking affects a single wheel and tyre, the increase in individual wheel/tyre load could be double the static load increase indicated by Table 3.43

Incipient dynamic force resulting from brake torque to restrain the truck in the hypothetical chocked position is experienced as virtual additional GMW - in effect a payload increase. Developed and provided in Table 3.43 are:

- Braking-torque generated virtual increases for both GMW and payload - identified as "Incipient Dynamic (Virtual) Loads".
- Distribution of Virtual Load Increase to front and rear axles – both relative to GMW and indicated payload (where payload is derived from increase in strut loads from a set tare i.e., NMW).

Table 3.43 - Direct Tyre Load Increase For Offset of Rear Axle Loads Braking Torque Virtual Load & Distributed Axle Loads

Cat 793C Data - Tyre diameter assumed:	= 3.48	metres
Wheel Base wb:	= 5.90	metres
A frame Trunnion Radius d_y:	= 2.54	metres
Rear Suspension Radius d_s:	= 0.98	metres
Payload/GMW:	= 0.58	

Displacement of Tyre Load					
Cat 793C 40.00 R57 Tyres	Degrees	8.26	16.70	25.53	35.08
	metres	*0.25*	*0.50*	*0.75*	*1.00*
Static Loads					
Tyre Load Increases To	%	101.048%	104.403%	110.824%	122.196%
Incipient Dynamic (Virtual) Loads					
Braking Torque Reaction B_T	%	9.84%	19.69%	29.53%	39.37%
At A Frame Trunnion Relative to Rear Wheel Group Load					
Virtual Load Increase					
Relative to GMW	%	103.25%	106.50%	109.74%	112.99%
Relative to Payload	%	105.59%	111.18%	116.77%	122.36%
Distribution of Virtual Load Increase					
Relative to GMW					
Front Wheels	%	101.66%	103.32%	104.99%	106.65%
Rear Wheels	%	101.59%	103.17%	104.76%	106.34%
Relative to Payload					
Front Wheels	%	102.86%	105.72%	108.58%	111.44%
Rear Wheels	%	102.73%	105.46%	108.19%	110.92%

The above-derived static and virtual loads for the hypothetical chocking case are illustrated in Figure 3.47.

It will be noted that, with the A frame arrangement for locating and transferring driving and braking forces to the chassis, virtual loads experienced by ride struts are distributed approximately equally to front and rear axles. Where trucks use a four bar linkage to locate the rear wheel group and transmit driving and braking forces, resulting torque induced forces are transmitted to the chassis in the approximately horizontal orientation of the links. In this case the static forces resulting from the chocking offset are essentially the same as for the A frame arrangement. The braking torque results in a reactive moment applied to front and rear ride struts. Front struts experience additional load, rear struts reduction in load distributed in inverse proportion to the lever-arm distance from the centre line of four-bar link connections to the chassis. The resulting outcomes will be practically similar for A frame and four-bar link systems with the generally shorter four-bar link and similar systems transferring slightly reduced virtual load to front ride struts.

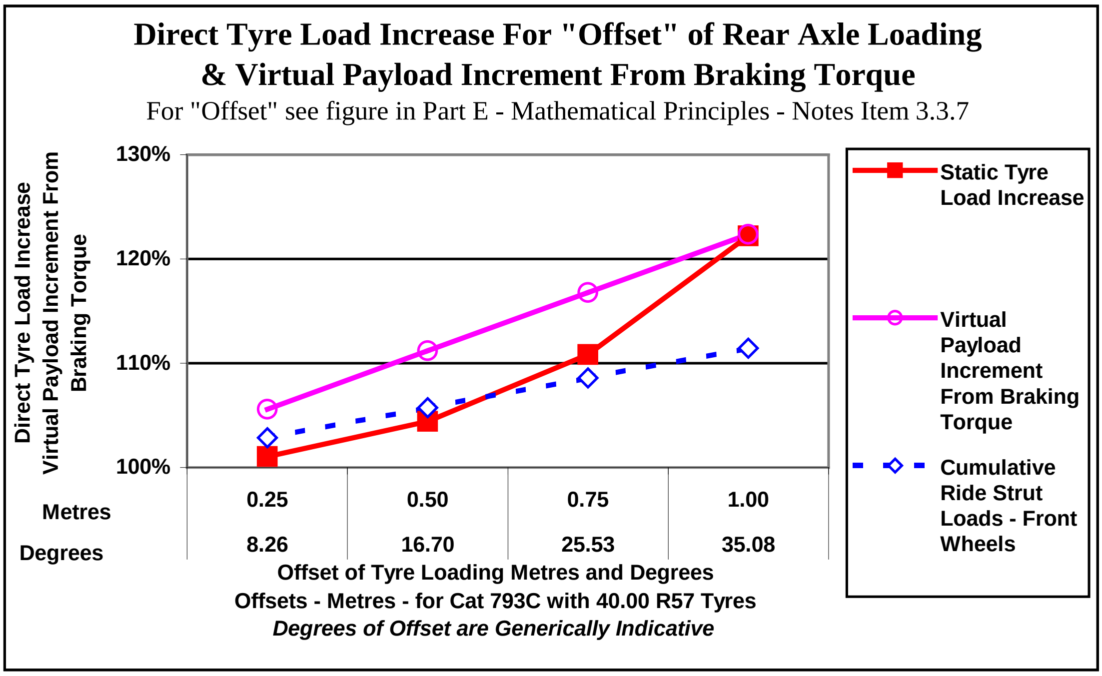


Figure 3.47 From Table 3.43

In summary:

- It is generally understood that reversing rear wheels onto the toe of the face or otherwise chocking the wheels at the truck-loading point should be avoided.
- Offsets up to one-third of wheel radius (some 0.6 metres for a 44.00R57 tyre) due to rough pit floors appear to be practically possible – higher offsets included in the analysis, to show the trend, are likely academic.
- A pair of dual wheels with one-third-radius offset loading affect will experience some 105% of normal tyre/wheel load both dual wheels sharing the load equally. If the chocking applies only to a single wheel tyre load for

that wheel could theoretically increase to more than double, some 210% of normal tyre and wheel load.

- Braking torque to restrain the truck in the chocked position induces loads on the chassis that manifest as virtual additional higher GMW and consequently increased payload as sensed by suspension components.
- Payload measuring facilities such as Caterpillar's VIMS will tend to record high when a rear wheel is chocked, the truck tends to roll forward, and brakes are used to maintain the truck in position; and vice versa if the truck tends to roll backwards.
- The various tables and illustrations referred to provide an indication of the potential degree of over-assessment of payload by using suspension sensing for chocked-wheel situations.
- When brakes are released and the truck is on a level surface load-suspension measurement facilities can be expected to return to normal.
- Since trucks are usually loaded with the brakes on (generally rear wheel parking brakes only) and driveline in neutral or off-power, the load sensing system will potentially over-record for forward movement tendency or under-record for rearward movement tendency.
- To avoid these potential sensing errors, VIMS reweighs the truck in forward motion in second transmission range by sampling ride strut pressures a number of times, deriving a mean value, deducting pre-determined NMW and recording the nett result as payload. Other OEM are understood to adopt similar techniques.
- It will be noted that the chocking induced static forces affect tyre and wheel loads and incipient dynamic, virtual, forces affect suspension loads independently.

Part F. Load Distribution on Ramps

In MPN 8, Part F Load Distribution On Ramps, Mathematical Principles – Notes appended in Volume 2, analysis indicates how axle-load distribution varies with ramp grade.

The analysis is based on hauling upgrade. Downgrade hauling can be considered using the same analytical arguments with appropriate adjustments to applicable geometrical configuration of the truck and loading system.

Essentially:

- Upgrade hauling transfers axle loads that reflect distributed GMW (including payload) from front to rear wheels.
- The degree of transfer increases with ramp grade and height of the CG of the loaded truck above the wheel centres.

Focus of this part F is limited to the order of magnitude of axle-load transfer in terms of the two variables – ramp grade and CG height.

MPN 8, Part F Load Distribution On Ramps, Mathematical Principles – Notes appended in Volume 2, provides details of the analysis. Calculations of axle-load distribution for ranges of effective ramp grades and CG height are summarized in Tables 3.44 – Upgrade and 3.45 – Downgrade, next two pages. The results in these tables are illustrated by Figures 3.48 - Upgrade and 3.49 – Downgrade. Comparing these two figures it is obvious that increase in front wheel loading downgrade is significantly greater than for rear wheels upgrade, e.g.;

At 10% Ramp Grade, CG height = 4.0 metres (assumed) – indicated wheel load variations:

	Front Wheels:	**Rear Wheels**
Upgrade	-30%	+14%
Downgrade	+40%	-20%

The significance of the above outcomes for expected tyre performance/life on downgrade-loaded hauling is manifest.

The discussion is limited to static loading only.

For upgrade hauling, driving torque applied at the wheels tends to further reduce load on the front wheels and increase load on the rear wheels. For downgrade-loaded hauling, braking torque has the opposite effect loading up front wheels and relieving load on rear wheels.

Table 3.44 GMW Distribution & Wheel Loads With Increased Grade – ***Upgrade Loaded***

Cat 793C wb = 5.90 m

Road Gradient						
Gradient:- Percent	3.33%	4.00%	5.00%	6.67%	10.00%	15.00%
Angle□ Degre	1.91	2.29	2.86	3.82	5.73	8.59
Sin□	0.033	0.040	0.050	0.067	0.100	0.149
wb'/wb = Cos□	0.999	0.999	0.999	0.998	0.995	0.989
wb'/wb	0.999	0.999	0.999	0.998	0.995	0.989
Example Cat 793C Estimated Height of Centre of Gravity "h" metres	3.00					
g'/wb	0.316	0.313	0.308	0.299	0.282	0.257
W'F/W'R	0.462	0.455	0.445	0.428	0.396	0.351
Wheel Load Distribution:						
Front Axle	0.316	0.313	0.308	0.300	0.284	0.260
Rear Axle	0.684	0.687	0.692	0.700	0.716	0.740
Individual Wheels% of Design:						
Front Wheels %	94.87%	93.87%	92.39%	89.94%	85.09%	77.98%
Rear Wheels %	102.56%	103.06%	103.80%	105.03%	107.45%	111.01%
Estimated Height of Centre of Gravity "h" metres	3.50					
g'/wb	0.313	0.309	0.303	0.293	0.274	0.244
W'F/W'R	0.440	0.429	0.413	0.388	0.342	0.282
Wheel Load Distribution:						
Front Axle	0.305	0.300	0.292	0.279	0.255	0.220
Rear Axle	0.695	0.700	0.708	0.721	0.745	0.780
Individual Wheels% of Design:						
Front Wheels %	91.62%	90.03%	87.67%	83.83%	76.44%	66.04%
Rear Wheels %	104.19%	104.98%	106.16%	108.08%	111.78%	116.98%
Estimated Height of Centre of Gravity "h" metres	4.00					
g'/wb	0.310	0.306	0.299	0.288	0.265	0.232
W'F/W'R	0.424	0.411	0.392	0.363	0.311	0.247
Wheel Load Distribution:						
Front Axle	0.298	0.291	0.282	0.266	0.237	0.198
Rear Axle	0.702	0.709	0.718	0.734	0.763	0.802
Individual Wheels% of Design:						
Front Wheels %	89.38%	87.41%	84.52%	79.89%	71.21%	59.42%
Rear Wheels %	105.31%	106.29%	107.73%	110.05%	114.39%	120.29%
Estimated Height of Centre of Gravity "h" metres	4.50					
g'/wb	0.308	0.302	0.295	0.282	0.257	0.219
W'F/W'R	0.413	0.398	0.377	0.345	0.290	0.223
Wheel Load Distribution:						
Front Axle	0.292	0.285	0.274	0.256	0.225	0.182
Rear Axle	0.708	0.715	0.726	0.744	0.775	0.818
Individual Wheels% of Design:						
Front Wheels %	87.62%	85.38%	82.11%	76.92%	67.36%	54.62%

Rear Wheels %	106.19%	107.31%	108.94%	111.54%	116.32%	122.68%

Table 3.45 GMW Distribution & Wheel Loads With Increased Grade – *Downgrade Loaded*

Cat 793C: wb = 5.90 m

	Road Gradient					
Gradient:- Percent	3.33%	4.00%	5.00%	6.67%	10.00%	15.00%
Angle□ Degre	1.91	2.29	2.86	3.82	5.73	8.59
Sin□	0.033	0.040	0.050	0.067	0.100	0.149
wb'/wb = Cos□	0.999	0.999	0.999	0.998	0.995	0.989
wb'/wb	0.999	0.999	0.999	0.998	0.995	0.989
Example Cat 793C Estimated Height of Centre of Gravity "h" metres	3.00					
g'/wb	0.350	0.353	0.358	0.367	0.384	0.409
W'F/W'R	0.539	0.547	0.560	0.582	0.628	0.705
Wheel Load Distribution:						
Front Axle	0.350	0.354	0.359	0.368	0.386	0.414
Rear Axle	0.650	0.646	0.641	0.632	0.614	0.586
Individual Wheels% of Design:						
Front Wheels %	105.04%	106.08%	107.66%	110.30%	115.70%	124.09%
Rear Wheels %	97.48%	96.96%	96.17%	94.84%	92.14%	87.95%
Estimated Height of Centre of Gravity "h" metres	3.50					
g'/wb	0.353	0.357	0.363	0.373	0.392	0.422
W'F/W'R	0.567	0.582	0.605	0.647	0.741	0.921
Wheel Load Distribution:						
Front Axle	0.362	0.368	0.377	0.393	0.426	0.479
Rear Axle	0.638	0.632	0.623	0.607	0.574	0.521
Individual Wheels% of Design:						
Front Wheels %	108.57%	110.37%	113.13%	117.83%	127.70%	143.81%
Rear Wheels %	95.71%	94.81%	93.43%	91.08%	86.15%	78.09%
Estimated Height of Centre of Gravity "h" metres	4.00					
g'/wb	0.356	0.360	0.367	0.378	0.401	0.434
W'F/W'R	0.591	0.612	0.647	0.710	0.870	1.253
Wheel Load Distribution:						
Front Axle	0.372	0.380	0.393	0.415	0.465	0.556
Rear Axle	0.628	0.620	0.607	0.585	0.535	0.444
Individual Wheels% of Design:						
Front Wheels %	111.45%	113.95%	117.80%	124.55%	139.53%	166.85%
Rear Wheels %	94.27%	93.02%	91.10%	87.72%	80.23%	66.57%
Estimated Height of Centre of Gravity "h" metres	4.50					
g'/wb	0.358	0.364	0.371	0.384	0.409	0.447
W'F/W'R	0.613	0.641	0.687	0.778	1.041	2.043
Wheel Load Distribution:						
Front Axle	0.380	0.391	0.407	0.438	0.510	0.671
Rear Axle	0.620	0.609	0.593	0.562	0.490	0.329
Individual Wheels% of Design:						
Front Wheels %	114.06%	117.23%	122.21%	131.26%	152.98%	201.41%
Rear Wheels %	92.97%	91.38%	88.89%	84.37%	73.51%	49.29%

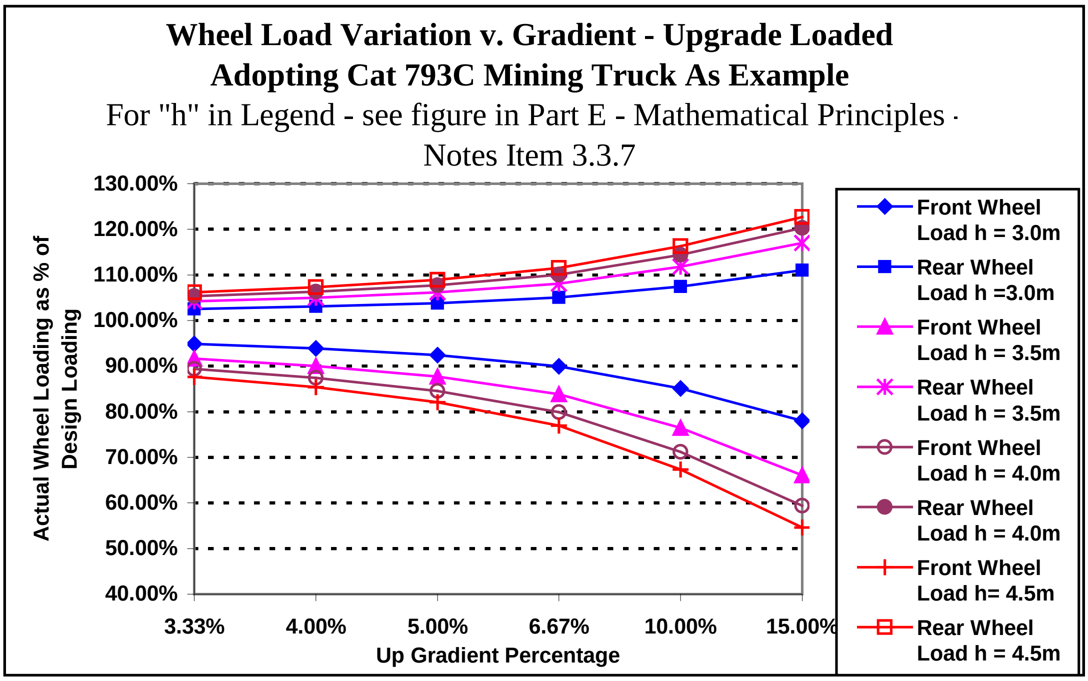


Figure 3.48 From Table 3.44

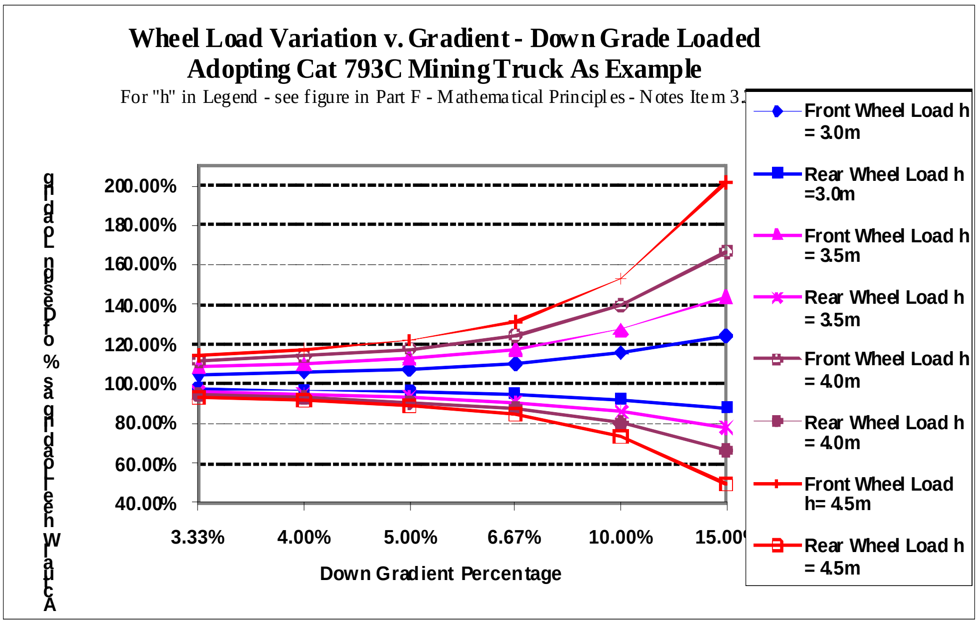
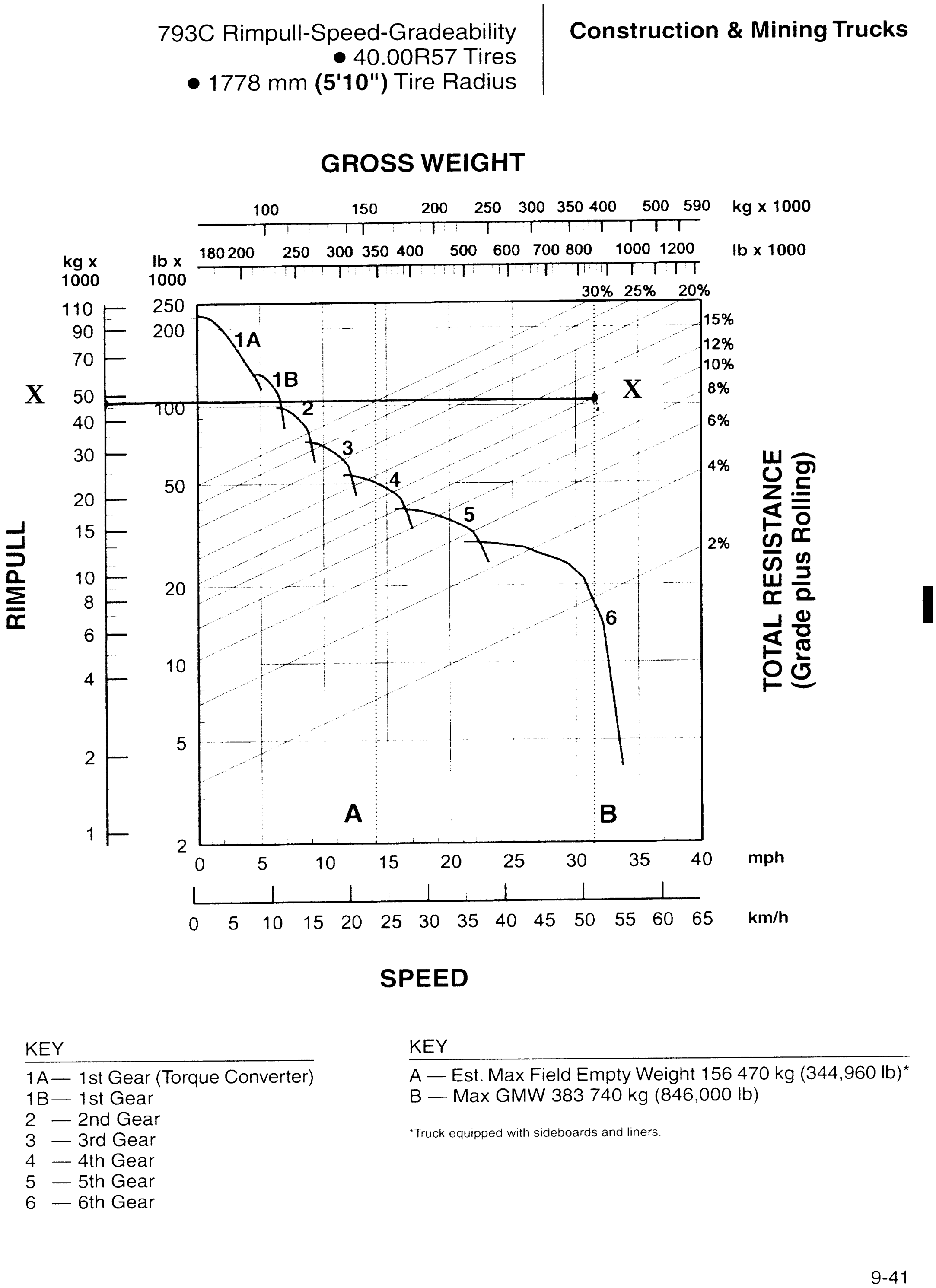


Figure 3.49 From Table 3.45

Consistent with the limited focus of this Part F, the dynamic driving and braking torque-generated forces have not been analyzed in detail. Some appreciation of the relative magnitude of these effects is indicated by rimpull - load diagrams generally used for performance estimating. An example of a rimpull – load diagram for a Cat 793C mining truck is provided as Figure 50,.

Figure 3.50 - Rimpull - GMW
(Caterpillar PHB 35, 2004 p9.41)



Part G. Driving and Braking Torque Effects On Grades

MPN 8, Part G, Mathematical Principles – Notes, appended in Volume 2, details the analysis basic to the following discussion. For a Cat 793C at 384 tonnes GMW on a 10% grade with 2% rolling resistance to realise an effective grade of 12% the required rimpull is approximately 12% of 384 tonnes = 46 tonnes. See line X – X on Figure 50 previous page. For downgrade hauling the braking effort will be approximately proportional to grade – but less the rolling resistance to realise the effective grade in the downhill case.

Reviewing the analysis in part E - indication of the driving/braking effect on wheel loads can be estimated as follows:

Hauling Loaded:	**Front Axle/Wheels**	**Rear Axle/Wheels**
Upgrade	-4%	+4%
Downgrade	+2.6%	-2.6%

Effect of rolling resistance, cumulative upgrade and deductive downgrade, is the reason for the difference between the above-tabulated effects on axle loads for upgrade and downgrade.

In summary:

For upgrade hauling, moderate increases in rear tyre loads will be experienced with complementary reductions in front wheel loads. Downgrade hauling induces significant increased front wheel loads with complementary reduction in rear wheel loads.

Drive torque for uphill hauling further increases rear wheel loading and conversely braking torque for downgrade hauling further increases front wheel loads.

Rolling resistance effectively increases the grade for upgrade hauling so increases dynamic force additions to rear wheel loads. The opposite effect applies to downhill hauling. For operations that have open pit configurations requiring significant downgrade-loaded hauling, haul ramp gradients, surfacing materials and finish should be given special consideration including reduced ramp grade for lower load transfer to front wheels.

A suitable reduction factor to be applied to up gradient to contain front axle loads on down gradients is 60%.

Ride-strut load sensing will be affected by load distribution variations for hauling on ramps. Payload measuring is not a practical requirement whilst hauling on ramps. But if ride strut sensing is used to measure payload, particularly to indicate when target payload has been achieved during ramping-down operations, special consideration will need to be given to output from Caterpillar's VIMS and similar systems supplied by other OEM. Generally static loads measured by ride struts on a 10% ramp should be in the order of 99.5% of measurements with the truck in normal level configuration. The effect of the incipient dynamic loads due to brake torque will complicate load measuring. Accuracy of strut-pressure sensing for load measuring is discussed in more detail below.

Payload Measuring Accuracy and Related Issues - Discussion

There is evidence that VIMS and equivalent load sensing systems from other OEM are affected by misplacement of payload in the body. Payload values indicated by VIMS or equivalents are not always consistent with scale weighings. VIMS *et al* are sensitive to both payload CG shifts and overloading. Analysis in Part D examined basic physical relationships between loads imposed on ride struts and mechanical systems that transfer loads from ride struts to axles and individual wheels. Intuitively ride strut pressure variations should provide a means of collectively measuring preloads and incremental payloads. This is the fundamental principle utilised for load measuring by VIMS *et al.* Analysis using simple statics verifies that, specifically for payload placed in the body, collective load increase on all ride struts determined from pressure variance must correspond to the total payload added to the body. Theoretically the total of strut load increases is independent of CG misplacement of payload in the body. But, as indicated in Part D there is a number of practical reasons why the simple theoretical model is inadequate in this case. There is substantial evidence that payload measurements using VIMS are sensitive to CG misplacement and overloading of trucks. It is likely that similar effects on accuracy of payload measuring systems supplied by other OEM will be experienced, albeit of differing effect depending on detail of suspension systems utilised.

Characteristics of payload measuring systems using suspension cylinder pressure sensing are discussed in terms of advice on Caterpillar's VIMS facility provided by David Rea from Caterpillar Global Mining (Rea D, personal communication, 2005).

VIMS accuracy is most dependent upon (in order of importance):

1. Correctly charged struts.
2. a. Flat loading surface for initial recording.

   b. Flat 0% grade for second gear reweigh recording.

   c. Correct axle distribution (33.3% front and 66.7% rear).
3. Correct payload (within 10/10/20 Policy guidelines).

The following summary is based on 3 different scales studies and 100+ loads.

VIMS absolute accuracy:

1. Is within 5% when the front axle is between 30.3% - 36.3% (Target 33.3%)
2. Is within 5% when payload is in the range 80% - 120% (target = 100%) so long as condition 1. is met.
3. Has always recorded light for payloads more than 10% overload.

The following Table 3.46 summarises some weighing studies and VIMS comparisons.

Table 3.46 Weighing v. VIMS Comparisons

				Arithmetic	Arithmetic	Front Wheel
				Mean	Mean	Mean
Site	Truck Model	Body	Material	VIMS error %	Overload %	Load %
A	793B	Dual Slope	Overburden	-2.09%	5.45%	30.63%
B1	793C	Duratray	Overburden	-2.20%	6.96%	34.13%
B2	793C	Flat floor	Overburden	-4.89%	-2.42%	35.92%
C	793C	MSD	Coal	0.26%	-15.83%	31.90%
D	793C	MSD	Overburden	-4.85%	6.11%	31.38%
E1	785C	Flat floor	Overburden	-0.76%	21.02%	33.50%
E2	789C	Flat floor	Overburden	1.01%	9.82%	33.81%

Site Legend:

Site A - 793B trucks hauling overburden - consistent overloading - max overload = 21.8%.

Site B1 - 793C trucks with DuraTray (third party) bodies in overburden with front axle bias and wide range of payloads.

Site B2 - 793C trucks with flat floor bodies in overburden with front axle bias and wide range of payloads.

Site C - 793C trucks with MSD bodies hauling coal with rear axle bias and consistent under loading - max under load = -26.7%.

Site D - 793C trucks with MSD bodies hauling overburden at a strip coal mine.

Site E1 - 785C trucks with flat floor bodies hauling overburden and gold ore.

Site E2 - 789C with flat floor bodies hauling overburden and gold ore.

Parts E and F indicate that, except where trucks are unnaturally restrained by severe wheel chocking and on ramp grades that loaded GMW will generally be distributed in the ranges of 33.3% front and 66.7% rear +/- 10%. This accommodates the VIMS front-axle distribution-criteria of range 30.3% to 36.3%.

Summarizing the empirical evidence – for actual payload in the range 80% to 120% of target payload: and provided the front axle distribution is in the range 30.3% to 36.3%, VIMS accuracy is in the range +/-5% for 95% of payloads.

The following reasons for VIMS payload accuracy varying between +/-5% are hypothesized:

- Response of suspension struts is affected by friction between seals and cylinder bore – (particularly noticeable when ride struts are new - Power A, pers. commun. 2005).
- When a truck is subjected to wheel chocking or on ramps, transverse loads will be developed across ride struts, particularly front struts that serve as steering king posts, increasing load on strut seals tending to reduce response of suspension cylinders to in-line loads.
- Front struts generally have greater travel than rear ride struts – so changing strut geometry; and, prospectively, application of transverse load components across struts.
- Rear ride struts generally are mounted to minimise effect of any transverse load components – described in more detail.

- It is hypothesized that front ride struts are subjected to relatively greater transverse force components – particularly where the strut acts as the steering king post.

Rear ride struts, and front ride struts of trailing link and double wishbone front suspension systems are commonly mounted using a double bush arrangement consisting a pin through a parallel-bore grease-lubricated bush with an outer spherical surface that matches up to two spherical half bushes, also grease lubricated, pressed with an interference fit into the mounting eye at each end of the strut. The end journals of the strut mounting pins are located and fixed within sacrificial bushes pressed into the mounting clevises with keeper plates to retain the pin in place longitudinally and prevent pin rotation. The above description is only one arrangement. Other designs may change in detail but the functional intention, to allow universal movement, will be similar.

The nett effect of the above-described mounting arrangement is to minimise transfer of transverse forces in any direction to rear ride struts or front struts with similar mounting details. Efficacy of the universal mounting system is dependent on:

- Efficiency of lubrication.
- Wear condition of the lubricated bushes that have a discrete life and occasionally will experience metal-to-metal contact.
- Auto-lubrication of all grease bearings is practically universal for large mining trucks; but grease systems do break down in part or totally from time to time.
- Bushes do wear and grease ways can be blocked or reduced in cross-section by wear.

Any or all of the above can contribute to spurious transverse load forces being transferred to ride struts.

Whatever causes contribute to variability in payload measuring by VIMS (or its equivalents), it has been empirically shown that, compared with scale weighing of mining trucks VIMS payload measuring can exhibit differences in the range +/-5%

in association with front-axle loading bias and/or overloading consequently resulting in lower-than scale-measured payloads being recorded by VIMS.

In addition to the causes identified, VIMS payload measuring is likely significantly affected by variability in loose density of material being loaded onto trucks and the consequent variable disposition of load in truck bodies. During the loading process, VIMS may indicate that payload is under target and the body is practically filled (due to lower-than-expected loose density). If loading continues additional “superimposed” load (see Part B) to make up target payload must be placed forward in the body so tending to bias load distribution to the front axle. In such circumstances empirical evidence indicates VIMS tends to record lower than actual payload. One reason for these hypothetical circumstances could be body capacity inadequate to accommodate target payload at loose density of material being hauled.

Selection of body type and capacity must, with high probability, accommodate the expected loose density range of material being hauled. There are several issues to be considered:

- If a range of material types - and loose densities, each with its unique distribution parameters - must be hauled, what determinative value of loose density should be used to design body capacity?
- What are the relative proportions of different material types – to determine whether to set body capacity to accommodate loose density for all material types (and with general excess body capacity for all but one specific type) or to accept under loading for relatively small proportions of one or more types of material?
- What distribution parameters apply to loose density of specific materials to be hauled to enable determination of a loose density value that will, with, say, 95% probability, ensure that body capacity is sufficient to accommodate target payload for each truckload.
- If VIMS indicates loading should be truncated before the truck body is filled volumetrically, potentially there may be load distribution bias either to front or rear depending on placement of the partial volumetric load within the body. There is obvious opportunity to overload trucks that will be avoided

by guidance from VIMS and high standards of loading equipment operating techniques.

Rear axle bias of loaded trucks is generally of less impact on individual wheel and tyre loads as indicated by Table 3.42, further illustrated by Figure 3.48 - upgrade hauling; and Figure 3.49- downgrade hauling. Much steeper curve gradients are manifest for front axle load increase as downgrade increases in Figure 3.49 compared with increase in rear axle loads, as upgrade increases, in Figure 3.48.

In the final analysis suspension-based load measuring systems such as TPMS/VIMS are the best payload monitoring option currently available. Compared with currently available volumetric payload measuring systems VIMS and equivalents have the following advantages:

- VIMS *et al* are onboard systems that can output to warning lights or digital displays externally on the truck, alternatively accessed on monitors by equipment operators, truck or loading, and any interested personnel in real time, both on site and at any location of choice serviced by modern communication mediums.
- Each truck is independently serviced by a facility non-intrusive to truck operation (does not dictate truck route or position at any time). Load recording is an adjunct to; and, apart from sharing data with other equipment management facilities, is independent of all other vital information recording systems and operational functions.
- Provides at-the-loading-point payload measuring facilitating payload control – available in real time to loading equipment operators and other interested parties.

Recent initiatives in volumetric sensing of payload volume in truck bodies, such as Trayscan (described in Section 3.3.6.), are located remote from the loading point on a haul road from pit to waste dump or ore processing. In large operations with multiple pit exits and several concurrently active haul roads multiple monitoring facilities (that can be mobile) will be required. As with suspension sensing load measuring, these types of systems do not interfere with the intrinsic productivity of trucks. Also, volumetric payload measuring is independent of the NMW and how that may change during hauling operations due to carryback and debris accumulated on the truck. But volumetric load determination will be dependent on volume of carryback unless unloaded trucks are scanned on return for loading. It is early days for volumetric payload measuring systems. The technique has potential for adaptation to loading point locations for incremental payload (bucket load) sensing and target payload control in volumetric terms.

Volumetric payload measuring to predict payload and facilitate measuring of body filling efficiency and payload location in the body likely has significant upside potential. Only time will reveal the extent and efficacy of volumetric sensing systems in control and management of hauling operations.

Review of VIMS for Bucket Load Analysis (Section 3.2.8)

Payload data, consisting 73 records, basic to the descriptive statistics in Table 3.12, were revisited. Payload distribution data for “Reweigh” (truck travelling in second transmission range – taken as the basis for comparison) were compared with “Static” data (stationary whilst loading). Differences between Static and Reweigh provide a third distribution. Descriptive statistics were derived for the three distributions as shown in the three left hand columns of Table 3.47.

Table 3.47 Payload Data for Comparison of VIMS Reweigh (Second Range) and Static (Loading)

			Differences				Less +/- 5%	
	Reweigh Payloads	Static Payloads	All	All	Less +5%	Less +/-5%	Reweigh Payloads	Static Payloads
	Calcs. In Tonnes			Calcs. In Percentage			Calcs. In Tonnes	
Number of Records	73	73	73	73	66	62	62	62
Maximum Value	259	270.5	18.6	8.18	4.87	4.87	259.00	270.50
Minimum Value	180.7	181.7	-32	-13.82	-13.82	-4.88	187.10	193.90
Range	78.3	88.8	50.6	22.01	18.69	9.75	71.90	76.60
Average of Range	219.85	226.1	-6.7	-2.82	-4.48	-0.01	223.05	232.20
Arithmetic Mean	225.38	226.81	1.43	0.69	0.08	0.70	227.40	228.95
Median	227.3	225.7	2.40	1.09	0.79	1.01	228.75	228.40
Variance	308.42	330.59	68.63	13.95	11.37	5.06	280.73	290.46
Std Dev Sample	17.56	18.18	8.28	3.74	3.37	2.25	16.75	17.04
Coefficient of Variation	0.078	0.080	N.A.	N.A.	N.A.	N.A.	0.074	0.074
Skewness	-0.16 To Left	0.01 To Right	-1.22 To Left	-1.11 To Left	-1.73 To Left	-0.34 To Left	-0.09 To Left	0.17 To Right
Kurtosis	-0.54 Platy-kurtic	-0.46 Platy-kurtic	3.31 Lepto-kurtic	3.03 Lepto-kurtic	4.43 Lepto-kurtic	-0.51 Platy-kurtic	-0.71 Platy-kurtic	-0.66 Platy-kurtic

Differences between Static and Reweigh were expressed as percentages. Qualitatively the Reweigh and Static payload distributions appeared to be symmetrical, normal distributions with some anomalous outliers. Using percentage differences as data, filtering of difference values outside the range +/-5% provided a residual 62 payload records. Descriptive statistics were derived for Reweigh and Static payloads corresponding to filtered, residual differences, some 85% of original data – as shown in the two right-hand columns of Table 3.47.

Descriptive statistics showed that:

- Reweigh and Static payload-data are similar, qualitatively exhibit central tendency; and distribution parameters indicate data sets are normally distributed.
- Difference-data sets also exhibit parameters indicating symmetry and normality.
- Residual payload distributions after filtering out records +/-5% of mean values, both Reweigh and Static, yielded distribution parameters with only small reductions in mean, standard deviation and coefficient of variation.

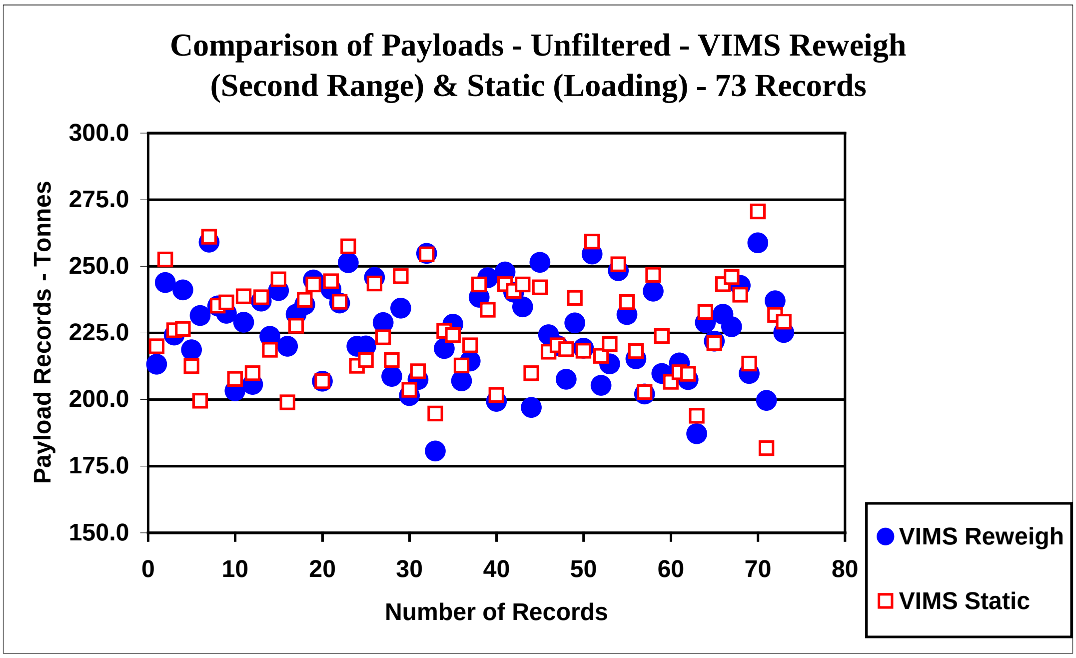


Figure 3.51 From Table 3.47

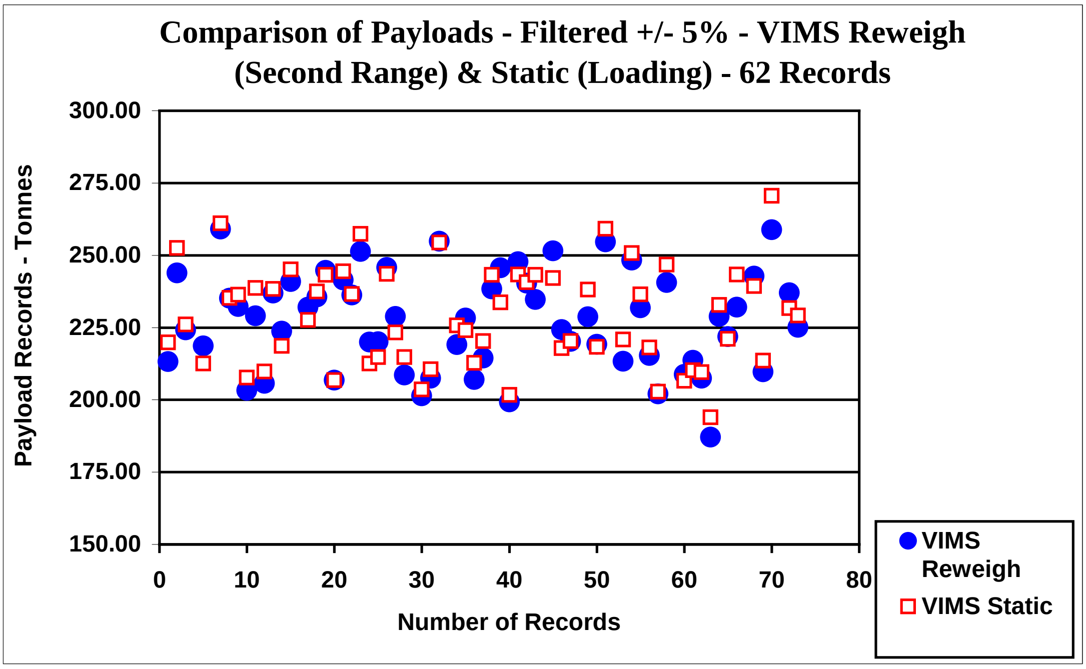


Figure 3.52 From Table 3.47

Distributions for both Reweigh and Static VIMS records are illustrated by comparative scatter diagrams in Figure 3.51 – unfiltered; and Figure 3.52 - filtered to +/-5% difference of Static compared to Re-weigh.

From the above discussion it is concluded that variability of VIMS records originates from several sources including:

1. External influences from the operating environment in which mining trucks work – such as effects from wheel chocking and ramps.

2. Misplacement of payload within the truck body, most likely due to loose density variability, possibly and/or operator error – considered non-intrinsic - i.e., an external influence.

3. Intrinsic mechanical effects including variable strut seal friction over seal life and degree of transverse loading across the centerline of suspension struts.

External influences from practical conditions are considered of greater effect compared with the hypothetical situations analysed in Parts E and F. Intrinsic effects are considered to be lesser causes of variation in VIMS (or equivalent) records compared with scale weighing.

In the final analysis, payload/GMW-measuring systems using suspension cylinder sensing deliver:

- Variable but acceptable accuracy at the loading point – variable to differing degrees depending on external affects not the least of which is variability of material loose density.
- More reliable and repeatable payload/GMW measurement by reweighing using a small sample of strut pressure recordings in second range of mechanical drive transmissions, or equivalent conditions for an electric wheel-motor mining truck.
- Typically VIMS and equivalents provide payload/GMW measurements that are generally repeatable and reliable within a reasonably limited range and with high probability.
- Accuracy of VIMS reweighing relative to scale weighing is claimed, with support of empirical evidence, to be +/-5% for 95% of truckloads measured.

Accuracy of VIMS Reweigh may be higher than claimed if the following are considered:

- Extreme inaccuracy values are for abnormal cases – where payload is misplaced or overloaded or other influences affect accuracy.
- Understood inaccuracy ranging to +/- 1% for in-ground scales is cumulative with any inaccuracy of VIMS.
- Recent and on-going technical improvements as keen competition for market share of large mining truck sales encourages OEM to provide customer ever-improved truck management systems.

A technologically advanced well-maintained on-board weighing system should generally deliver +/-2% accuracy (Power A, 2005, pers. commun.). Consideration of this advice is supportive particularly where trucks are loaded as per design and truck payloads are in compliance with the applicable control standard such as Caterpillar's "10/10/20 Policy".

For the single case of 73 records analysed as described, static payload measurement at the loading point compared with reweighing of the moving truck included individual payload variations in the range +8% to –14%. But as described and illustrated, descriptive statistics for the two distributions, static and re-weigh, were comparable. Particularly mean values were little different varying only by some 0.6%. This is interpreted as, at least initial, evidence that using VIMS for payload monitoring during loading will likely result in mean reweigh payloads that meet chosen control criteria - such as Caterpillar's 10/10/20 Policy. This of course, is with the proviso that the various factors influencing payload variability such as material condition, loose density variability and operational techniques are within design and best practice limits.

In Section 3.2.8 bucket loads included in small samples that make up truck payloads were downloaded from static loading data. Each bucket load recorded for each small sample of bucket loads accumulating to a truck payload was proportionally modified to convert static data to equivalent reweigh data. Based on the evidence of comparable descriptive statistics for static and reweigh data distributions; and considering that bucket load data were treated as a selection of distributions that were analysed and compared in terms of descriptive statistics, in hindsight it is concluded that proportional correction of static bucket loads to reweigh-based data was likely unnecessary. It is believed that analyzing bucket loads based on static data, as recorded, would have provided similar outcomes leading to identical interpretations and conclusions.

Descriptive statistics in Tables 3.15, 3.16, and 3.17 compare distributions of truck payloads, bucket loads, selected bucket load groups (first, last and intermediate) and sub-groups of four and five-pass bucket load/truck payload data. Measures of expected values (means), dispersion (standard deviation) and comparative variability (coefficient of variation) are generally consistent and reconcilable in terms of mathematical relationships.

It is concluded that the consistency and comparability of descriptive statistics of static and reweigh data is empirical evidence that data remotely downloaded from VIMS using Caterpillar's Minestar system, subsequent analysis discussed in Section 3.2.8 has resulted in generally valid interpretations and conclusions.

Interpretation, Conclusions and Comments

Earliest considerations in this Section 3.3.7 on "Truck Payload Centre of Gravity (CG) exposed a wide field of relevant issues. Each issue considered seemed to raise more questions than were answered. Consequently the issues expanded out to include consideration of:

- Body loading efficacy and implications of payload CG misplacement in distribution of NMW and truck payload as axle loads.
- Load transference between truck axles due to practical but abnormal operations including rear wheel chocking and uphill or downhill hauling.
- Leveraging affects from incipient or actual dynamic situations that are registered by truck suspension as additional "virtual load" – outcomes of analysis of these first three considerations are provided above or in Mathematical Principles - Notes appended in Volume 2.
- Review of intrinsic accuracy of payload measuring systems using suspension-cylinder pressure sensing.
- Review of reasons for output variability from such payload measuring systems.
- Brief comparative comments on volumetric payload measuring systems currently on offer and in process of development.
- Review of validity of using static and reweigh payload data in terms of individual bucket loads to examine the relationship between number of passes, bucket load anomalies and truck payload dispersion.
- Acknowledgement of the continual presence and significant influence of loose density variability in control and management of payload dispersion – a presence that grew in impact and importance in the course of the research.

Truck Selection and Productivity

Relevant issues in truck selection arising from this section 3.3.7 include:

- In the process of procurement, through contact with dealers and OEM supplying mining trucks, performance information and management systems will be offered – a typical example is Caterpillar's VIMS and its many integral and adjunct options.

- Integral with, or separable from, the total vital information system, on-board payload measuring using suspension cylinder pressure sensing will generally be offered.
- Due diligence in investigative stages of the procurement process must deliver an understanding of the benefits of payload measuring and what it promises for monitoring and management control of hauling operations.
- Body type and design for specific service – and, most importantly, adequate truck body capacity - that will avoid the too-often-seen "ducktail extension" and "side extensions" as untidy remedies. These afterthought, remedial extensions should not be confused with the addition of colloquially-termed "hungry boards" when owners, particularly civil construction and mining contactors, have deliberately sought to overload smaller construction trucks on typically short hauls.
- Particularly there needs to be an understanding of accuracy claims and what payload measuring systems can be expected to deliver.
- Implications in terms of management commitment to processing the wealth of data from payload measuring and related systems and compile it into acutely focused and user-friendly reports for the benefit of managing load and haul operations.

Effective use of systems acquired to monitor and manage hauling operations should be a natural progression from selection in the process of mining truck procurement. Specific issues include:

- Securing comprehensive records that are interpreted and compiled in user-friendly reports; and developing techniques to maximize productivity and equipment performance benefits from the systems.
- Accommodating within operational and maintenance management plans best practice for utilisation and maintenance of equipment, also for management and information systems, applying the same standards and importance as granted to critical operating componentry of the equipment.

As mining trucks increase in size and capacity there is ever increasing demand for sophistication and attention to operational control and management – particularly of payload dispersion, body designs, placement of payload in truck bodies, avoidance of unacceptable operational severity and misadventure. To satisfy this demand onboard information systems, VIMS and equivalents, are essential adjunct componentry for large mining trucks to realise the expected benefits from increased unit capital investment. It is not so much a matter of justifying adoption of onboard information, management and control systems for large mining trucks but justifying any decision not to adopt them.

3.3.8 Tyres

Pneumatic tyres fitted to mobile earthmoving equipment are generally treated as a consumable for purposes of cost estimating and budgeting. Tyre costs are normally expressed as unit cost per operating hour of the equipment or per unit of production – normally per tonne, but optionally per BCM. Unit cost of tyres is not directly comparable between operations and is essentially a local performance measure where productivity is measured in BCM.

But, in effect, tyres are an essential driveline component for equipment running on them. Compared with other driveline componentry tyres have a relatively short life. Tyre life for mining trucks is generally in the range 2,000 to 6,000 hour; or, say, 40,000 to 120,000 kilometres – with tyre life for larger mining trucks at the lower end of the range.

Tyre life, Costs and Truck Scale

As truck size increases tyre life tends to reduce as shown in Table 3.48; and tyre costs tend to increase as a proportion of total hauling unit costs. It will be noted that Tyre Life Index for larger mining trucks has substantial upside potential. This is consistent with industry history. As larger trucks have been developed and marketed, tyre manufacturers have responded with larger tyres designed for the increased service. Tyre development tends to be delayed compared with other product improvements for mining trucks. It involves substantial field-testing and experience. Onsite management of larger tyres by operators also needs to be more diligent. To provide a basis for improving tyre life, road surfaces, general running conditions, tyre pressure monitoring and tyre record keeping need to be best practice.

Notes on Table 3.48:

1. Based on Michelin XDR E4 tyres unless indicated otherwise.
2. Advice received March 2004 based on prior experience. Subsequent and recent sharp escalation of oil price and worldwide shortage of earthmover tyres should be considered.
3. Tyre life indices for the largest trucks (+300 tonnes payload) can be expected to plateau about 55% to 60% - interpreted from industry sources.

Table 3.48 Tyre Life Indices
(Cutler#2, 2004)

Caterpillar Truck	Nominal Payload Tonnes	Tyre Specification	Tyre Life Index %	Comments
773E	54	2400R35 XKD1	*100*	Estimate based on limited data
777D	90	2700R49	**100**	
785C	140	3300R51	**90**	
789C	180	3700R57	**90**	
793C	220	4000R57	**80**	
797A	350	59/80R63	**55**	Initially 40%

Considering a reverse view to the discussion, tyre costs will reflect tyre life. Table 3.49 indicates tyre costs as a proportion of operating costs.

Table 3.49 - Tyre Costs as a Proportion of Hauling Costs

Source	Date	Proportion % Including Depreciation	Proportion % Direct Only	Proportion % Tyres + Fuel + Operator	Comments
Gregory (1)	2002	13	17.5	25.5	Mean value over range of truck scale and application
Cutler (2)	2002	NA	NA	32 for 793 (4) 54 for 797 (5)	Chile operation with trucks running side by side
Kirk (3)	2000	23.5	32	38	For large trucks. Inferred from tyres as 9% of total mining costs

Notes:

1. (Gregory, 2002)
2. (Cutler, 2002)
3. (Kirk, 2000)
4. Caterpillar 793C
5. Caterpillar 797A

Reconciliation of cost proportions from the three independent sources is reasonably consistent. Data from Gregory is for a range of trucks – not limited to the larger mining trucks – and can be reasonably reconciled with Kirk by increasing all proportions by some 80% to reflect the significantly reduced life of largest truck tyres. After this reconciliation adjustment the tyre-cost proportion of total of Tyres Fuel and Operator is of the same order from the three sources.

Recent prices for tyres have become volatile. Together with the steep learning curve experienced by manufacturers in design, production and application of the largest truck tyres, unit cost of tyres has become a significant component of cost of haulage. The volatility of tyre prices is reflected by Table 3.49 where, at the times indicated, tyre costs represented:

- For the largest trucks - 220 tonne and above - 30 to 50% of the total costs of operating labour, fuel and tyres;
- Some 20 % to 30% of the cost of hauling; and
- Some 5% to 9% of total open pit mining costs.

At the time of writing this thesis large earthmover radial tyres are:

- Generally unavailable to new customers and in short supply to high-user, established customers with existing supply contracts.
- Being substituted with bias tyres from non-traditional sources (Power, Anthony, personal communication 2005).

Also currently large earthmover tyres are:

- Commanding spot sales prices to 200% of the normal supply price.
- Have been escalating over the more recent years at some 5% per annum until 18 months ago.
- In the most recent 18 months there has been a discrete increase of 25% to 30% in tyre prices to existing customers.
- Future prognosis for cost escalation for the next two years or more is in the range 8% to 10% per year – until oil prices settle to a long term tend and world wide shortage of earth mover tyres is alleviated, perhaps by a necessary flattening of demand for commodities produced by open pit mining..

A reasonable consolidation of the above information is that from 2002 tyre costs have increased some 40%. This is considered a reasonable increase for estimating purposes and updating historical cost proportions.

That tyre costs increase significantly with truck size is obvious from the discussion. Tyre costs are further discussed in Section 4.3.

Tyre Wear Rates and Failures

Tyre wear rates are affected by road gradient as indicated in Table 3.50. It will be noted that the wear rates are substantially higher than the axle load increases calculated and illustrated in Tables 3.44, 3.45, and Figures 3.48, 3.49. This increased wear rate is illustrated by Figure 53. It will be noted that wear rate is presented in Figure 53 as 150% for 130% load on tyres with a close-to-linear relationship.

The wear rate indices in Table 3.50 are for drive axles. By allowing for one-third of wear on front tyres the gradient-effect on tyre life can be analysed as shown in Table 3.51:

Table 3.50 – Affect of Gradient On Tyre Wear
(Michelin Presentation)

Slope	Uphill Laden	Downhill Laden
8%	100	195
10%	136	292
12%	183	426

)

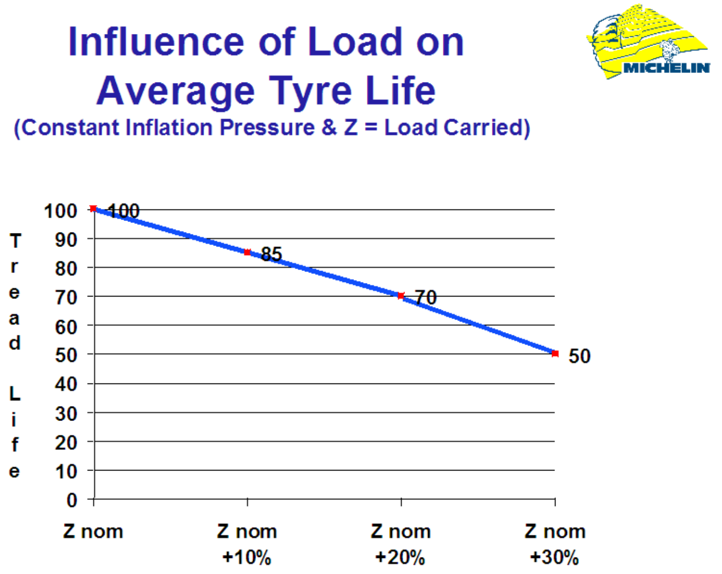


Figure 3.53 – Tyre Wear Rate and Tyre Load
(Michelin technical presentation)

Table 3.51 Comparative Tyre Life v. Road Gradient
(Michelin technical presentation)

Slope	Uphill Laden	Downhill Laden
8%	100	61
10%	81	44
12%	64	32

In practice, steep gradients loaded are only a proportion of the haul distance. So the high wear rates indicate in Tables 3.50 and 3.51 need to be discounted accordingly. Obviously accelerated tyre wear will be experienced in deep open pits with steep ramps.

Tyre Failure Mode and Setting Tyre Specification

The cost of accelerated tyre wear rates is substantially overshadowed by premature failure before tyres wear out. Tyres that survive to actual discard due to wear are generally in the minority. Relative distribution of reasons for replacement of tyres at a "world-class metal mine" in Table 3.52 illustrates this point.

Table 3.52 – Failure Modes for Tyres
Caterpillar Forward Mining Seminar 2001

Failure Mode	Cut	Impact	Separation (Heat)	Wear	Other	Total
Proportion of Tyres %	**45**	**29**	**11**	**7**	**8**	**100**

The above evidence from a single operation is not necessarily typical for all open pit mines. The proportion of each mode of failure is dependent on road surfaces, road maintenance – particularly presence of support equipment - not only to keep road geometry and running surface in order, but also to clear spillage and debris from roads; also operator care during operations to avoid potential tyre damage. Data on tyre life and costs are necessarily based on actual tyre life regardless of the mode of failure. There are generally cost benefits from investment in road improvements and best practice maintenance for all open pit operations. Particularly in early years of the project road maintenance must achieve best practice standards. As open pits develop demands on tyre performance steadily increase; so road maintenance becomes more critical. Also, as haul distances increase, tyre composition needs to be tuned to the increasing tonne-kilometres-per-hour (TKPH) requirements of the operation.

For the purposes of equipment selection, particularly trucks, it will be necessary to understand tyre requirements for the operation, site conditions and petrography of the stratigraphy to be mined. Collecting data from experience at similar sites in-house, or operated by others, is important as a guide for setting tyre specifications, particularly for trucks; but also for other mobile equipment running on tyres. Competing tyre suppliers, dealers and OEM offering mining trucks can be of substantial assistance with establishing an initial tyre specification. There are a number of specialist earthmover tyre service and management groups with experience that can contribute significantly during equipment selection investigation. Tyre maintenance and service, with or without supply of tyres, is commonly outsourced by open pit mine operators and mining contractors. It is necessary during due diligence and investigation to seek the benefit of specialist experience. Exploring and commercially examining the option of a tyre service contract, parallel with a maintain-and-repair-contract (MARC) for general equipment maintenance (discussed in Section 3.4), needs be included in due diligence for selection and procurement of mining equipment.

Summary – Equipment Selection and Productivity Issues

Criteria for equipment selection relating to tyres are relatively simple:

- Due diligence in the investigative phase of procurement of mining trucks and other mobile equipment running on tyres should effectively detect and quantify any significant tyre life/cost issues.
- Tyre manufacturers (generally practically limited to two or three for large earthmover radial tyres) provide advisory services at the investigative stage.
- Experience, advice and service contracts offered by a number of tyre service and management groups is a valuable source of tyre performance and cost information – as well as providing the option of tyre service and management contracts as an adjunct to equipment selection, particularly trucks.

- It is essential to determine the nature of the tyre rating recommended by the suppliers to relate to the road construction specifications in relation to road geometry and running surface finish.
- The larger the scale of mining trucks, the more demanding will be the standards of haul road construction, surface finish and ongoing maintenance.
- Impact of tyres on performance and cost increases with the scale of trucks and increased haul distance – for the largest trucks operational performance limitations controlled by tyres - may be most significant factors in determining productivity and cost criteria for determination of project economics.

Failure to align tyre specification with service and adopted road standards will prevent realisation of expected tyre costs.

Accelerated tyre wear due to open pit designs with steep ramps, albeit an economic necessity, will also mean more downtime for tyre replacement so reduced productivity – a relatively moderate, but important issue in terms of load-and-haul productivity and cost.

Truck-performance effects due to tyre wear are further addressed in Section 3.3.11.

3.3.9 Bucket Passes and Payloads

Introduction

Analysis of bucket loads and resulting payloads in Section 3.2.8 arrived at significant conclusions including:

- As expected bucket loads tend to be normally distributed with truck payloads exhibiting greater central tendency.
- Means of first bucket loads are manifestly higher and means of last bucket loads generally lower than means of the population for bucket loads.
- Coefficient of variation (CV) of bucket loads or truck payloads is a convenient statistic to compare sub-populations from a range of loading operations.

- Caterpillar’s 10/10/20 Policy was related to the truck payload CV statistic as a comparative compliance criterion.

Further analysis in Section 3.3.6 introduced “digability” as the measure of loading productivity. The significant relationship between variability of voids ratio of material to be loaded and hauled and truck payload variability was developed analytically. The most important direct relationship between drilling and blasting outcomes, in terms of fragmentation and size/shape grading and efficacy of subsequent load and haul activities was discussed and emphasized.

In this Section 3.3.9 a number of load and haul operations are analysed and compared using a suite of descriptive statistics to show that the hypotheses and theories developed in the initial phases of the research are consistent with operating experience.

Discussions in this section extend to consideration of the number of passes and implications for:

- Truck payload consistency
- Loading time consistency
- Variability of truck loading time and truck trip times.

The benefit from achieving acceptable operating “rhythm” in load and haul operations is discussed with the ultimate objective of cyclical loading and hauling operations simulating a continuous transport system as closely as possible. Protocols for load and haul practice with the ultimate objective of achieving acceptable “rhythm are recommended.

In the process of selecting load and haul equipment, the protocols recommended in the discussion to follow are prerequisites for ensuring realization of productivity expectations.

Load and Haul Case Studies - Statistics

Load and haul data from three open-pit mining operations were analyzed, specifically:

Case 1 2,632 records from an open pit gold mine using nominal 225 tonne trucks

Case 2 3,208 records from an open pit iron ore mine using nominal 177 tonne trucks

Case 3 1,545 records from a mineral sands mining operation using nominal 135 tonne trucks

All data was downloaded from VIMS management systems of Caterpillar trucks.

Data used for statistical analysis of bucket cycle and truck-loading times in 3.2.9 unfortunately did not include payload data. But the data and outcomes of 3.2.9 have provided a valuable contribution to analysis of the relative merits of additional passes to ensure a full payload and foregoing those passes with the benefit of improved rhythm and potentially reduced bunching effects.

Descriptive statistics for Case 1 are provided in Table 3.53; and for Cases 2 and 3 in Table 3.54.

Table 3.53 Case Study 1 - Comparative Descriptive Statistics for Payloads and Passes

	CASE 1									
			Number of Passes per Payload							
Description	**All Payloads**	**Passes per Payload**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
Number of Records	2,632	2,632	165	1659	547	99	44	25	29	64
Maximum Value	283.50	10	259.60	274.20	283.50	273.20	257.70	252.40	259.40	246.00
Minimum Value	161.40	3	171.60	172.00	161.40	177.20	202.10	181.90	182.00	209.80
Range	122.10	7	88.00	102.20	122.10	96.00	55.60	70.50	77.40	36.20
Ave of Range	222.45	6.5	215.60	223.10	222.45	225.20	229.90	217.15	220.70	227.90
Ave of Sample (Arith. Mean)	225.09	4.51	220.84	224.06	228.89	225.43	226.44	225.94	226.14	228.28
Mode	220.50	4								
Median	225.20	4	220.50	224.10	230.30	227.20	227.40	224.40	224.70	228.55
Variance	232.46	1.62	246.19	214.11	293.29	221.49	151.21	210.86	182.26	67.86
Std Dev Sample	15.25	1.27	15.69	14.63	17.13	14.88	12.30	14.52	13.50	8.24
Coefficient of Variation	**0.068**	**0.282**	**0.071**	**0.065**	**0.075**	**0.066**	**0.054**	**0.064**	**0.060**	**0.036**
Skewness	-0.15 To Left	2.70 To Right	-0.31 To Left	-0.14 To Left	-0.24 To Left	-0.22 To Left	0.26 To Right	-0.89 To Left	-0.66 To Left	-0.15 To Left
Kurtosis	0.59 Lepto-kurtic	8.22 Lepto-kurtic	0.23 Lepto-kurtic	0.39 Lepto-kurtic	0.65 Lepto-kurtic	1.23 Lepto-kurtic	0.28 Lepto-kurtic	2.67 Lepto-kurtic	3.71 Lepto-kurtic	-0.59 Lepto-kurtic

Table 3.54 Case Studies 2 & 3 - Comparative Descriptive Statistics for Payloads and Passes

Description	CASE 2A All Pay-loads	CASE 2A Passes per Payload	CASE 2B All Pay-loads	CASE 2B Passes per Payload	CASE 2C All Pay-loads	CASE 2C Passes per Payload	CASE 3 All Pay-loads	CASE 3 Passes per Payload
Number of Records	3207	3207	3189	3189	3004	3004	1545	1545
Maximum Value	237.80	10	237.8	10	237.8	10	207.10	8
Minimum Value	109.60	3	109.6	4	117.9	5	97.30	3
Range	128.20	7	128.2	6	119.9	5	109.80	5
Ave of Range	173.70	6.5	173.7	7	177.85	7.5	152.20	5.5
Ave of Sample (Arith. Mean)	180.63	5.90	180.70	5.92	181.75	6.04	138.40	4.05
Mode	177.50	6	177.5	6	177.5	6	140.40	4
Median	181.10	6	181.2	6	182.3	6	138.50	4
Variance	278.19	1.07	277.15	1.03	260.38	0.85	157.50	0.74
Std Dev Sample	16.68	1.03	16.65	1.01	16.14	0.92	12.55	0.86
Coefficient of Variation	0.096	0.159	0.096	0.145	0.091	0.123	0.091	0.212
Skewness	-0.20 To Left	0.38 To Right	-0.20 To Left	0.49 To Right	-0.16 To Left	0.87 To Right	0.15 To Right	1.39 To Right
Kurtosis	0.31 Lepto-kurtic	0.82 Lepto-kurtic	0.31 Lepto-kurtic	0.74 Lepto-kurtic	0.26 Lepto-kurtic	1.10 Lepto-kurtic	1.17 Lepto-kurtic	3.02 Lepto-kurtic

Data as received consisted:

Case 1 – 2,641 records

Case 2 – 3,209 records

Case 3 – 2,048 records

The data sets were examined for anomalous records. Truck payloads for all 1 and 2 pass records were obviously too high so these records were deleted.

For Case 1 only, the 2-pass loads were filtered out to leave a residual of 2,632 records.

All 10-pass loads in Case 2 were examined. One record with low payload for 10 passes was deleted. Case 2 was separated into three Cases 2A, 2B and 2C filtered to 3-pass plus, 4-pass plus and 5-pass plus respectively on the basis of anomalously low loading times and high average bucket loads.

Case 3 was filtered to 3-pass plus payloads.

From Table 3.54 it appears that, for Cases 2 and 3, truck payload CV are in excess of the Caterpillar 10/10/20 Policy. Correlation with comparative scale weighing studies would be needed to confirm non-compliance. This is not a reflection on the accuracy of VIMS/TPMS. High truck payload CV values in Cases 2 and 3 are not a reflection on management of the operation. The data from the three operations was recorded more than three years ago and was a snapshot at that time. Operating practices likely have changed over time and management/maintenance of VIMS and recording/reporting have likely improved.

Case 1

As shown in Table 3.53, descriptive statistics, particularly mean payload and CV are consistent with the analysis and discussion in Sections 3.2.8 and 3.3.6. Non-parametric Kolmogorov-Smirnov (K-S) analysis results for the 2,632 payload records (copy in Distribution Testing, Volume 2, Appendices) confirm that the payload data is normally distributed.

Figure 3.54 provides a histogram of the 2,632 records with a normal distribution curve for comparison.

The data was sorted to provide sub-samples of payloads of 3,4 through 10 passes and descriptive statistics derived for each sub-sample, particularly payload CV.

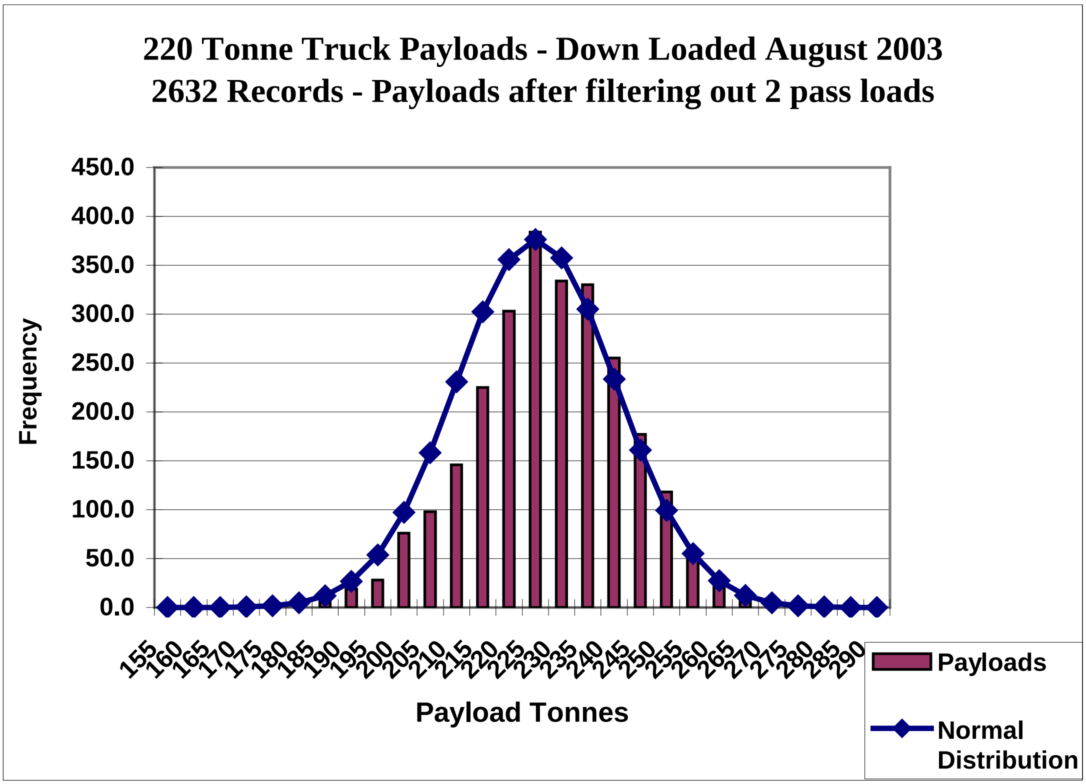


Figure 3.54 - Related to Statistics of Table 3.53

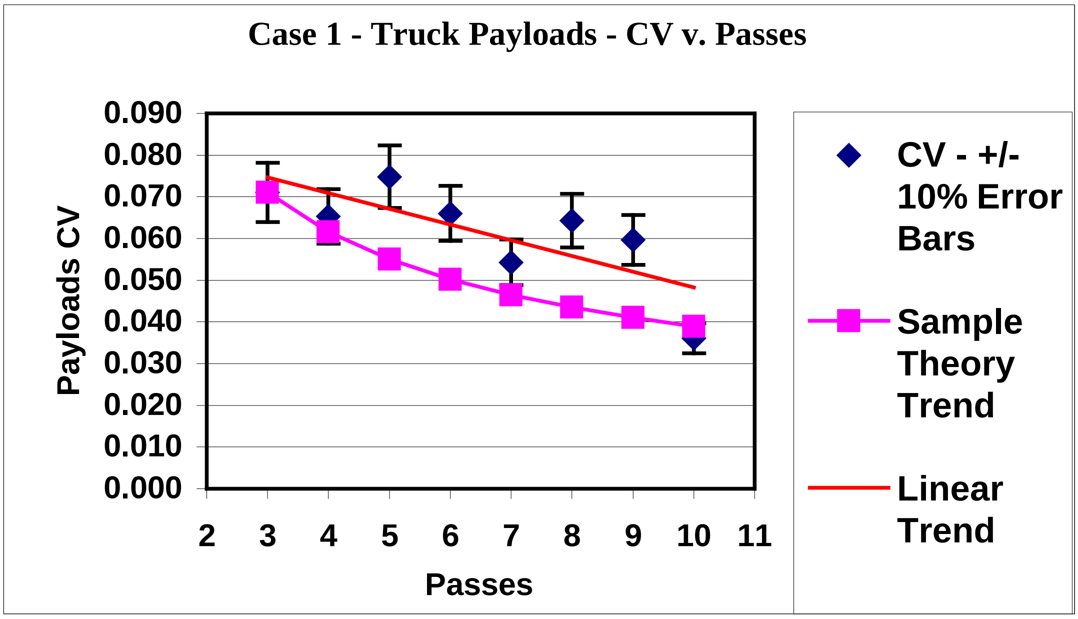


Figure 3.55 From Table 3.53

The CV values were plotted against passes as shown in Figure 3.55 against a trend line and theoretical values of diminishing values of CV with number of passes as discussed in Section 3.2.9, illustrated by Figure 3.27, and developed in Item MPN

3.3, particularly equation (10), in Mathematical Principles - Notes appended in Volume 2. CV values were provided with +/- 10% error bars. Figure 55 shows the expected reducing trend with increasing passes but correlation is only moderate.

Centralizing tendency of increasing passes is not the only driver of variability of payloads. As discussed in Section 3.3.6 condition of material to be loaded and hauled significantly influences truck payload variability.

Passes per truck payload in Case 1 are also consistent with the expected mode = median < mean relationship. It should be noted that the number of passes per truck payload is a result of two differing-capacity loading machines. Passes 3 to 7 relate to 700 tonne hydraulic shovels. Passes 8 to 10 relate to 190 tonne wheel loader. If the mean number of passes is limited to passes 3 to 7, the average number of passes is reduced from 4.51 to 4.28 generated as the mean from weighting as per indicated percentage proportions below.

If the 6th and 7th passes are sacrificed then the average number of passes is reduced to 4.16; also generated by weighting as per adjusted proportions. The benefit by way of improved hauling "rhythm" is obvious.

Table 3.55 Distributions of Bucket Passes/Truck Payload

Passes	**%**	**%**
3	6.6	7.0
4	66.0	70.0
5	21.8	21.0
6	3.9	
7	1.7	
Total %	100.0	100.0
Mean Passes	4.28	4.16
Mode/Mean %	93.5	96.2

It will be noted that the mode/mean ratio for number of passes is directly related to the mode/mean ratio for loading times – termed Productivity Quotient PQ_T generically for bucket cycle of truck loading times - as discussed in Section 3.2.9. So a Passes Quotient – PQ_P say – could be used as a measure/monitor of dispersion of passes that is interrelated with hauling "rhythm" and the issue of foregoing additional passes as discussed in this section. Adoption of PQ_P could be considered over-complicating. Use of the simple familiar average number of passes as a KPI is likely just as effective and convenient.

Simply – if the actual mean number of passes is less than or equal to the selected notional mean number of passes for loading a truck built into the loading and hauling equipment selection criteria, say, 105% of the modal value, then likely there is no need to look further. It the mean number of passes is greater than the planned mean number of passes i.e., say, >105% of modal value - then records need to be examined for additional passes that may be sacrificed cost-beneficially to realize the expected standard of operational "rhythm" inherent in the selection criteria.

Productivity and cost implications of foregoing additional passes for marginal top up of truck payloads are discussed.

Productivity and Sacrificing Passes

To justify an additional pass or passes, production lost by sacrificing one or more additional passes with the result of an incompletely filled truck body must be more than the truck production gained by reducing delay to all trucks in the hauling fleet.

The above statement can be expressed as:

$$PL_X > PL_L \qquad \text{(A)}$$

PL_X = Payloads proportion gained - by foregoing additional pass(s)

PL_L = Payload proportion lost.

Simplified equation (A) is developed in detail in MPN 9 in Mathematical Principles – Notes, appended in Volume 2. The analysis in this section is in terms of production alone without consideration of productivity v. payload variability or cost implications. Cost analysis related to bucket-pass sacrifice is provided in Section 5.4.

Productivity relationship with variable payload of mining trucks is analysed and interpreted in Section 3.3.10. The further steps to connect driveline component life and fuel burn with payload variability and to analyse cost implications are dealt with in Section 5.3. Table 3.57 is intended to provide guidance by indicating, in terms of production, when an extra bucket load should be sacrificed to benefit overall productivity and "rhythm" of load and haul operations.

Bucket pass sacrifice tends to become more justifiable as:

- Pass to be sacrificed trends downwards, i.e., sacrificing the 8^{th} pass for 6-pass mode loading is more justifiable than the 7^{th} for 5-pass mode that is, in turn more justifiable than the 6^{th} pass for 4-pass mode loading.
- Bucket cycle time T_{BC} increases.
- The number of trucks time-benefiting from bucket-pass sacrifice increase.
- The truck trip time T_T decreases.

Sacrificing more than one bucket load, if practical, could also be justified provided the truck payloads proportion gained is favourable when compared with production proportion of payloads lost by sacrifice.

Validity of Bucket Load Sacrifice Criteria

In terms of production alone, the above justification criteria:

1. Will be valid when loading equipment appears to be over trucked – a common case where maximum shovel productivity is the focus with acceptance of the resulting haulage cost premium – the "maximum-production-with-mining cost-penalty" case.
2. Will likely be valid generally when loading equipment and trucks appear to be exactly matched based on mean values for operating criteria – the hypothetical case that rarely if ever happens, and when it does the experience is fleeting; also the maximum bunching case, where variability in truck loading and trip time components has maximum effect and bunching productivity loss is a maximum.
3. Generally invalid where loading equipment appears to be nearly always waiting for trucks – an additional bucket load or loads will reduce shovel/loader waiting time – could add productivity at marginal load and haul cost without affecting other trucks and shovels in the combined fleets; but also, theoretically, could cause a subsequent delay of all trucks in a future bunching occurrence. This so-called "undertrucking" – the author prefers "over-shoveling" (OS) - is likely the lowest cost option with trucks working at maximum efficiency and loading equipment underutilised.

The expressions "appear(s) to be" used in the three cases are deliberately precautionary. As discussed in more detail in Section 5.4, any operating group of loading and hauling equipment may instantaneously appear to be any one of the three cases. But mean parameters of sampling observations of the production group may be consistently specifically one of the three cases.

Case 3. – the OS case - is the practical operating circumstance that is most complex and likely more appropriately should be modelled and subjected to simulation to determine outcomes.

The above analysis and criteria are relatively rudimentary and limited to production considerations alone. They are considered of sufficient accuracy for initial estimating and for setting of standards for equipment selection; but always if matching of trucks and loading equipment and the alternatives of over-trucking or over-shoveling are concurrently considered.

Bucket sacrifice justification in terms of haul costs is revisited in Section 5.4 together with further discussion on criteria for application.

The above-developed criteria have interrelationship with bunching/queuing effects. Sacrificing additional bucket loads to fill a truck body when localized low bucket-fill factors are experienced will tend to reduce productivity discounting due to bunching/queuing.

In Section 3.2.9 bucket cycle times and truck loading times were analysed in some detail for an operation where the bucket fill factor, on the evidence of VIMS data, at that time, was a relatively low 0.65. Some 13% of bucket passes were 7th pass or more. Allowing for a reduced bucket load of some 75% of the mean of bucket loads, there was potentially justification for truncating loading operations at 6 passes.

Table 3.17 indicates that last loads are substantially reduced compared with mean of all bucket loads and mean of intermediate bucket loads (excluding first and last passes). Adoption of a discount factor on last bucket loads for the analysis to follow is considered reasonable – 0.75 is indicated as an acceptable default discount factor.

Practitioners and researchers can reproduce Table 3.57, appended in Volume 2, from item MPN 9 in Mathematical Principles – Notes, also appended in Volume 2. For convenience a copy of the spreadsheet file including Table 3.57 is on the CD in the pocket inside the back cover of this thesis document.

Reduced Payload and Productivity

Hanby, in Part 3 of his three-part paper, has analyzed the affects on productivity and unit haulage cost as a result of reducing truck payload (Hanby, 1991). Relationships between truck mass and dynamic performance parameters - velocity, acceleration, time and distance - were developed. Hanby chose the quotient - travelling time/percentage payload - to represent haulage cost per unit volume or mass as convenient so assuming a linear relationship with unit hauling for both truck trip time and payload. Hanby's choice of haulage cost parameter is valid only within the limitations of his analysis.

The assumptions and limitations adopted by Hanby included:

- Individual comparative range of cases each of specific total resistance.
- A single mining truck hauling constant distance at that specific resistance; ≈.to one-way haul over a single haul course segment.
- Over a range of payloads from 30% to 170%
- Calculation of time for the selected distance.
- Derived haulage unit cost indicator as the quotient - time / % payload.

Hauling unit costs can be considered a linear function of truck trip time – but only when compared for limited hypothetical cases where all truck payloads are mean, where all trucks have exactly the same GMW : NMW ratio; and performance and haul distance is the same for all cases – limitations that generally apply to Hanby's analysis.

But unit haul costs for variable payload are measurably non-linear. Caterpillar has observed that driveline component life decreases more, and non-linearly, for overloads than increased life is realised for under loading. The author has analysed and discussed cost implications of this non-linear payload - maintenance cost relationship in previous research (Hardy#1, 2003). This non-linear relationship is further analyzed and discussed in more detail in Sections 3.3.10 and 5.3. Unit cost for haulage per unit of distance in practical truck cycles varies with one-way haul distance; but the effect would be small except for the "fixed" time events with practically fixed cost that is a smaller proportion of total unit costs as one-way hauls increase. So unit cost of haulage per unit time varies as one-way hauls vary, but not linearly, due to changing proportion of so-called “fixed” time included in truck trip time – a factor excluded by limitations of Hanby’s analysis.

Notwithstanding the above comments on Hanby’s cost-indicator quotient and the assumption of linear cost – time; and cost – payload relationships, the outcomes of Hanby’s research, limited as discussed, provides valuable insight that:

- Up to a total resistance of 24% it appears to be financially disadvantageous to operate a mining truck with generally reduced payload – as total resistance in practical situations is unlikely to approach 24% by a substantial margin, this outcome applies generally in practice.
- Most importantly, generally overloading trucks is likely to be counter-productive, not realise any significant cost benefit, and ultimately could be unsafe (Hanby, 1991).

The outcomes of Hanby’s research provide a foundation and opportunity to further investigate the increased complexity of actual load and haul operations.

In this section the interrelationship between numbers of trucks being serviced by loading equipment has been analysed in terms of sacrificing relatively small portions of truck payload to gain additional truck payloads. It is hypothesized there are circumstances where extending the number of bucket passes to realise full payloads is not cost beneficial. Analysis and interpretation in this section has tested the hypothesis and has established that, for favourable operating circumstances, sacrifice of additional bucket passes will be justified – particularly where loading equipment is over trucked. This hypothesis is further discussed in Section 5.4 in terms of comparative costs. Certainly planning prior to equipment selection needs to include definition of the loading and hauling operating circumstances prior to setting service requirements of loading equipment and mining trucks.

Interpretation and Summary

This section has examined case studies of substantial truck payload data samples from operations using differing size of mining trucks, loading equipment to load and haul and different materials. The analysis and concepts developed in Sections 3.2.8 and 3.3.6 have been generally verified by the case studies.

Sacrificing extra passes to top up trucks to full loads to realize the benefit of improved productivity load and haul "rhythm" was also analyzed. A case was made to sacrifice additional bucket loads in favourable circumstances including:

- Where the truck fleet appears to be operated with loading equipment over trucked or close to "matched" – there may be a case where the truck fleet appears to be over-shoveled (= under trucked) but this circumstance needs to be modelled for simulation with appropriate recognition of stochastic parameters.
- Where production lost by pass sacrifice is compensated by increased productivity of the truck fleet by time saving.

Analysis of the sacrifice productivity trade-off has been summarized in a relatively simple formula that has been interpreted in a selection table – Table 3.57 - appended in Volume 2 – to provide a decision facility.

The importance of modal values of bucket passes became obvious in the course of analysis. A concept of a Productivity Quotient PQ_P, consisting passes mode/mean is discussed above similar to the concept of PQ_T to monitor bucket cycle or truck-loading times as discussed in Section 3.2.9. The alternative to the concept of PQ_P, i.e., establishing the relationship of mean of actual passes to planned mean of passes, is likely as convenient and effective.

Estimators building up productivity data for the purposes of testing feasibility, with flow on to criteria for equipment selection; also for compiling operating budgets must be aware that when four, five or more passes are adopted as a deterministic design parameter they are really selecting the modal value. Chou provides: "The 'mode' is often the concept in most people's minds when they speak of 'averages'." (Chou, 1969).

There must always be realization that, by selecting, say, four-pass loading that actual truck loading times will reflect more than 4 x mean bucket cycle time. Suggested values of mean number of passes for "best practice" and "less than best practice", i.e., a\measures of loading time quality, are shown in Table 3.56.

Table 3.56 Comparative Measures of Loading Practice Quality

Nominal Passes	Mean Passes	PQ_P %	Comment
4	4.2	95	Best practice
	4.5	90	Average practice
	5.0	80	Manifest improvement upside
5	5.25	95	Best practice
	5.56	90	Average practice
	6.25	80	Manifest improvement upside

The above concepts and criteria need to be recognized in the process of planning and developing production estimates as preparation for load and haul equipment selection for open pit mining.

An intuitive check that has stood the test of time in reviewing productivity estimates based on proposed equipment is the number of passes included in truck loading time. Where the number of passes is a round digit - and there is no transparent allowance for mean passes to exceed the selected modal value – Beware!

The author's intuitive awareness of the need for loading time to reflect mean passes, not modal values, and the payload-dispersion reducing effect of increased passes was the basis for a paper entitled "Four-Pass Loading – Must Have or Myth" included in Supplementary Information in Volume 2 (Hardy#1, 2003). Revisiting the research for that paper has substantially extended the analysis and understanding of the issues as described herein.

3.3.10 Truck Performance and Productivity

Introduction

This section of the research moves on to analyze truck trip times in terms of four time components in two general categories:

Fixed:

1. Manoeuvre-and-spot (T_S) for loading
2. Manoeuvre-and-dump (T_D)

Variable:

3. Loading (T_L)
4. Travel (T_V)

Accumulation of the four components is commonly termed "truck cycle time"; but is termed herein as "truck trip time" T_T.

In preparing for equipment selection, productivity estimates are dependent on truck trip time T_T.

Although traditionally designated as "fixed" both T_S and T_D are, in reality, alike with all time components, variable — intrinsically, a function of equipment capability; and non-intrinsically due to extraneous events that cause extraordinary time losses. Variability of loading time was analyzed and discussed in Section 3.2.9. Travel time variability is analyzed below.

The four components and truck trip time are discussed in turn.

1. Manoeuvre and Spot Time - T_S

Mean T_S times for large mining trucks are suggested as:

- "Usually between 0.4 to 0.7 minutes" (Hays, 1990)
- "Typically between 0.6 to 0.8 minutes" (Caterpillar#1, 2004)

The more recent publication and commercial responsibility attached thereto are likely significant. The author's experience is that T_S time of 0.75 minutes – intrinsic time only – for large trucks to 220 tonne nominal payload is a reasonable typical value.

In Section 3.2.9 T_S-time is conveniently analyzed in two parts:

1. The major proportion of T_S time from departure of the previously loaded truck and the waiting truck is clear to manoeuvre through the turn and reverse until the shovel is in the spot position ready to dump – the time coincident with the shovel cycle time through to position ready to spot and dump; and
2. The minor residual of T_S time until the spot position is reached and the shovel dumps - the nett exchange time experience by the shovel whilst waiting for the truck to spot.

The division of T_S time is only relevant for analysis of truck loading time and effect on productivity of loading equipment.

Obviously T_S time for trucks is substantially coincident with the first bucket cycle time. It can be extended by any extraordinary time loss by the shovel during the first bucket cycle as discussed in Section 3.2.9.

Typical nett truck exchange time (intrinsic only) from loading equipment ready to dump and the truck spotting is 12 to 15 seconds. Ramani has used 0.2 minutes for the "incremental exchange time" (Ramani, 1990)

As T_S times are a relatively small proportion of truck trip time, certainly ≤ 10%, and typically in the range 4% down to 1.5%, errors in estimating this time component are of small effect. Adoption of 0.7 or 0.75 minutes will be generally acceptable – always provided that any extraordinary time losses are dealt with separately and transparently.

The above discussion assumes reasonable working face width, manoeuvering area and competent bench surface material. Incompetent floors and resulting high rolling resistance, potential for bogging of trucks, limited working face and restricted work area and interference from support equipment attending to "housekeeping" activities can significantly elevate T_S times, possibly by as much as 100% in worst possible combinations of unfavourable conditions.

Table 3.58, provides a recommended range of working widths for a range of trucks with indicated nominal payloads, at various operating-comfort levels, based on consideration of dimensions of trucks and appropriate loading equipment.

Table 3.58 Truck Working Face Widths Metres

Nominal Payload - Tonnes	90	140	180	220	340
Adequate – Recommended General Planning	60	70	75	80	110
Planning Minimum – Some Productivity Penalty Expected	40	50	55	60	90
Absolute Minimum – Significantly Reduced Performance – “Goodbye Benches”	25	30	35	40	60

Absolute minimum cases are typically the bottom benches of the pit where minimum pit bottom width is controlled by dimensions of the equipment. (Bozorgebrahimi, Hall and Scoble, 2003). This reference implies that equipment selection will be completed before the pit bottom width is determined that, in turn, can affect the production plan and service required from load and haul equipment. Obviously there will be intuitive understanding of scale of the operation and likely limited range of load and haul equipment applicable. Experience has a role in the early planning decisions.

Referring to Table 3.58, recommended T_S times should apply without reserve to the “Adequate” widths indicated; also generally to “Planning Minimum” width with allowance of an additional 20% at ends of working faces – against pit walls and bench edges. Each case of absolute minimum width conditions needs to be considered on its merits. Trucks may not be able to pass for long stretches of bench. Special turning bays may have to be provided with long reverse travel. It is impractical to forecast a generally applicable adjustment of T_S time for potential non-intrinsic effects. Where necessary, time adjustments for such effects should be separate and transparent.

2 Manoeuvre and Dump Time - T_D

T_D time consists of two parts:

1. Raising and lowering the body, generally ranging from 0.3 to 0.7 minutes.
2. Manoeuvering time.

Combined typical T_D times for rear dumps are 1.0 minute and for bottom dumps are 0.5 minute (Hays, 1990). A typical range of T_D times for rear dumping mining trucks is 1.0 to 1.2 minutes (Caterpillar#1, 2004).

Soft running conditions and poor dumping point housekeeping (dangerous dump edges and inadequate safety windrows etc.), or queuing at crushers and/or stockpile areas add to T_D times. These inhibiting factors should be preferably be eliminated; but in default should be treated by applying separate, transparent contingency time allowances.

3 Truck Loading Time - T_L

Discussed at some length in Section 3.2.9, in summary, loading time is:

- From the loading equipment perspective, an accumulation of a limited number of bucket-cycle times and nett truck exchange time – T_C as applied in Section 3.2.4.
- From the truck perspective, an accumulation of delivery of the first pass immediately on spot – normally 3 to 5 seconds provided in estimates – and the second through to the final bucket pass to realise a full truck body load, determined by the operator visually, and/or assisted by onboard payload weighing, with dispatch signaled by the loading equipment operator – designated T_L herein.

Results of statistical analysis of loading times T_C in Section 3.2.9 relevant to this section can be briefly summarized:

- Filtered to intrinsic loading time levels, distributions of T_C exhibit central tendency that, with the addition of extraordinary time losses in practical application, becomes increasingly positive (right) skewed with increasing mean values and CV.
- From the analysis in Section 3.2.9, the CV range for T_C are in the order of 0.10 to 0.15 when filtered to exclude non-intrinsic, extraordinary time losses; and increasing to the order of 0.15 up to 0.20 retrogressing through the adopted filtering levels back to raw data.

The difference between T_C and T_L has been explained above. It is the author's opinion that these two loading times should exhibit similar variability as the basic time components, bucket cycle times, are generally represented similarly.

Loading time variability suggested by others include Elbrond who has used CV of 0.30 for both loading time T_L and truck travel time T_T (truck trip time herein) – (Elbrond, 1990, p743).

Typical loading times can estimated by:

$T_C = n \bullet T_{BC} + T_{EX}$ (1) Loading equipment

$T_L = T_F + (n-1) \times T_{BC}$ (2) Trucks

T_{BC} = Bucket cycle time

T_{EX} = Nett truck exchange time ≡ shovel wait time to spot truck.

n = Number of bucket passes

T_F = Time for dumping first bucket load

It will be noted that the ratio:

$n \times T_{BC} / (n \times T_{BC} + T_{EX})$

is equivalent to the pass-dependent loading efficiency developed and discussed in Section 3.2.6 and illustrated by Table 3.4.

Estimation of Bucket Cycle Times

Table 3.59 – Parts 1 to 4, Volume 2, Appendices, summarize a bucket cycle time calculation method. Estimated bucket cycle times T_{BC} are built up from incremental time components including:

- Dump bucket load - T_F
- Return rotation of loading equipment (or manoeuvre of wheel loader)
- Pause to position bucket and address digging face
- Fill bucket
- Rotate loaded to bucket dump position

Nett truck exchange time, T_{EX}, may be included in the first bucket load by some estimators and data collection systems – as with data analysed in Section 3.2.9. For the purposes of this analysis of bucket cycle time, nett truck exchange time is considered a separate time event.

Table 3.59 is a prototype forecast facility for T_{BC} based on general industrial practice and experience of the author. Although empirical to a large degree, the inputs adopted need to be confirmed for specific applications against time and motion data for each of the time elements to prove validity of assumptions and calculation criteria adopted. The table is limited to electric rope shovels and hydraulic shovels, but covers a substantial selection of the range of capacity offered by OEM. The four-part T_{BC} selection Table 3.59 is not a finished product; but it is a useful start, covering a substantial range of loading equipment. It presents an opportunity for future researchers to refine, extend as necessary and confirm empirically.

Assumptions basic to compiling Table 3.59 Parts 1 to 4 include:

- T_{EX} - is not included and must be separately estimated – see discussion below.
- T_F – for all shovels is set at 3 seconds – average for and applicable for all loads including the first – allows for "feeling" loads into the truck body to avoid damage and spillage – wheel loaders should be provided with more time to dump by extending T_F to 5 seconds.
- Return rotation unloaded – assessed from revolving speed of shovel superstructure from OEM specifications. Allowance is made for acceleration and deceleration correlated with industry standards where available estimates providing for a range of swing angles. Slewing acceleration and braking characteristics were discussed with OEM and dealer representatives and adjusted as necessary consistent with the author's experience.
- Pause and position the bucket to address the face of material to be loaded - allowing a mean 3 seconds for any necessary position correction or extra back swing of the stick to position for pickup of large particles tailed off the slumping face and so clear the face in front of the shovel.
- Fill bucket - provision for four levels of digging difficulty – based substantially on experience and with due consideration of standard industry references including:

- Caterpillar Performance Hand Book (Caterpillar#1, 2004)
- SME Mining Engineers Handbook (Atkinson, 1990) recommends values for bucket filling times that have been modified to reflect more recent improvements in digging performance – particularly upgraded specification of electric rope shovels, hydraulic powered shovels and excavators.

Table 3.59 Parts 1 to 4, appended in Volume 2, were developed as a Microsoft Excel spreadsheet. A copy of the file is included on a CD in a pocket inside the back cover of this thesis.

Practical Loading Time Distributions

Loading time data is generally available from load and haul data from dispatch management systems. Time components that make up truck loading time, in the context discussed above, are assumed to have specific instantaneous beginning and end points that may be, to some degree, automatically triggered by events sensed at truck or shovel. But sometimes beginning and end points in time are triggered by proximity observation using GST facilities in dispatch systems. Such observations are not discrete events so there may be some inaccuracy, albeit modest, in loading time data. One such definition of time events for a dispatch system describes loading time as follows:

"When a haul truck **hauling empty** goes within a 50m radius and first stops it automatically goes into **wait at Shovel** unless it is waiting behind other trucks, then it will go into **queuing at shovel**. If the shovel at this time is not loading another truck it will automatically go into **face preparation**.

If the shovel finishes loading a truck and there is not another truck already in **wait at shovel** then the shovel is placed in **Shovel waiting** (i.e. waiting for a truck to arrive).

When the haul truck goes within 25m of the shovel and stops the truck and shovel will automatically go into **loading time.** As soon as the truck heads off and hits a nominal speed (to allow for minor repositioning under the shovel - when it hits second gear) it will automatically go to **hauling full.** The shovel will then revert back to face preparation before being triggered back into load by the arrival of the next truck.

The entire **haul cycle** is the time from dump to load to dump. This is the entire time from the time the truck dumps until it dumps its next load.” (Cutts, 2005).

It should be noted that, for the practical case above:

- The truck has to stop to change a time category - so event triggers are practically discrete.
- The 25m-proximity trigger will generally be when the truck has completed its 180^{o} turn and stopped. - the time taken to reverse, say, 25 metres will be in the order of 10 to 15 seconds, approximately corresponding to the nett truck exchange time as discussed in this section, also in Sections 3.2.6 and 3.2.9.
- Loading time data from the system as configured will be comparative for that site.
- Data from dispatch systems and traditional time and motion field studies are reconcilable – when in doubt, system data can be compared with field observations – or conveniently with video recording reconciled to the relevant real time period.

Table 3.60, provides descriptive statistics for a sample of 2,630 loading times (recorded in accordance with the definition) and a further sample of 51 truck trip times extracted from the 2,630 records with the selected, same one-way haul distance with equal loaded and return hauls. Yield of suitable trip time data was limited due to the imposed dispatching prescription. A busy dynamically programmed dispatch system sends trucks to differing load and dump points over differing routes.

Table 3.61, is an extract from Table 3.60 demonstrating:

- For loading times - T_L - filtering of outliers increases central tendency with CV converging on 0.20.
- For truck trip times – T_T – filtering outliers shows a tendency for CV in the range 0.20 to 0.10.

Table 3.60 Loading Times and Truck Trip Times

Number of Records	**51**	**47**	**2,630**	**2,512**	**2,472**	**2,308**	**2,191**
Maximum Value	3,133.0	2,459.0	711.0	429.0	238.0	238.0	237.0
Minimum Value	945.0	1,547.0	68.0	68.0	68.0	78.0	78.0
Range	2,188.0	912.0	643.0	361.0	170.0	160.0	159.0
Average of Range	2,039.0	2,003.0	389.5	248.5	153.0	158.0	157.5
Average of Population (Arithmetic Mean)	1,804.7	1,826.0	143.9	130.5	127.7	129.1	126.9
Mode	#N/A	#N/A	117.0	117.0	117.0	110.0	110.0
Median	1,784.0	1,788.0	124.0	123.0	122.0	123.0	122.0
Variance	117746.3	49468.7	5811.1	1252.8	750.5	756.5	616.5
Std Dev Sample	343.14	222.42	76.23	35.40	27.40	27.51	24.83
Coefficient of Variation	**0.19**	**0.12**	**0.53**	**0.27**	**0.21**	**0.21**	**0.20**
Skewness	0.79	1.01	3.96	2.70	1.13	1.13	1.00
	To Right	To Right	To Right	To Right	To Right	To Right	To Right
Kurtosis	4.43	0.53	18.11	12.75	1.49	1.38	1.22
	Lepto-kurtic	Lepto-kurtic	Lepto-kurtic	Lepto-kurtic	Lepto-kurtic	Lepto-kurtic	Lepto-kurtic

Table 3.61 Coefficients of Variation – Truck Trip & Loading Times

Description	Truck Trips	Truck Trips Filtered	Loading Times - Successive Filtering of Outliers				
Case	**1**	**2**	**3**	**4**	**5**	**6**	**7**
Number of Records	51	47	2,630	2,512	2,472	2,308	2,191
Coefficient of Variation	**0.19**	**0.12**	**0.53**	**0.27**	**0.21**	**0.21**	**0.20**

The above descriptive statistics are illustrated by Figure 3.56.

These outcomes are consistent with:

- For T_L – the results of analysis of bucket cycle and loading time data from another relatively large-scale deep open-pit operation as analysed and discussed in Section 3.2.9.
- For T_T – CV standards of 0.10 to 0.20 suggested by Hays (Hays, 1990)

- Also Caterpillar has conducted in-depth cycle (truck trip) studies to determine cycle (truck trip) variation under different operating conditions. The outcome of these studies was finding the CV to vary between 0.10 and 0.20 (Gove, 1994). A further finding was that, when the "fleet match" was 1.0 – "a perfect match of haulers to loader" for the given average cycle (T_T), productivity was reduced by a factor of one minus the CV of cycle time (T_T).

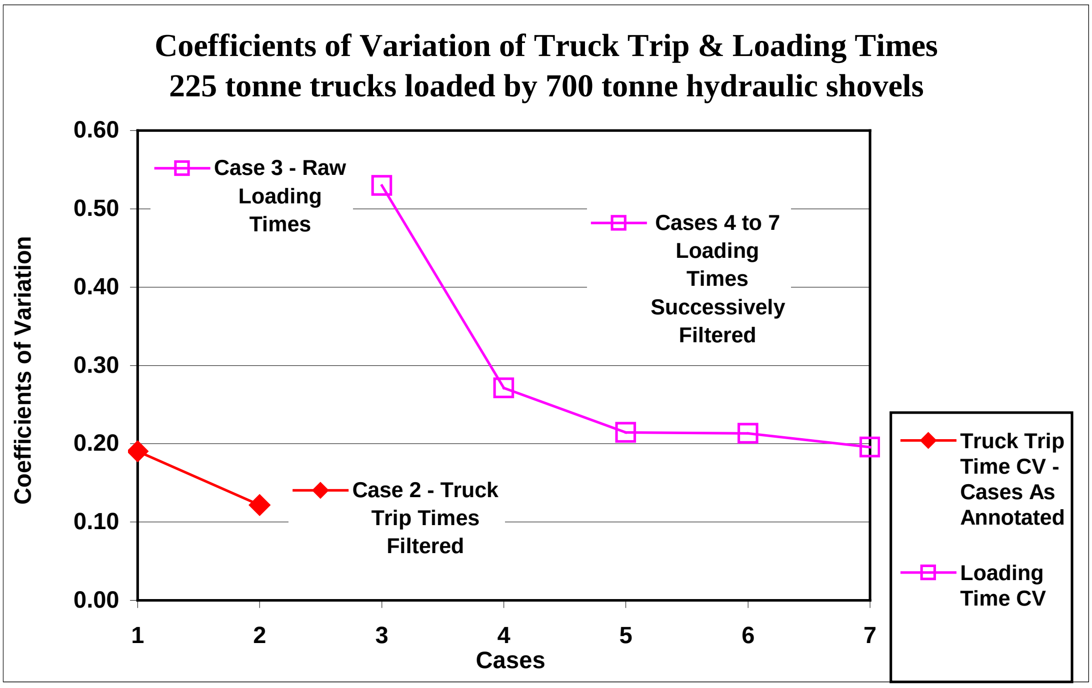


Figure 3.56 From Table 3.61

Thus with:

- CV of T_T = 0.10 and perfect fleet match, productivity is discounted due to bunching by: 1 – 0.10 = 0.90 of theoretical fleet productivity.
- CV of T_T = 0.20 and perfect fleet match, productivity is discounted due to bunching by: 1 – 0.20 = 0.80 of theoretical fleet productivity (Gove & Morgan, 1994)

These two discounts of 0.90 and 0.80 are recommended and applied by Caterpillar as the extremes of productivity discounting due to bunching as discussed below. This will be further analyzed and discussed in Section 3.5.4.

4 Truck Travel Time - T_V

As a basis for consideration and analysis for truck travel time, three separate conceptual deep open-pit haul courses were analyzed using Caterpillar’s Fleet Performance and Cost, Ver 3.05B (FPC) – simulation software

The three deep open-pit conceptions were for pits 50 metres, 200 metres and 500 metres deep. Details of the three courses are provided in Table 3.62. The three conceptual courses are considered practical, suitable for comparison and are, together with all parameters chosen, are not necessarily optimum or exactly similar to any current existing operation.

Table 3.62 Conceptual Deep-pit Haul Road Course Details - for FPC Analysis

COURSE 1									
Haul Loaded					**Return Empty**				
Segment	**Distance**	**Rolling Resist’ce**	**Grade**	**Speed Limit**	**Segment**	**Distance**	**Rolling Resist’ce**	**Grade**	**Speed Limit**
	metres	**%**	**%**	**Kph**		**metres**	**%**	**%**	**Kph**
1	200	4	0	20	**5**	180	3	0	
2	500	2	10		**4**	250	2	-8	50
3	100	2	0		**3**	100	2	0	
4	250	2	8		**2**	500	2	-10	40
5	180	3	0		**1**	200	4	0	20
COURSE 2									
Haul Loaded					**Return Empty**				
Segment	**Distance**	**Rolling Resist’ce**	**Grade**	**Speed Limit**	**Segment**	**Distance**	**Rolling Resist’ce**	**Grade**	**Speed Limit**
	metres	**%**	**%**	**Kph**		**metres**	**%**	**%**	**Kph**
1	250	4	0	20	**5**	400	3	0	
2	2000	2	10		**4**	940	2	-8	50
3	100	2	0		**3**	100	2	0	
4	940	2	8		**2**	2000	2	-10	40
5	400	3	0		**1**	250	4	0	20
COURSE 3									
Haul Loaded					**Return Empty**				
Segment	**Distance**	**Rolling Resist’ce**	**Grade**	**Speed Limit**	**Segment**	**Distance**	**Rolling Resist’ce**	**Grade**	**Speed Limit**
	metres	**%**	**%**	**Kph**		**metres**	**%**	**%**	**Kph**
1	300	4	0	20	**5**	1300	3	0	
2	5000	2	10		**4**	1875	2	-8	50
3	200	2	0		**3**	200	2	0	
4	1875	2	8		**2**	5000	2	-10	40
5	1300	3	0		**1**	1300	4	0	20

Haul road construction and operating conditions were assumed generally in accordance with best-practice standards as described in references including:

- Walter Kaufman and James Ault - (Kaufman & Ault, 1977)
- Caterpillar PHB 35, 2004, p27-1 – (Caterpillar#1, 2004)
- Caterpillar FPC 3.05B, 2004, pp106, 107 – (Caterpillar#2, 2004)
- Ronald Hays – (Hays, 1990, p677)

It will be noted that:

- Grade of the in-pit ramp was set at 10% with grade of the dump ramp at 8% - all other course segments were assumed horizontal.
- Rolling resistance was chosen consistent with correctly constructed haul roads using acceptable materials – haul roads that are well maintained.
- Speed limits were set at 20 kph in-pit loaded, 50 kph down dump ramp and 40 kph down in-pit ramp on return – based on the author's experience for in-pit bench travel and from consideration of Caterpillar recommendations (Caterpillar#2, 2004) – see Figure 3.57.
- Operator Efficiency was applied by FPC in accordance with a job-studies based relationship between operator efficiency and one-way haul distance as default that can be over-ridden if desired – the analysis accepted default (Caterpillar#2,2004) – see Figure 3.58.

The FPC simulations were compiled for a range of six mining trucks from nominal 54 to 349 tonnes payload. Table 3.63, summarises the FPC results. Manoeuvre and dump times - T_D - adopted over the truck range are shown in the table.

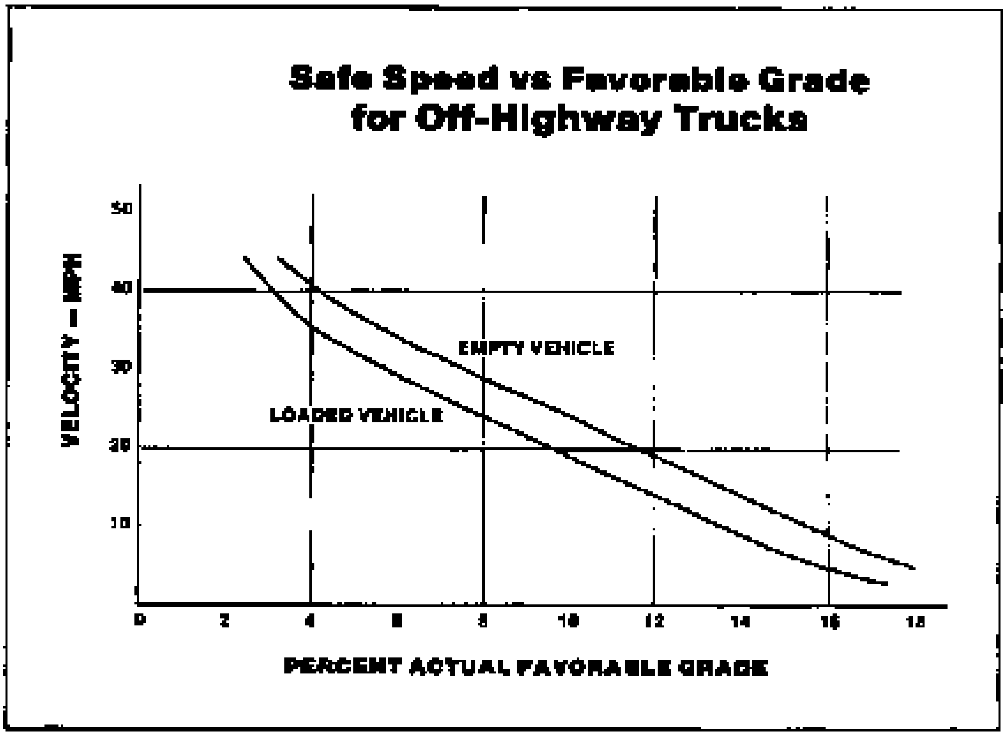


Illustration 1. Safe Speeds for Off-Highway Trucks on Favorable Grades

Figure 3.57
From Caterpillar FPC 3.05B Manual, Appendix D

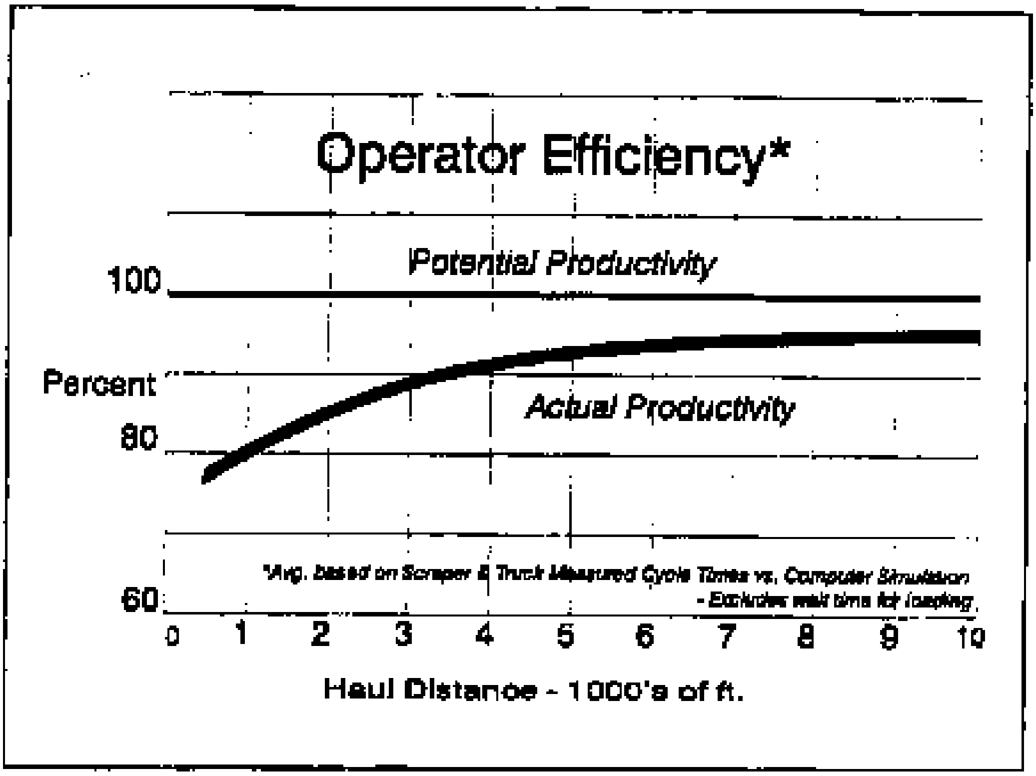


Figure 3.58
From Caterpillar FPC 3.05B Manual, Appendix B

Table 3.63 - FPC Results - Comparison Across the Range of Caterpillar Mining Trucks - Deep-pit Application

Fleet Number **Caterpillar Truck Model**		1 773E	2 777D	3 785C	4 789C	5 793C	6 797B
Nominal Payload Tonnes		53.90	88.12	140.24	180.07	227.27	349.02
Course 1							
FPC Results							
One Way Haul - Metres	**Units**	**1,230**					
Load & Truck Exchange	**Minutes**	2.15	2.20	2.49	2.49	2.81	3.10
Loaded Travel	"	4.35	4.78	4.96	4.86	4.80	4.77
Manœuvre & Dump	"	1.00	1.00	1.00	1.10	1.10	1.20
Return Unloaded	"	2.14	2.16	2.16	2.15	2.15	2.15
Potential Truck Trip Time	"	9.63	10.13	10.61	10.60	10.86	11.22
Slow Hauler Allowance	"	0.00	0.00	0.00	0.00	0.00	0.00
Average Bunching	"	2.31	1.99	2.97	2.63	4.65	4.82
Total Estimated Truck Trip Time	"	11.95	12.12	13.58	13.23	15.51	16.04
Interpretations							
Av Speed - Travel Only	**kph**	22.74	21.27	20.73	21.06	21.24	21.33
Av Speed Potential Truck Trip	**kph**	15.33	14.57	13.91	13.92	13.59	13.16
Average Speed Estimated Truck Trip	**kph**	12.35	12.18	10.87	11.16	9.52	9.20
Potential Production per Payload Tonne per Hour =Trips per Hour	**Tonnes**	6.23	5.92	5.66	5.66	5.52	5.35
Potential Production - Tonne-km per Payload Tonne	**Tonnes-km**	15.33	14.57	13.91	13.92	13.59	13.16
FPC Forecast Production per Payload Tonne per Hour	**Tonnes**	5.02	4.95	4.42	4.54	3.87	3.74
Fuel Consumption	**l/Tonne**	NA	0.182	0.174	0.180	0.169	0.171
Fuel /Tonne- km Total Haul	**l/Tonne**	NA	0.074	0.071	0.073	0.069	0.070
Course 2							
FPC Results							
One Way Haul - Metres	**Units**	**3,690**					
Load & Truck Exchange	**Minutes**	2.15	2.20	2.49	2.49	2.81	3.10
Loaded Travel	"	14.28	16.03	16.57	16.27	16.03	15.93
Manœuvre & Dump	"	1.00	1.00	1.00	1.10	1.10	1.20
Return Unloaded	"	5.60	5.63	5.64	5.63	5.63	5.92
Potential Truck Trip Time	"	23.03	24.87	25.70	25.49	25.56	26.15
Slow Hauler Allowance	"	0.00	0.00	0.00	0.00	0.00	0.00
Average Bunching	"	0.30	0.13	0.54	0.40	1.24	1.34
Total Estimated Truck Trip Time	"	23.33	25.00	26.24	25.89	26.80	27.49
Interpretations							
Av Speed - Travel Only	**kph**	22.27	20.44	19.94	20.22	20.44	20.27
Av Speed Potential Truck Trip	**kph**	19.23	17.80	17.23	17.37	17.32	16.93
Average Speed Estimated Truck Trip	**kph**	18.98	17.71	16.88	17.10	16.52	16.11
Potential Production per Payload Tonne per Hour	**Tonnes**	2.61	2.41	2.33	2.35	2.35	2.29
Potential Production - Tonne-km per Payload Tonne	**Tonnes-km**	19.23	17.80	17.23	17.37	17.32	16.93
FPC Forecast Production per Payload Tonne per Hour	**Tonnes**	2.57	2.40	2.29	2.32	2.24	2.18
Fuel Litres/Tonne	**l/Tonne**	NA	0.554	0.525	0.543	0.497	0.498
Fuel /Tonne- km Total Haul	**l/Tonne**	NA	0.075	0.071	0.074	0.067	0.067

Table 3.63 FPC Result - Continued

Fleet Number Caterpillar Truck Model Nominal Payload Tonnes		1 773E 53.90	2 777D 88.12	3 785C 140.24	4 789C 180.07	5 793C 227.27	6 797B 349.02
Course 3							
FPC Results							
One Way Haul - Metres	**Units**	**8,675**					
Load & Truck Exchange	**Minutes**	2.15	2.20	2.49	2.49	2.81	3.10
Loaded Travel	"	32.90	37.00	38.39	37.73	37.05	36.77
Manœuvre & Dump	"	1.00	1.00	1.00	1.10	1.10	1.20
Return Unloaded	"	12.31	12.38	12.50	12.49	12.54	13.09
Potential Truck Trip Time	"	48.37	52.58	54.39	53.81	53.52	54.16
Slow Hauler Allowance	"	0.00	0.00	0.00	0.00	0.00	0.00
Average Bunching	"	0.00	0.00	0.00	0.00	0.00	0.00
Total Estimated Truck Trip Time	"	48.37	52.58	54.39	53.81	53.52	54.16
Interpretations							
Av Speed - Travel Only	**Kph**	23.03	21.08	20.46	20.73	20.99	20.88
Av Speed Potential Truck Trip	**Kph**	21.52	19.80	19.14	19.35	19.45	19.22
Average Speed Estimated Truck Trip	**Kph**	21.52	19.80	19.14	19.35	19.45	19.22
Potential Production per Payload Tonne per Hour	**Tonnes**	1.24	1.14	1.10	1.12	1.12	1.11
Potential Production - Tonne-km per Payload Tonne	**Tonnes-km**	21.52	19.80	19.14	19.35	19.45	19.22
FPC Forecast Production per Payload Tonne per Hour	**Tonnes**	1.24	1.14	1.10	1.12	1.12	1.11
Fuel Litres/Tonne	**l/Tonne**	NA	1.270	NA	1.239	1.131	1.131
Fuel /Tonne- km Total Haul	**l/Tonne**	NA	0.073	NA	0.071	0.065	0.065

FPC results show/confirm that:

- Loading times – T_L – increase over the truck range consistent with generally moderate increases in bucket cycle times – T_{BC} - as loading equipment increases in capacity – all loads were manually adjusted to four passes per truck payload and all truck payloads to target payload allowing bucket fill factors to adjust automatically.
- For each course "Loaded Travel" times are reasonably consistent over the truck range – as expected from generic design over the range.
- Times returning unloaded were consistent due to adoption of speed limits – in practice this is somewhat artificial as discussed below.

Interpretations of the FPC results compared:

- Average speed for travel only segments of truck trips – exhibiting expected narrow range of 20 to 23 kph consistent with the 20 to 22 kph observed and used by the author in estimates for more than 20 years – see Figure 3.59.

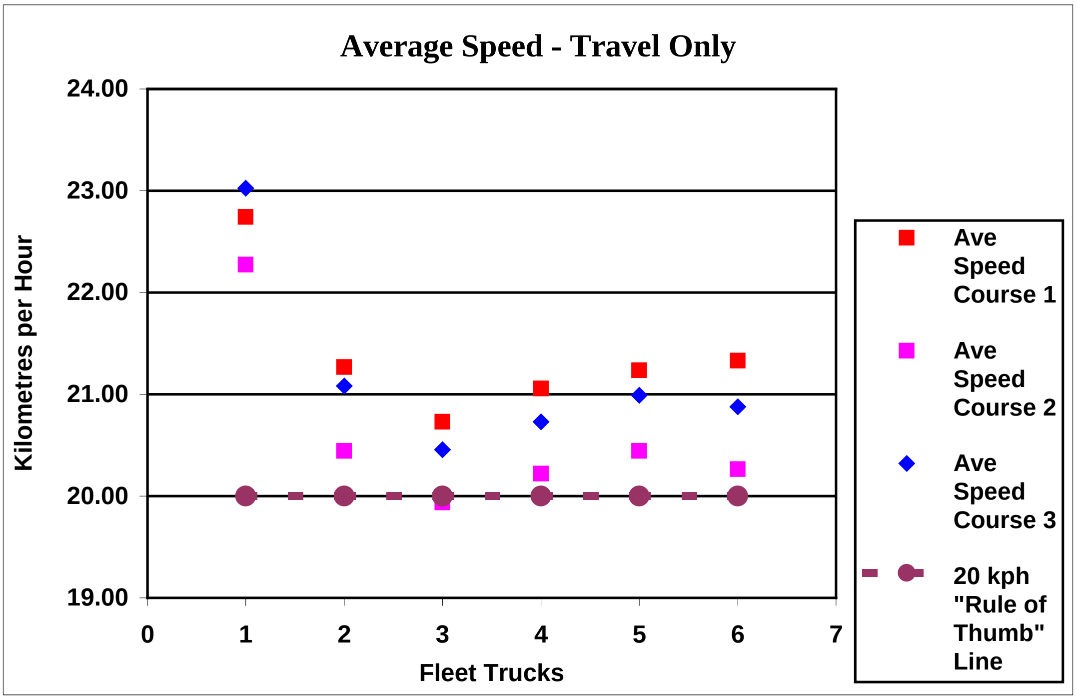


Figure 3.59 From Table 3.63

- Potential production per payload tonne for the truck range – this is equivalent to potential truck trips per hour - provisions for average bunching, waiting time for trucks were ignored as unnecessary complications for this comparative analysis.

- Potential production, expressed in tonne – km per hour, per payload tonne, to reduce the outcomes to be more comparable over the courses – increasing relative productivity with haul distance due to reduced proportion of "fixed" times in truck trip times.

- Fuel consumption, litres per tonne, obviously increases with haul distance; but, reducing fuel consumption to litres per tonne-kilometre, illustrates similar fuel burn per tonne of payload for equivalent haulage duties across the range of trucks analyzed shown by Figure 3.60,.

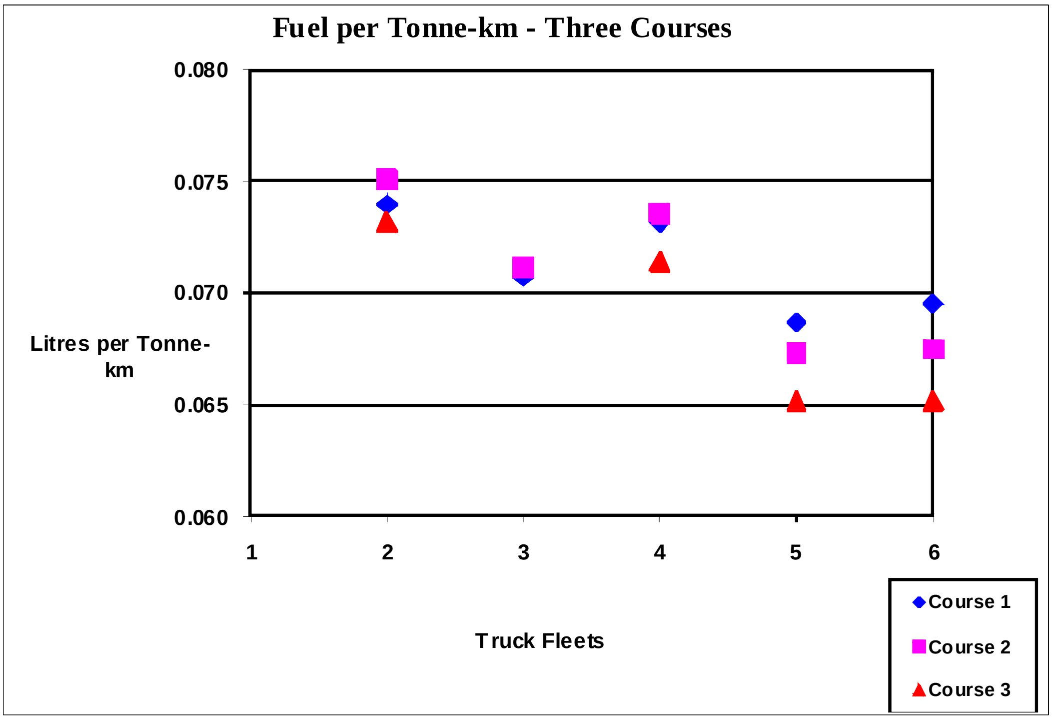


Figure 3.60 From Table 3.63

Significant conclusions, confirming expectancies from previous research, provided by FPC results to this stage of the current research include:

- Due to generic design over the truck range, overall similar unit-production performances are manifest. Productivity and fuel consumption are generally consistent when these parameters are reduced to a common unit basis.

- Productivity on a unit capacity basis tends to decrease modestly with increasing truck size – Figure 3.61.

- Fuel usage per tonne-km tends to moderately decrease as truck size increases, likely due to increasing fuel efficiency of engines in larger trucks; also symptomatic of modified engines with longer stroke and lower brake-mean-effective pressure (BMEP). There is potential for retro application of this means of improving fuel economy for medium to small mining trucks.

Further comments follow on differences that can be expected in practice compared with the hypothetical conceptual courses and artificially controlled inputs to FPC cases investigated and reported herein.

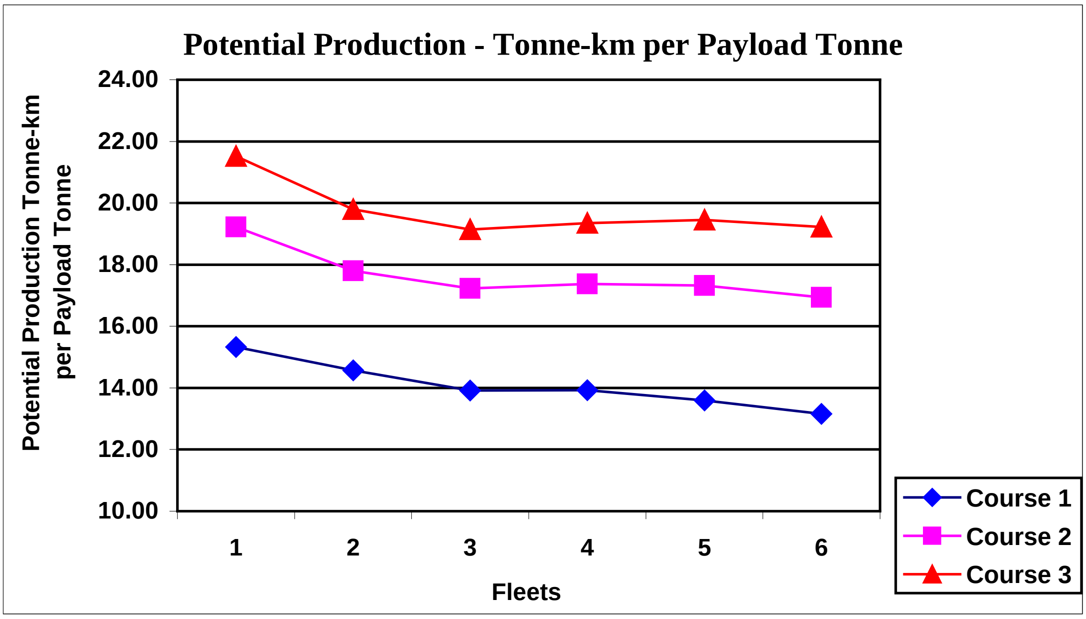


Figure 3.61 From Table 3.63

Very similar return times:

FPC-generated results as described, over the truck range for a specific course, notwithstanding the influence of speed limits, are unlikely to be observed in practice. Loaded travel is generally at full throttle all the way. The author still remembers the training instructor's admonition - "anticipate last pass, foot on service brake, release park brake (or retarder), select first range, when loader horn sounds, throttle (accelerator) flat to the floor all the way (subject to safety and acceptable road conditions) until allowing speed to fall off entering manœuvring to dump" – the reader might recognize this as a "contract-mining-operation syndrome".

Return travel is rarely, if ever, under full throttle. For long sections the retarder (mechanical or electrical) is operating. Operator practices, slowing or stopping for up-ramp loaded trucks (with right-of-way to overtake on narrow sections), road maintenance equipment, obstructions on ramps and other essential traffic essentially determine return time. Blackwell's comments are relevant here: "*The conclusions to be drawn are that full haul times are well predicted, but return empty times are not. There is little the operator can do when travelling up ramp loaded, except run the engine at full power ----. When returning with empty trucks, some drivers will be more skilled and take less time, and empty trucks will wait for full trucks at narrow sections of the road, adding a large amount of randomness----.*" (Blackwell, 1999).

Considering speed alone, down-ramp is obviously more risky than up-ramp operation. Road surfaces that are over-watered or wet from rain always encourage a prudent reduction in down-ramp speed.

Contingency allowances for increased return time are dependent on the specific operation, competence of road surfacing materials and efficacy of road maintenance. It is not practical to set specific guidelines. As a broad guide the optimum return time to loaded travel time ratio will be in the range 35% to 45 %. But in special cases return time proportion may exceed 50%. Longer than expected return times require investigation and adoption of suitable allowances. The author has observed one open-pit coal mine in a high rainfall region where, for a number of reasons, including wet, incompetent road surfacing materials, return time for a relatively short haul exceeded loaded travel time.

Practical Loading Time Adjustments

Considerations include:

- Truck exchange time – discussed in Section 3.2.9..
- Additional loading time for part or the whole of a bucket pass – T_{BC} or part thereof – as discussed in Section 3.3.9.

It will be noted that the total of loading time T_L plus manoeuvre and dump time T_D range indicated in Table 3.63, is 3.15 to 4.3 minutes. Passes were limited to 4. For an initial conservative fixed time in truck trip time T_T it is recommended that an additional 0.4 minutes providing a practical range of 3.4 to 4.5 minutes. For initial deterministic analysis of trucks, before loading equipment options have been identified, a fixed time of 4 minutes is a reasonable provision.

Payload Variability and Truck Travel Times

To analyze cost penalties from overloading large mining trucks; also for examination of the affect of payload dispersion on operating costs, a matrix of cases was analyzed using Caterpillar's FPC as follows:

- Course 1, 2 and 3 and the range of six mining trucks used in the analysis were retained.
- Nine cases of truck-payload variation in 5% payload increments through the range +/-20% were created for each course within FPC by, for convenience,

adjusting the rolling resistance and grade for the loaded haul segments. The total resistance adjustments used for each payload step were appropriately reconciled to ensure rimpull effects were commensurate with the GMW.

- FPC input was manually adjusted to 4 passes with the bucket fill factor allowed to auto-adjust from values in the previous analysis.
- The validity of the technique of simulating payload increase by increasing total resistance was verified by adjusting a sample course and truck fleet to original nominal grades and rolling resistances and adjusting the truck payload by the comparative payload increase keeping bucket passes at the constant 4 and allowing the bucket fill factor to auto-adjust.
- The resulting comparison confirmed there was no significant difference in the results using either method to develop performance data for varying payload – notwithstanding that the grade adjustment was to the tangent when, more correctly, it should have been to the sine of the grade angle.
- FPC calculates grade resistance using the sine of the grade angle. At grades of 20% or less the sine and tangent are practically equal – as was confirmed by the FPC calculation to audit the simulated payload variations.

Results from FPC analysis are provided in Tables 3.64 and 3.65. Table 3.64 is expressed in comparative performance indices. Only fleets 4, 5 and 6 were summarized and further analyzed.

Table 3.64 Comparative Performance Indices for Payload Range +/- 20% - For Three Courses

Potential Truck Trip Times	Truck Payload %								
	80	85	90	95	100	105	110	115	120
Course 1									
Fleet 4 Cat 789C	1.056	1.043	1.030	1.012	1.000	0.994	0.989	0.981	0.972
Fleet 5 Cat 793C	1.047	1.036	1.025	1.013	1.000	0.982	0.973	0.968	0.963
Fleet 6 Cat 797B	1.042	1.034	1.024	1.012	1.000	0.988	0.973	0.959	0.954
Course 2									
Fleet 4 Cat 789C	1.090	1.071	1.050	1.018	1.000	0.993	0.985	0.973	0.959
Fleet 5 Cat 793C	1.079	1.054	1.037	1.021	1.000	0.968	0.956	0.948	0.941
Fleet 6 Cat 797B	1.074	1.059	1.039	1.018	1.000	0.981	0.957	0.934	0.927
Course 3									
Fleet 4 Cat 789C	1.106	1.085	1.059	1.021	1.000	0.992	0.984	0.970	0.955
Fleet 5 Cat 793C	1.092	1.064	1.045	1.025	1.000	0.962	0.949	0.941	0.933
Fleet 6 Cat 797B	1.085	1.068	1.045	1.021	1.000	0.977	0.949	0.921	0.915

Table 3.65 Comparative Productivity Indices for Payload Range +/- 20% - For Three Courses

Potential Truck Trip Times	Truck Payload % 80	85	90	95	100	105	110	115	120
Course 1									
Fleet 4 Cat 789C	0.845	0.887	0.927	0.962	1.000	1.044	1.088	1.128	1.167
Fleet 5 Cat 793C	0.838	0.881	0.922	0.962	1.000	1.031	1.070	1.113	1.155
Fleet 6 Cat 797B	0.833	0.879	0.921	0.961	1.000	1.037	1.070	1.103	1.145
Course 2									
Fleet 4 Cat 789C	0.872	0.911	0.945	0.967	1.000	1.042	1.083	1.118	1.151
Fleet 5 Cat 793C	0.863	0.896	0.934	0.970	1.000	1.016	1.051	1.091	1.130
Fleet 6 Cat 797B	0.859	0.900	0.935	0.967	1.000	1.030	1.053	1.074	1.113
Course 3									
Fleet 4 Cat 789C	0.885	0.922	0.953	0.970	1.000	1.042	1.082	1.115	1.146
Fleet 5 Cat 793C	0.874	0.905	0.940	0.974	1.000	1.010	1.043	1.082	1.120
Fleet 6 Cat 797B	0.868	0.908	0.941	0.970	1.000	1.026	1.044	1.060	1.098

Performance indices for each fleet are relative to the target payload set at 1.000. Each index is the ratio of potential truck trip time T_T (excluding provisions for bunching) with target payload to truck trip time (e.g., T_T+10%, T_T-15% etc.) with payload over or under target by the selected percentage.

Performance indices of Table 3.64, previous page, were multiplied by payload percentage in each case to create productivity indices as summarized in Table 3.65. The relationship between performance and productivity indices over the fleet range for each course, 1, 2 and 3 are illustrated by Figures 3.62, 3.63 and 3.64.

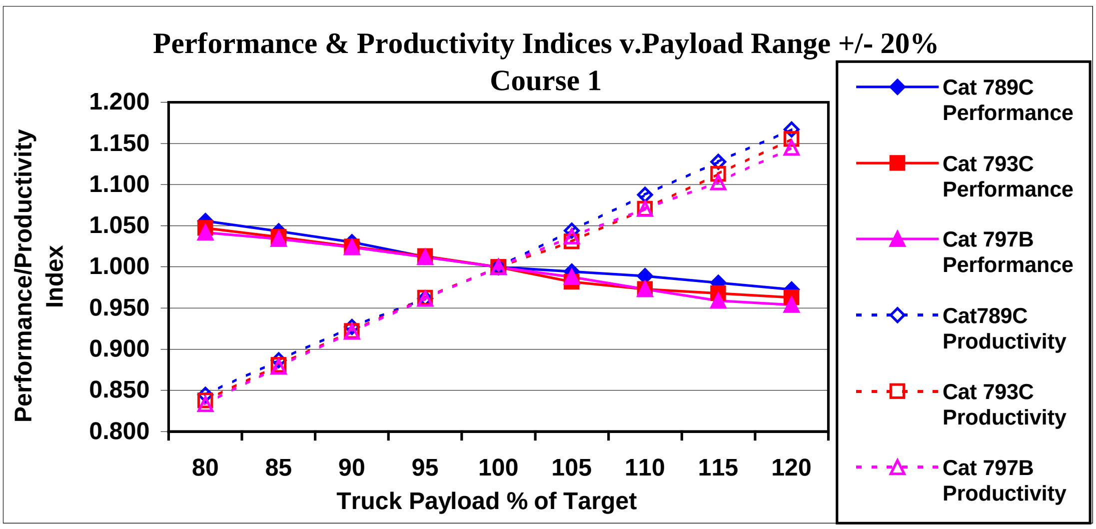


Figure 3.62 From Tables 3.64 and 3.65

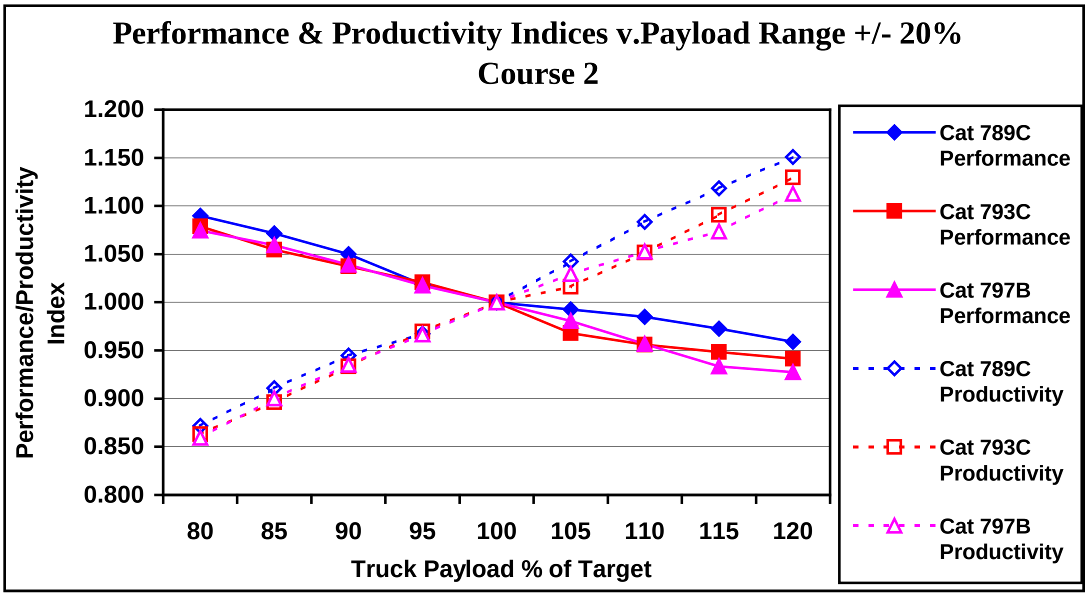


Figure 3.63 From Tables 3.64 and 3.65

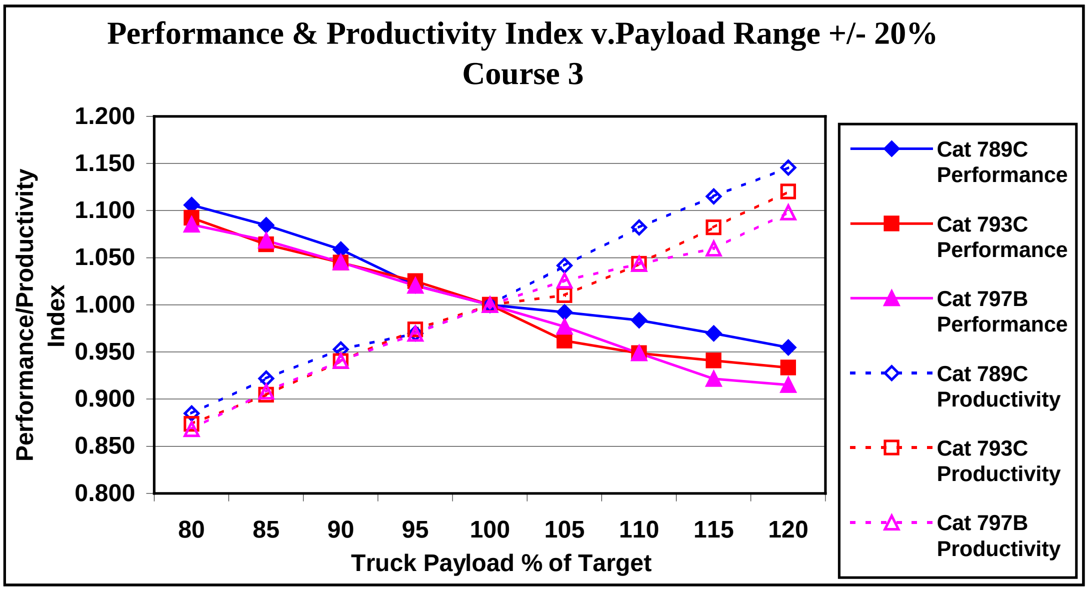


Figure 3.64 From Tables 3.64 and 3.65

Interpretation and conclusions from variable payload analysis include:

- Overloading of mining trucks results in reduced performance, but some 75% of the additional payload effectively increases productivity.
- Increased productivity effect of overloading described is generally complemented by reduced productivity effect when trucks are under loaded.

- Performance and productivity improvement both exhibit a subtle tendency to fall away as haul distance increases – understood to be due to the affect of increased travel time in total trip time.

In isolation, overloading of trucks seems to deliver some productivity benefit, an obvious attraction, if productivity is the sole objective. But this is only part of the way to the ultimate outcome of the analysis of potential benefits, if any, from truck overloading It is a fundamental truth that: "One seldom receives any benefit for no complementary consideration." So it is with mining-truck overloading. Additional productivity comes at some cost. This productivity-cost issue will be further pursued and analyzed in Section 4.3.

Truck Trip Time T_T

The several time components that accumulate to T_T have been analyzed and discussed above. Overall T_T determines haulage performance of trucks. Similarly as discussed in Section 3.2.9 in relation to bucket cycle time and truck-loading time, T_T can be considered in terms of two general categories:

1. Intrinsic – functions of truck capacity and capability only.
2. Non-intrinsic – extraordinary time losses, interruptions and delays to operations affecting truck performance that are functions of the operating and management environment in which trucks are required to operate.

Obviously any inefficiency on the part of the operator is reflected by reduced truck performance. This also applies to any other item of mining equipment that requires permanent hands-on operation in control of the equipment. In all cases operator control is subject to operating procedures policy and/or on-board and remote management systems providing guidance or direction - an ever-increasing assistance in planning and execution of mining operations. Accordingly, selection criteria must include, and investigations must consider, the abundant array of systems management options on offer with equipment from all OEM and dealers.

Comparisons based on FPC analysis of potential performance and productivity discussed above are essentially based on intrinsic performance of trucks. In the process of establishing required performance of equipment and identifying suitable options prior to equipment selection, circumspect consideration needs to be given to all potential time loss items to arrive at practical time parameters.

Any and all non-intrinsic time-loss provisions need to be transparent and justified by duly diligent investigation of local, national and international mining operations. Operational experience, either in-house or imported, provides models for guidance on non-intrinsic time-loss items.

3.3.11 Haul Roads and Site Severity

For mining equipment selection, particularly trucks, it is sufficient to specify that investigation and analysis is based on specific standards for road construction and maintenance that can be set in terms of measurable criteria. Performance and productivity implications of non-conformance with adopted standards need to be quantified and costed. All stakeholders need to be involved in the adoption of standards for pit configuration, particularly for haul road geometry standards, construction materials and formation profile dimensions and details. The final road specifications and other details need to be commercial in the context of project feasibility studies or justification for investment expenditure. At the time of investigating equipment selection, pit design details, particularly haul road construction and mining truck option interrelationships, should be developed concurrent with selection preliminaries and identification of initial options.

Only a brief introductory treatment of haul roads and site severity implications was addressed by the research.

Haul-road layout and design, formation shape, structural profile and running surface quality have become more important as size of mining trucks has increased. Trucks in the 90 tonne to 135 tonne range have, historically, proved more tolerant of less-than-optimum road construction and surface finish standards.

For optimum performance mining trucks require:

- Safe road width for the number of traffic lanes, single lane roads to have bays strategically placed for passing and with adequate allowance for safety windrows and drainage with allowance under high walls to accommodate small debris particles that may fall from the face.

- Sound road formation from competent sub-grade materials designed for truck wheel loads and suitable surfacing for safe operations under all weather conditions.

- Road layout designs that, for truck options under consideration, comply with standards for bend radius, maximum speed ≥ maximum speed of the truck, appropriate super-elevation for the maximum speed and widening at bends to accommodate the wider radial cover whilst trucks negotiate bends.
- Standard transition curves into and out of bends; also vertical transition curves where vertical profile of haul roads changes direction – particularly where the loaded truck runs off the bench onto the access ramp.
- Quality maintenance of road surfaces to retain cross-section and running surface in design condition; and expedient clearance of spillage to ensure expected truck trip time T_T and realization of expected life for tyres and rims, suspension componentry and all structural elements of the truck frame and superstructure.

All of the above truck performance prerequisites are operational rather than planning and equipment procurement process. But truck selection is performance based and made on the basis of an expectancy of consistent acceptable working conditions. These conditions must be specified, understood and a commitment of all stakeholders for final truck-selection decisions to meet requirements.

Road Construction, Criteria and Standards

Road layout and construction criteria for the full range of mining trucks are available from the following standard and references:

- Kaufman W W and Ault J C, (1977) *Design of Surface Mine Haul Roads, Information Circular 8758,* US Department of the Interior, Bureau of Mines, Washington DC USA. (Kaufman & Ault, 1977)
- Feddock J E, (2002) *Haul Roads,* Mining Reference Handbook, Ed. Lowrie R L, Society for Mining Metallurgy and Exploration Inc., Littleton CO, USA (Feddock, 2002)
- Caterpillar (2004) *Performance Handbook Edition 35,* Caterpillar Inc., Peoria. Illinois USA, for curve radii, speed and super-elevation, (Caterpillar#1, 2004)

Kaufman and Ault visited some 300 mines in USA and Canada and based analysis on available specific standards to produce a de facto general standard for all significant aspects of haul road alignment, road width, braking, construction, materials, all road safety and operating accoutrements, special safety provisions and maintenance.

Feddock has summarized Kaufman and Ault's recommendations and Caterpillar have provided curve, speed and road super elevation options based on Kaufman and Ault.

Braking standards for large mining trucks appears to be subject to interpretation. The Society of Automotive Engineers, SAE 166, has set minimum values for vehicle in various GMW classes. For vehicles > 400,000 lbs GMW (> 181.5 tonnes ≡ 100 tonnes payload) the maximum stopping distance from 20 mph allowed = 175 feet (53.4 metres) on dry flat concrete. Kaufman and Ault report on empirical testing of mining trucks where brake fade effects were tested. A safe braking distance of 61 metres was suggested by the research and recommended by Walter Kaufman and James Ault. They comment that "most" mining trucks exceed (*sic* meet the requirements of) that limitation" (Kaufman & Ault, 1977). If braking distance for loaded down-ramp operations is an issue it should be raised with dealers and OEM early in the selection process.

As commented above, road layout configuration, construction and maintenance are essentially operational issues, Any problems or issues with roads in terms of the several factors discussed need to be resolved as prerequisites to proceeding with equipment selection. It needs to be born in mind that the larger the truck or loading equipment the more vulnerable to potential problems with accommodating marginal, close-to-unacceptable, operational limits.

The research has not dealt with the matters discussed above in any detail. This section is intended to provide some starting points for readers who have need to pursue specific haul road topics in depth.

Monitoring Site Severity

Mining trucks exhibit increasing sensitivity to site severity of operations, particularly roads as truck capacity increases. To ensure expected life of suspension and structural componentry, operators of large mining trucks generally accept as necessary site severity monitoring systems that can report in real time and/or accumulate data and down load periodic reports.

In the larger truck range all OEM and dealers provide site severity sensing and reporting systems. Caterpillar, pioneers in the technique, provides a typical system. The VIMS management system records suspension strut pressure transients that are caused by dynamic suspension loads. Pressure transients registered and damped by nitrogen over oil suspension mediums register short-time duration variations transmitted from the road surface through the tyres and suspension. Amplitude of the pressure transients is a measure of quality of the road surface. Initially an electronic control module adjunct to VIMS and software that measured and reported racking and pitching was marketed as Application Severity Analysis (ASA). More recently, in the process of development of the add-on hardware, software and output reports, the system is currently offered as Road Analysis Control (RAC). A significant improvement in RAC is the Fatigue Equivalent Load Analysis (FELA) that stores and compares actual racking and pitching measurement with intended design values for life cycle of components affected by haul road induced stresses – tyres, rims, suspension componentry and super-structure. RAC provides for real-time warnings at increasingly serious levels to operators as well as periodic management reports. Remote interrogation of the system is also provided for supervisory monitoring of operation of the truck.

In the process of mining truck selection, OEM and dealers will offer monitoring systems such as Caterpillar's RAC, as add-ons to VIMS and similar management systems. For large mining trucks such systems are of invaluable assistance to monitor haul-road maintenance and general truck operating conditions to provide a management facility for realization of:

- Expected travel times and truck productivity.
- Operating "rhythm" in haulage operations.
- Expected life of componentry including suspension, truck frame and superstructure.
- Expected life of tyres and rims.

3.4 EQUIPMENT SELECTION AND MAINTENANCE

3.4.1 Introduction and Research Limits

This section substantially extends criteria for equipment selection to include maintenance issues. Consideration must also be given to operational maintenance issues; but only` to the extent that selection decisions have significant impact on operational maintenance facilities and general long-term maintenance efficacy.

Loading and hauling equipment must be maintained mechanically and/or electrically to a predetermined standard - including tyres for trucks - to achieve expected performance. While loading and hauling equipment is being maintained it is generally not able to produce. Only the largest shovels and draglines used in strip mining permit limited maintenance activity whilst in operation. Productive time, nett of the time necessarily lost for maintenance, must obviously be maximized for best production practice. Criteria for equipment selection need to include acceptable availability of productive time. Most importantly, load and haul equipment needs to be reliable and have predictable mechanical performance.

Both selection and operation of loading and hauling equipment are dependent on cogent maintenance-related issues that demand duly diligent investigation of:

- Relative merits of optional equipment considered in the process of selection.
- Detailed criteria for specific operational factors in the form of benchmarks (BM) and key performance indicators (KPI).

Consideration of equipment maintenance in the selection process widens the field of investigation to require a multi-disciplinary effort. From a focus on performance and productivity investigation extends to basic reliability and convenience of maintenance. High maintenance reliability and efficiency results in optimum operating time for acceptable productivity to realise required total production.

3.4.2 Selection Criteria Related to Maintenance

Industry Experience

As discussed in Section 4.1.1 an essential early stage in the equipment selection and procurement process is investigation of experience of current (in-house and externally) and previous owners of the equipment options being considered.

Sources of current owners of equipment under consideration include:

- OEM and dealers who can be expected to be keen to facilitate site visits to demonstrate equipment offered – it is part of the choreography of mining equipment marketing - so enjoy.
- Service industry consultants and organizations keen to be involved in development or expansion projects early in the planning stage.
- Experienced colleagues both internal and external to the mining entity that is investigating equipment procurement – a benefit of (and justification for) industry "net-working" by mining professionals.
- Consumable suppliers, including tyres and ground engaging tools and third-party offsite maintenance service providers.
- Personal knowledge of all stakeholders interfacing with the load-and-haul equipment procurement process.

Maintenance experience of other owners of mining equipment under consideration for procurement is valuable evidence in the equipment selection process.

It is important to determine from all owners and industry contacts having experience with some or all of the equipment options under selection consideration if, with the benefit of hindsight, the same procurement decisions would be made; and, if not, why not?

Preparation of lists of discussion topics is a necessary prerequisite for best returns from due diligence examination of equipment options. Following are some general topics that should be included in preparation for investigation.

Maintainability

The coined term "maintainability" refers to the convenience of effecting all necessary maintenance activities. Assessment of maintainability requires involvement of personnel with specific expertise, preferably internal, who will be taking delivery of equipment on site and have the responsibility for establishing facilities and managing operational maintenance on site.

A suitable checklist of items to investigate maintainability and other essential features of mobile equipment includes:

Generally:

- Expected life of major driveline (including wheel groups, tyres and rims), suspension and structural components such as bodies and buckets; also travelling componentry, tracks, track drives and support frames.
- Initiatives in modular designs to facilitate component-exchange with the benefit of reduced maintenance response times.
- Improved component designs to extend life – modular designs with focus on component exchange that facilitate rationalization of in-house or supplier component inventory - favouring component inventory over spare parts on site; or off-site on call, so optimizing maintenance resource commitment and minimizing maintenance costs – ties into product support discussions below.

Detailed:

- Operator station appointments, operator-friendly controls and comfort to minimise fatigue.
- Review onboard fire suppression installations in the engine compartment and mounting position for luggable fire extinguishers on equipment.
- Identify and inspect actual installation of lubrication services such as fast fuelling facilities, auto-lube grease systems, centralized lubricant top-up points, waste oil extraction,.
- Access around and particularly over driveline componentry.
- Position of all breather locations, access for cleaning and any owner initiatives by way of in-field modifications to extend breather lines to a “clean-air” location.
- Any maintenance problems experienced by other owners, remedies applied and, particularly, any outstanding problems.
- In terms of maintainability, would each owner repeat order specific replacement component items; and, if not, why not?

Preparation of comprehensive checklists to facilitate due diligence is of paramount importance. Any residual blanks in check lists following site visits need to be followed up before finalizing the equipment selection investigation.

Discovery of more faults in, say, one truck option compared with others, on its own, is not necessarily a sufficient reason for discounting or excluding that particular option. The complete picture has to be considered including:

- Remedies applied, promptness of vendors, efficacy of remedies.
- How the faults remedied or otherwise were discovered – if disclosed by the OEM or dealer accompanied by advice that improvement has/is being actioned should be given significant credit.
- Market share of the specific model of equipment is also a good guide.

Beware the faultless equipment item. It is the author's experience that a faultless machine or one with few faults is not necessarily a credit to the specific equipment item but is often a commentary on the inefficacy of the investigator's due diligence.

Larger mining organizations can benefit from establishing internal generic configurations and standards for mining equipment for all options offered for the basic OEM/dealer specification. A general specification including mounting positions and details for add-ons such as fire extinguishers and buyer-requested modifications can be established. Future equipment selection projects can benefit from the established owner-approved specifications and detail drawings. Future purchases can then be specified by exceptions necessary for the specific application, updating as necessary for current models. In the absence of such owner standards, it is possible for different sites within a large mining group to have different configurations of the same equipment for no good reason, a manifestly unsatisfactory and potentially costly position.

Product Support

Achieving expected availability of mobile equipment is dependent on support by OEM and dealers by way of:

- Spare components and parts for components.
- Offsite overhauls and test facilities for quality assurance of rebuilds.
- Training of servicing personnel, operating demonstrators and trainers and specific training of special tooling and equipment management systems.

- Interface with the OEM on warranty issues and unforeseen equipment application problems.
- Performance guarantees, generally offered by dealers with OEM sanction.
- Availability of maintain-and-repair-contracts, MARC, to reduce the maintenance-related operational risk for owners; also to divest the owner of responsibility for insurance spares as discussed below.

Adequate product support is a most important prerequisite to realising expected productivity of mobile equipment.

Special working conditions could possibly favour a normally relatively minor attribute of a particular equipment model to the exclusion of other options. Such circumstances may be possible and could be promoted by focused marketing. But the author has not experienced such circumstances. It is the author's opinion and experience that, when independently analysed, generally the differences between mobile equipment models of approximately equivalent capability and performance offered by the field of OEM tend to be subtle in terms of productivity and unit cost of production.

In any objective analysis (say, using weighted matrix methods) considering purchase price, estimated operating costs, forecast productivity, product support and other factors the analyst considers relevant, product support should, and normally does, receive high priority and weight.

In the author's opinion, product support should be the paramount factor in any procurement process aiming to arrive at a decision to purchase an item of mobile equipment.

Product support:

- Should be included as a group of topics during site visits.
- Needs to be thoroughly probed in discussions with OEM and dealers.
- Particularly availability of service exchange of major driveline components, engines, torque-converter/transmissions, differential groups, alternators, inverter groups, wheel motors, management system componentry should be determined and included in selection considerations.

The larger and more specialised an item of production equipment becomes the more insurance spares will need to be carried by an owner/operator. For example It may be that an owner may well have a spare slewing ring assembly for a large shovel but rely on a supplier for service exchange truck engines. Investigation of insurance-spares policy of other mining equipment operators (including rationale for policy) is an essential due diligence activity in the process of equipment selection.

Essentially assurance of product support is a risk treatment to cover likelihood and consequences of excessive equipment downtime due to inadequate supply of backup components and parts; also to ensure that specialist maintenance and operating skills are applied to realise expected equipment performance.

Large operations, say, more than 15 large mining trucks (135 tonne plus payload) should consider carrying their own insurance spares (unless the policy is to carry a spare truck or trucks) included in the initial delivery programme. Insurance spares (or spare truck[s]) are an alternative to service exchange that has the advantage of the owner being in control of the componentry. There is an element of competition for service exchange components in the hands of dealers or maintenance-service third parties where a number of customers have to be satisfied.

Owning insurance spares also has the advantage that the condition of the replacement components is consistent with the residual value of each mining equipment item in the asset schedule. Also, increasing number of trucks in the fleet reduces unit-cost write off for insurance componentry inventory. Investment in insurance spares for trucks is relatively easy to justify.

It should be noted that, for every 3 large trucks, approximately one engine replacement per year would be required. As fleets increase over 20 trucks towards 30 trucks or more, one the spare engine will be cycled 7 to 10 times. As change outs are not evenly spaced, more than one spare component may be justified. Simulation techniques are obviously applicable to the problem of component inventory for large truck fleets.

Generally smaller numbers of loading equipment items tend to limit the degree of insurance spares that can be justified. Particularly for diesel-powered shovels spare engines are difficult to justify for 1 to 2 engine changes per year on average for 3 to 4 shovel operation. The author’s experience with a logistically difficult operation

(Telfer, Western Australia) was best productivity and economics were realised by carrying spare components as a complete loading equipment item. The operation was "over-shoveled". Trucks controlled the total productivity and operated with minimum bunching. This operational circumstance will be further addressed in 3.5 and Section 4.3.

A medium-sized development of a "greenfields" mining project with 1 or 2 loading equipment items and 8 to 12 medium sized mining trucks may have difficulty in financing the up-front cost of insurance spares. Dealer credit can likely be negotiated as a condition of equipment purchase to supply insurance componentry with payment deferred, say, one year. The alternative is to ensure that service exchange is available and rely on it; a factor that may become significant in the selection process. Obviously the operational risk is increased but may be acceptable to a new mining house with limitations on development capital funding.

So there are several options for equipment support by way of insurance spares. Duly diligent investigation of the options needs to be conducted by the future owner's equipment selection personnel or third party expert investigators to reduce risk of unavailability of replacement componentry to an acceptable residual. Selection of load and haul equipment needs to address product support – particularly supply of insurance spares. If supplied as part of a fleet package there may be some purchasing advantage or financial/tax benefit from including insurance spares in the deal.

Mechanical Availability – MTBF and MTTR

Productivity of mining equipment is essentially dependent on the equipment being in good order and available for work. Given equipment is available for work it is then a responsibility of production management to utilise available equipment productively. In the context of production management, there are two distinct time categories that exclude productive work:

1. That time, commonly termed "downtime" when equipment is disabled or in the hands of maintenance staff and consequently not available for work.
2. That time when equipment is available but is not required by production management to work.

Item 2 - non-operating - time has been expressed as a percentage historically termed **"Job Management Efficiency"** that is generated as a quotient of utilised time and available time.

$$\textbf{JME \%} \quad = \quad \frac{\textbf{Utilised time x 100}}{\textbf{Time available for work}}$$

Traditionally provision for non-utilisation and JME has been facilitated by the "**50 (or 45) Minute Hour"**. In estimating practice, available hours were discounted to 50 (or 45) minutes – equivalent to applying a JME of 83.3% (or 75%).

Item 1 time can be expressed as a percentage termed "Mechanical Availability" that is generated as a quotient as follows:

$$\textbf{MA \%} \quad = \quad \frac{\textbf{Time available for work x 100}}{\textbf{Time available for work + Downtime}}$$

Combining these two time bench marks as a product:

$$\textbf{Utilisation \%} = \quad \frac{\textbf{Utilised time x 100}}{\textbf{Time available for work + Downtime}}$$

The "50 (or 45) minute hour" and JME are traditional terms and concepts with reducing current use. Availability of onboard recording of equipment activities enables classification of time components to be as specific as necessary with current practices, providing for detailed time analysis within maintenance activities; also time available for production and reporting systems. Utilisation and related time issues are further discussed in Sections 3.5 and 3.6.

MA is a time-honoured benchmark for industries using mobile or static equipment that requires periodic maintenance either planned or unplanned. Unfortunately, as a KPI or benchmark, MA is more applicable locally, within rather than between operations where the definitions of "downtime" and "time available for work" are understood so providing consistency of comparison.

It will be noted that "time available for work + downtime" can be interpreted as the total calendar time in any specific period such as a day, week, month or year. In such cases MA is more comparable between mining operations. But there is still wide ranging interpretations of the definition of "time available for work".

For the avoidance of doubt it is sound practice to make the availability decision dependent on function. If in the hands and under control of the maintenance function, equipment is not available. If in the hands and under control of production management, equipment is available.

Terminology, definitions and concepts in the following discussion generally follow relevant product support materials from Caterpillar. But interpretations and any errors are the responsibility of the author.

In the context of maintenance functions MA is considered differently - characterized by the more recent concepts of **Mean Time To Repair (MTTR)** and **Mean Time Between Failures (MTBF), Downtime (MTBD) or Stoppages (MTBS).**

Early practice adopted “Failures”. Recognition of “Failures”, as not a strictly correct description of non-availability resulted in substitution of “Downtime”. In turn, “Downtime” does not allow for subtle interpretation of necessary, routine, planned maintenance, at least in some part, to be inherently unavoidable maintenance time with downtime reserved to break downs and major componentry replacement or repair. So the gravitation to “Stoppages” as a more strictly correct and acceptable term for time dedicated to maintenance activity. MTBS is used by Caterpillar Global Mining and was adopted throughout this thesis.

“Reliability” is the prime measure of maintenance efficacy. Expressed as **Mean Time Between Stoppages (MTBS)**, reliability is arguably the single most significant impact upon mechanical availability; and therefore upon productivity.

MTBS is a direct measure of reliability derived by the following equation:

MTBS = Operated Hours / Number Of Downtime Incidents

In the above equation the term “incidents” means machine stoppages or shutdowns.

Operational stoppages (or shutdowns) at shift change, lunch, or fueling (including top ups) or equipment testing are not counted. Maintenance or mechanically related stoppages – including scheduled lube service during preventative, planned maintenance; but not daily servicing or oil level checks - are counted.

Efficiency, in the mechanical maintenance context, is a measure of turnaround time or efficient use of downtime in getting a machine back into service. **Mean Time To Repair (MTTR),** as a benchmark, is the most significant direct measure of repair and/or maintenance efficiency.

MTTR is derived from the following equation:

MTTR = Downtime Hours / Number Of Downtime Incidents

All delays - mechanical downtime without active work - are included in MTTR.

Repair turnaround expressed as MTTR number has an impact upon mechanical availability. But, as shown, the impact from MTBS is more significant. That is, it is more effective in terms of productivity to realize reliability than maintenance efficiency. But both reliability and maintenance efficiency are essential for best practice and neither can be neglected in favour of the other

As noted above, production management use mechanical availability as a measure of operating time available relative to total time. Maintenance management and supervision use Mechanical Availability as an indicator of overall maintenance system performance and effectiveness. From the maintenance perspective mechanical availability is a function of MTBS and MTTR as shown by the following equation:

$$\textbf{MA \%} = \frac{\textbf{MTBS x 100}}{\textbf{(MTBS + MTTR)}}$$

Although this availability equation looks different than the one discussed in the production context, viz., MA% = [Operated Hours / Operated Hours + Downtime Hours] X 100, in reality they are the same. Substitution of the factors for MTBS and MTTR into the “production availability formula” will yield the “maintenance availability formula”.

The relationship between MTTR, MTBS and Mechanical Availability is shown in Table 3.66.

Table 3.66 Mechanical Availability v. Maintenance Benchmarks

MTTR Hours	3	4	5	6	8	10	12
MTBF Hours	Mechanical Availability %						
10	76.9	71.4	66.7	62.5	55.6	50.0	45.5
20	87.0	83.3	80.0	76.9	71.4	66.7	62.5
30	90.9	88.2	85.7	83.3	78.9	75.0	71.4
40	93.0	90.9	88.9	87.0	83.3	80.0	76.9
50	94.3	92.6	90.9	89.3	86.2	83.3	80.6
60	95.2	93.8	92.3	90.9	88.2	85.7	83.3
70	95.9	94.6	93.3	92.1	89.7	87.5	85.4
80	96.4	95.2	94.1	93.0	90.9	88.9	87.0
90	96.8	95.7	94.7	93.8	91.8	90.0	88.2
100	97.1	96.2	95.2	94.3	92.6	90.9	89.3

The mechanical availability data generated on Table 3.66 are graphed against MTBS for a range of MTTR in Figure 3.65; and against MTTR for a range of MTBS in Figure 3.66.

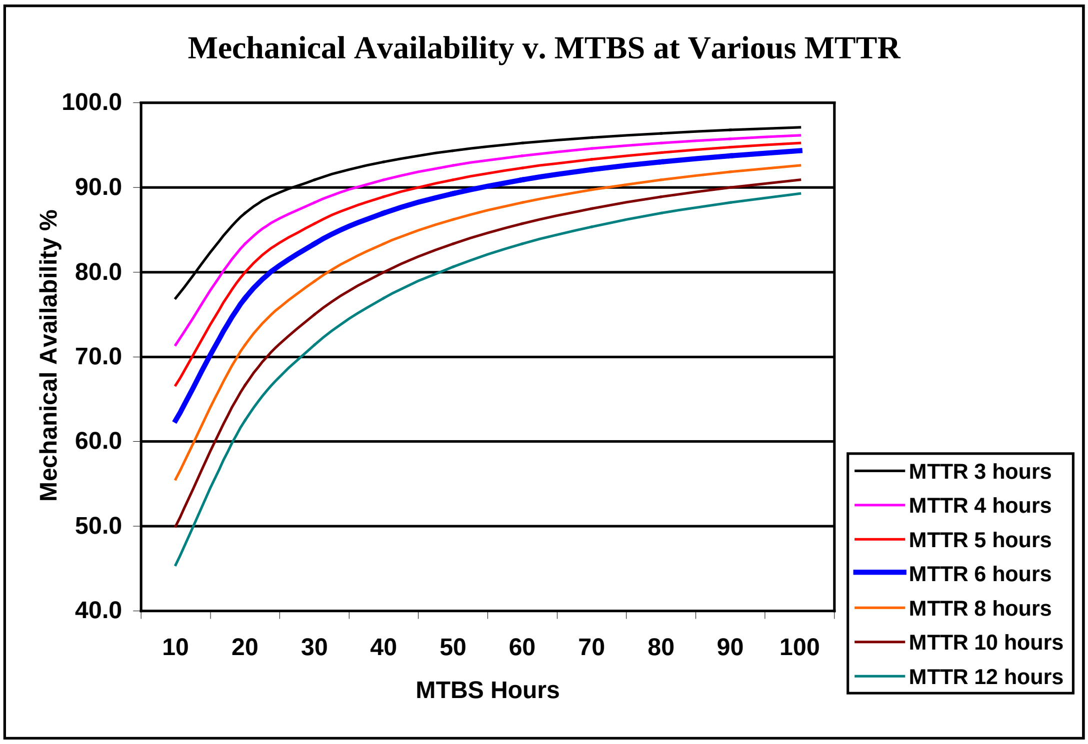


Figure 3.65 From Table 3.66

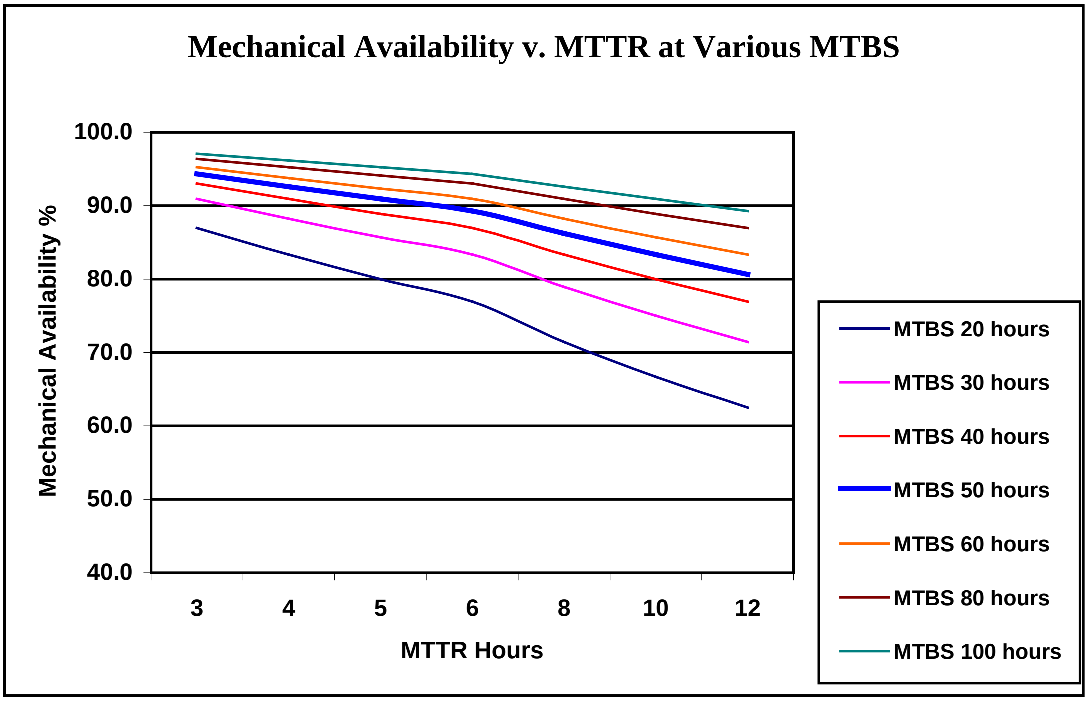


Figure 3.66 From Table 3.66

From Figure 3.65, previous page, it will be noted that, at MTBS of 70 hours or more, MA close to 90% or more will be realized over the full range of MTTR cases. From Figure 3.66, previous page, it will be noted that, at MTTR in excess of 6 hours realising MA of 90% or more, requires high reliability. The obvious implication of Figures 3.65 and 3.66 is that increased reliability reduces pressure on maintenance resources and, for a required availability, provides more flexibility in allocation of resources within the maintenance facility.

MA, MTBS and MTTR are important equipment management benchmarks. From an equipment selection perspective they indicate, for specific equipment, reliability and maintenance efficiency that might be expected; also the capability of equipment to reach production objectives.

There is a small ambiguity in the above normal practices for calculating MA, MTBS and MTTR that needs to be clarified by internal policy and identified for any site in the course of due diligence investigation. It was indicated above that stoppage for fuelling and lube top ups is not counted in compiling MTBS. Accordingly, the time for fuelling and lube top up is not counted as downtime. This is a maintenance viewpoint. Historically fuelling, lube top ups and concurrent opportunity-repair of minor defects have been considered maintenance activities. Omission of refueling and lube top up as an activity and time element from MTBS is likely justified by MTBS providing a measure of reliability. Unavoidable stopping of equipment for refueling and lube top up is obviously not a reflection on equipment reliability.

Discussion on time-activity definitions including the above-indicated ambiguity is continued in Section 3.6.

3.4.3 Benchmarks and Key Performance Indicators

Benchmarks (BM) and key performance indicators (KPI) were briefly introduced in Section 2.1.2, particularly Table 2.1 appended in Volume 2.

It is necessary to determine or assume suitable maintenance BM and KPI in the process of determining operating hours as an essential input to estimate production capacities of load and haul equipment for comparison with the mine production plan.

Table 3.67, lists BM and KPI recommended in 2003 by Caterpillar Global Mining for "World Class Operations".

Table 3.67 Benchmarks for World Class Operations
(Caterpillar Global Mining)

Bench Mark	BM Value/Comments
Mechanical Availability	92% (Pre-planned Component Replacement) 88% (Post-planned Component Replacement)
Mean Time To Repair	3 to 6 Hours
Mean Time Between Stoppages	80 Hours (Pre-planned Component Replacement) 60 Hours (Post-planned Component Replacement)
Production Utilisation	85% - 90%
Maintenance Ratio	0.2 Hours (Pre-planned Component Replacement) 0.3 Hours (Post-planned Component Replacement)
% Planned Activity	85% - 90%
Record Keeping	100%
Servicing Accuracy	95% within +/- 5% of Target

In the early stages of preparation for equipment selection, acceptable values for those BM itemized in Table 3.67 that impact on equipment production performance must be determined by company policy, internal consensus of stakeholders, or due diligence investigation of similar mining operations. In default of firm empirical or policy values, BM based on the standards indicated by Table 3.67 or from similar references may be utilised, at least as initial criteria.

It should be noted that the BM data in Table 3.67 are for planning and monitoring purposes. Utilisation of some BM as a basis for determining necessary maintenance resources should be circumspect. Particularly the Maintenance Ratio, i.e., the ratio of maintenance manhours to operating hours - both utilisation values - must only be used for comparison. What "maintenance hours" includes needs to be understood. Maintenance manhours, including maintenance planning and supervision established to ensure that the maintenance hours demand is satisfied, to realise expected MTTR, inflate the maintenance ratio up to as much as double the BM values in Table 3.67. Determination of maintenance manpower numbers is an operational requirement that likely can benefit from binomial probability modelling and simulation – activities beyond the scope of this thesis.

3.4.4 Maintain-And-Repair-Contracts

The following extract from a paper by the author is relevant.

“Throughout the history of contract open-pit mining, mobile equipment dealers have been offering an extension of the traditional business of sales (new and used), parts, offsite component/machine overhauls and training to include contract, onsite, total service of mobile plant. Until more recently the style of equipment design, labour costs and overheads, (and, particularly, reasonable availability of skilled maintenance labour) made it difficult for equipment dealers to compete with in-house routine servicing and maintenance of mobile equipment. This situation has changed. Currently both earthmoving contractors and owners are executing contracts with equipment dealers for a total onsite service – a maintain-and-repair-contract (MARC)” (Hardy #3, 2005).

The business cycle that favoured inclusion of supply, maintain and repair of mobile equipment and the many other peripheral supply or service opportunities in contracts offered to earth moving contractors has moved on to a phase where owners are considering alternative options. Mining contractors have traditionally managed the logistics of establishing equipment, maintenance facilities, a suitably skilled workforce and outsourced:

- Supply of equipment and finance;
- Major component rebuilds and equipment spares;
- Subcontracted tyre fitting and management service;
- Supply of tyres and accessories;
- Supply of fuel, lubricants and cleaning fluids;
- Down-the-hole service of explosives and supply of accessories;
- Supply of all other consumables;
- Sometimes blast hole drilling; and,
- Pit dewatering when included in their work.

In recent times there has been a tendency for owners to take over and in-house manage many of the supply and service functions historically included in mining contract works. Of interest is the degree of actual hands-on management control that owner miners are currently choosing to retain. Table 3.68- provided by Linton Kirk (Kirk, 2000), shows that recent conversions to owner mining have retained hands-on control of only a small proportion of total mining cost. For the two examples in Table 3.68 the new owner operators retained day-to-day management of only some 20% of total mining cost.

Owners have recognized that some 55% to 60% of the value of an earth moving contract for open pit mining is in the procurement of mobile plant and getting it on to the "ready line" (or "go line") available for work. The facility to execute a MARC, i.e., to partially outsource mining operations provides an alternative for owners.

Table 3.68 Cost Items As % of Total Mining Costs
(Kirk, 2000)

COST ITEM	KCGM %	ERNEST HENRY %
Equipment Ownership/Lease	**16**	**20**
Equipment Maintenance	**17**	**19**
Drill and Blast	**24**	**19**
Fuel and Lubricants	**13**	**12**
Tyres	**9**	**9**
TOTAL	**79**	**79**

Except for special cases where there is firm policy, cogent commercial reasons or an imperative need to outsource mining operations in total, such as limited capital funding for mining equipment ownership, owners may consider MARC-supported owner mining as an attractive alternative. The introduction of MARC services has added another dimension to both the procurement and maintenance of mobile mining equipment. "Original equipment manufacturers (OEM) and their equipment dealers can supply a complete service from initial supply, comprehensive support with parts and component rebuilds through to equipment on the "ready line" – that is, financing and MARC services." (Hardy#2, 2003)

In the process of load and haul equipment selection, the option of attaching a MARC to equipment purchase needs to be considered, analysed as necessary, and a firm policy acceptable to stakeholders and management determined as a plank in the equipment selection strategy platform. It may be that a MARC is carried as an option through to final equipment selection. If there is potential for a MARC to be an add-on to selected and procured equipment, the BM in Table 3.67 and additional KPI will take on increased importance. These measures will become contractual responsibilities and obligations of the MARC. The need for special consideration of maintenance strategy early in the equipment selection and procurement process is obvious.

3.4.5 Equipment Sales Agreements

Warranties, performance guarantees and formal agreements are necessary protective instruments for purchasers of mining equipment. It is important during the due diligence phase of equipment selection to establish reasonable expectancies that are neither optimistic nor too protective of operating performance in the future. In the earliest selection stages discussions may be initiated on warranty, particularly extensions, performance guarantees for new, relatively untried equipment models under consideration. At some stage, particularly if MARC is an option, formal agreement(s) may need to be prepared. Facilities for such specialised document preparation need to be arranged with the greatest possible lead-time to avoid unnecessary delays when the procurement process is nearing conclusion.

3.5 FLEET MATCHING, BUNCHING, QUEUING

3.5.1 Introduction

In the process of equipment selection it is necessary to utilise effective procedures and practices to arrive at a robust estimate of the necessary number of trucks in a fleet to satisfy the planned production programme. Procedures and practices adopted need to ensure that sufficient trucks are productive for sufficient time to fulfill production requirements.

Of necessity the number of loading equipment items and capacity are complementary considerations to resolve interactive effects with trucks that determine productivity of the combined fleet. Such interactives include number of bucket passes, bucket cycle times and other time components affecting truck loading time.

The discussion also extends to consideration of probabilistic analysis of truck availability for single and multiple loading equipment cases as a more effective alternative in comparison with traditional, deterministic practices.

A-productive Factors – Discounting Productivity

Productivity of loading and hauling equipment is reduced from theoretical to actual experienced performance by a number of a-productive factors. In Section 3.4 a-productive factors of mechanical availability and utilisation were discussed. This discussion is extended to further a-productive factors related to fleet matching and variability of truck loading and trip times – generally termed “bunching”.

Such a-productive factors can be considered intrinsic to loading and/or loading equipment where there are direct performance differences attributable to equipment or the normal operation of that equipment. In contrast a-productive effects can be considered non-intrinsic where there are extraneous effects not related to the capability of the equipment or the direct operation of it.

Examples of intrinsic effects are variabilities of engine power, power transmission of other driveline componentry, tyre outside diameter due to wear, NMW, quantities of debris on frames and carry back in truck bodies (and buckets particularly for wheel loaders); and operator efficiency that has been empirically been shown to be a function of haul distance, see Section 3.3.10 and Figure 3.58. Generally intrinsic a-productive effects are causes of "bunching" as described in literature and used in common operational terminology.

Non-intrinsic a-productive effects are the result of extraneous activity, sometimes related, but often unrelated, to load and haul equipment characteristics and capability. Examples include traffic and stationary obstructions on roads, road re-construction or construction of road extensions, general pit housekeeping, specific housekeeping at the working face, large particles in the face, road and trafficable area watering, providing access for other functions such as blasting agent and accessory transport, latent ground conditions that interrupt operations, re-routing one or more trucks from a specific duty cycle, assigning too many trucks – overtrucking - and any casual un-scheduled stoppages of loading equipment or trucks.

Terminology

The terms "bunching" and "queuing" are commonly used with overlapping meanings and sometimes for the same condition in load and haul systems. Particularly Caterpillar in the FPC Handbook (Caterpillar#2, 2004); and Dan Gove and Bill Morgan use the term "bunching" only in analysis and discussion that is focussed on optimizing truck and loader matching (Gove & Morgan, 1994). Elbrond describes, as an alternative to simulation, development of a deterministic process for calculating waiting time for trucks where a queue exists at the loader in terms of "queuing theory" (Elbrond, 1990). Ramani provides solutions to fleet matching using deterministic, (including "average delay" as a time element), statistical (probabilistic) and simulation methods without referring to either “bunching” or “queuing” (Ramani, 1990).

Christina Burt and co-authors (Burt et al, 2005), in an extended abstract, provide the following definitions:

“*Bunching Theory:* Bunching models capture the tendency of moving objects to bunch together when moving in a line. This is generally due to some of the objects being operated or moving more efficiently than others.

Queuing Theory: Queuing theory is the study of waiting times, lengths, and other properties of queues.”

The focus of this part of the research and thesis is on a-productive effects of “bunching” phenomena and what drives them. Wait time and other queuing issues are incidental to the analysis and interpretations. “Bunching” has been adopted herein as a generic term to cover the physical affect of any delay to trucks and loading equipment that causes loss of production.

Analysis Methods

Initial calculations of fleet estimates and matching are generally deterministic using expected (mean) values. Consideration of affects of a-productive factors that are independent random variables, such as mechanical non-availability, requires application of probabilistic techniques to realise acceptable results for fleet matching.

It is necessary to realize an acceptable treatment of the a-productive effect of interaction between independent, randomly variable time-performance characteristics of loading equipment and trucks that manifests as “bunching”. This can be achieved by application of empirical deterministic relationships or introduction of simulation techniques utilizing probabilistic techniques modelling empirical operational data.

This section generally follows the indicated analysis process including deterministic application of empirically based criteria to quantify “bunching” effects.

3.5.2 Preliminary Fleet Numbers

Preliminaries

The following preliminary analysis assumes that all variables are expected (mean) values. Any practical application of the relationships and equations is necessarily deterministic. Using the symbols as defined in Sections 3.3.9 and 3.3.10, truck-loading time can be expressed in terms of bucket cycle times as:

- From loading equipment perspective – $T_C = T_{EX} + n \bullet T_{BC}$
- From truck perspective – $T_L = T_F + (n-1) \bullet T_{BC}$

  n = number of bucket cycles.

Truck trip time can be considered in terms of components:

- $T_T = T_V + T_D + T_S + T_L$
- $T_V = T_{VL} + T_{VE}$ - subscripts "VL" and "VE" indicate "loaded" and "empty" respectively.

In the process of load-and-haul equipment selection the "which comes first?" question must be resolved. Many practitioners and authors indicate that loading equipment selection is resolved, at least generally, before proceeding to truck selection. It is the author's opinion that loading equipment selection considerations should only proceed to the limited extent necessary to allow progress of truck selection. When truck selection has been resolved to an acceptable conclusion, then selection of loading equipment can be concluded. Of course, concluding loading equipment selection should, prudently, be followed by an audit of truck selection criteria and process to ensure that the fleet configuration is optimum.

Some estimating formulae offered by other authors include (symbols for variables substituted to be consistent with usage herein; also units assumed consistent with context of equations):

Truck loading time - T_L:

$$T_L = PL/P_e \qquad (1)$$

PL = truck payload

P_e = shovel productivity

$$T_L = PL \bullet T_{BC}/B_{cap} \qquad (2)$$

T_{BC} = bucket cycle time

B_{cap} = bucket capacity utilised (after application of fill factor)

Travel Time – T_V, Haul Distance – D_H, Relationships:

Subscript "L" = "loaded"

Subscript "E" = "empty"

$$T_{VL} = D_{HL}/S_{AVL}$$

$$T_{VE} = D_{HE}/S_{AVE}$$

$$T_V = (D_{HL} + D_{HE})/S_{AV} \qquad (3)$$

T_V = travel time

D_{HL} = haul distance loaded

D_{HE} = haul distance empty

S_{AV} = average speed

From Table 3.63 - $S_{AV} \approx 21$ kph

$$T_V \text{ minutes} = D_H \text{ metres x } 60/21{,}000$$

$$= 0.00286 \text{ x } D_H \qquad (4)$$

Consider a loader has loaded a single truck that is *en route* to dump and return. Assuming for the purposes of analysis that all variables are expected (mean) values; also assuming that the "return" time is substantially larger than the manoeuvre and spot plus loading time - then the loader experiences waiting time – W_L - as follows:

$$W_L = (T_V + T_D) – (T_S + T_L)$$

Generalized for N trucks in the fleet (N is always an integer) – and a single loading unit – mean loader waiting time W is derived from:

$$W = (T_V + T_D) – (N-1)(T_S + T_L) \qquad (5)$$

If the loader does not experience waiting time W = 0. Simple algebraic manipulation yields:

$$N = T_T/(T_S + T_L) \qquad (6)$$

If $W \neq 0$; but $0 < W < (T_S + T_L)$ then:

$$W = (T_V + T_D + T_S + T_L) – N \bullet (T_S + T_L)$$

$$N + W/(T_S + T_L) = T_T/(T_S + T_L) \text{ where } 0 < W/(T_S + T_L) < 1.0$$

$[N]^* = [T_T/(T_S + T_L)]^*$ (7)

$[N]^*$ = number of trucks after rounding (up or down)

$[T_T/(T_S + T_L)]^*$ = signifies rounding up or down to an integer.

The above equations (1) to (7), for a single truck loading group are offered by, or derived from, a number of references particularly, Ramani (Ramani, 1990) and Fourie and Dohm (Fourie, 1992).

Initial Fleet Estimates and Matching to Loading Equipment

Estimating truck fleet requirements to match a single loading unit to satisfy a planned production programme is an iterative process. Initial estimates, to be used as a basis for analysis to accommodate a-productive factors discussed, can be developed by several methods including:

1. Application of equation (7) where the total truck trip and truck loading times, T_T and T_L are known or can be estimated with reasonable accuracy;
2. In the absence of T_T and T_L the following method for deep open pits can be applied.

   The depth of pit is multiplied by the access ramp grade expressed as a percentage; and, to estimate one-way haul distance, apply one-way haul distance factor K_D - for pits to 100 metres $K_D = 2$, for pits to 200 metres $K_D = 1.8$ and for pits to 500 metres $K_D = 1.6$.

   To establish an estimated total haul double the distance.

   Take 0.3% to derive total travel minutes.

   Add fixed times 0.75 + 1.0 +3.0 minutes for manoeuvre and spot, (fixed times can be modified if desired but reasonable changes will be only of small effect in the outcome) manoeuvre and dump and loading time to arrive at estimated truck trip time.

   Initial Estimate of Number of Trucks - N_{ie} - for a single loading unit:

   $N_{ie} = (0.003 \times D_{Hie} \times K_D \times 2 + T_S + T_L + T_D)/(T_S + T_L)$ (8)

The following examples illustrate the method:

Example 1: 50m deep pit, 10% access ramp $K_D = 2$.

N_{ie} = (50 x 10 x 2 x 2 x 0.003 +4.75)/3.75

= 10.75/3.75

= 2.87 trucks

Example 2: 200m deep pit, 10% access ramp K_D = 1.8.

N_{ie} = (200 x 10 x 1.8 x 2 x 0.003 +4.75)/3.75

= 26.35/3.75

= 7.03 trucks

Example 3: 500m deep pit, 10% access ramp K_D = 1.6.

N_{ie} = (500 x 10 x 1.6 x 2 x 0.003 +4.75)/3.75

= 52.75/3.75

= 14.07 trucks

It will be noted that the estimating technique using equation (8) is a simple extension of equations (1) to (7). Reference to Table 3.63, shows the second of the estimating methods (in the absence of T_T and T_L) is reasonably consistent with truck trip times as generated by Caterpillar's FPC and sufficiently accurate for initial estimates.

It must be emphasized that the above-recommended "rule-of-thumb" estimating method is confined to deep open pits for a single, or each, loading unit. Preliminary estimation methods for open pits with configurations and geometry differing from pits normally described as "deep" can be developed similarly.

Initial estimates are not an essential step in the equipment selection process; but, in the absence of intuitive insight and experience, may be necessary to narrow down the options for suitable trucks and loading equipment to match to required productivity. Indicative truck numbers established from equation (8) need to be checked against truck numbers from equation (7) when T_T and T_L are ultimately known.

Basic Truck Fleet Numbers

The following discussion is in context with deep open pit mining. Other open pit configurations can be similarly treated with appropriate adjustments to process. Assuming:

- Possession of a production plan;

- Indicated necessary productivity; and
- Some preliminary thoughts on the range of applicable trucks:

It is necessary to set up some typical haul road profiles through the depth of the pit to provide a range of cases for fleet selection and matching determinations.

The following description is more a mine planning activity than equipment selection. There is interactive overlap where discrimination between planning and equipment selection is practically difficult.

Sufficient typical haul profiles need to be analysed to provide a suite of estimated performance data for comprehensive truck fleet and loading equipment analysis.

Suggested bench levels for typical haul road profiles:

- Say two benches down (more or less as necessary) to cover the vertical zone through clearing, topsoil and subsoil harvesting for rehabilitation purposes. Through the upper weathered zone where lateritic-cemented horizons may be harvested for road building/surfacing materials; and where highly weathered zones may be "free digging" – where some of barren pit overburden may be mined by scrapers as an option to loading and truck haulage.
- At elevations where there are substantial changes in the weathering profile from weathered to transition to fresh rock – where there will be significant changes in bank and loose density and voids ratio of material to be hauled and necessary fragmentation methods must change radically, e.g., introduction of drilling and blasting.
- At one-third of the depth – approximately 50% of pit volume remains to be mined.
- At half the depth – approximately 25% of pit volume remains to be mined.
- At planned pit bottom.

For the suite of haul road profiles, develop for the pre-selected range of alternative trucks, truck trip times, loading times and other time components necessary for deterministic estimating of basic fleet numbers. It is common practice to use commercially developed software such as Caterpillar's FPC (see Section 3.3.10) or Runge Mining's Talsim. It should be noted that both of these software packages tend to be moderately conservative in application as a result of fortuitous assumptions built into the basic algorithms. But the degree of conservatism is sufficiently small to be ignored.

As inferred by the above discussion, load and haul equipment must accommodate ranges of operational conditions and duties for loading and hauling. Table 3.69 indicates, for a hypothetical deep open-pit mining project, a suitable planning process necessary to arrive at a reasonably limited range of operating options for applicable trucks.

Table 3.69 Load and Haul Equipment Preliminary Selection Planning

Operational Description	Productivity Phase See Figure 3.67	Comments
Clearing, detritals, tertiary sediments, laterites and highly weathered strata. Mining partial benches to set pit perimeter, access ramp crests, safety structures and stockpiling of road construction and rehabilitation materials. Only includes ore mining if the ore zone outcrops or has little cover.	**A**	Primary truck fleet may not be available. Possibly contract clearing and pre-strip of greenfields open pit. Generally smaller trucks on short hauls, nominal 3 and 4 pass loading. Scrapers could be an alternative for short hauls depending on pit stratigraphy and site geography.
Weathered zone to transition – depending on cover to first ore could be a large proportion of pit volume – otherwise included in Phase A	**A or B1**	Select load and haul fleet for bench at one-third pit depth that includes weathered zone (Work up or down for reduced or additional trucks)
Transition – weathered to fresh rock	**B2**	Select load and haul fleet for weathered or transition volume at one-third of pit depth (Work up or down for reduced or additional trucks)
Fresh rock to, say, 75 % of pit volume – more specifically where waste : ore ratio reduces significantly	**B3**	Add trucks to sustain fleet productivity
Fresh rock balance to “goodbye bench”	**C**	Allow productivity to diminish as waste: ore ratio decreases by fixing or reducing truck numbers. For multiple-loader fleets reducing loading equipment numbers and matching trucks is an option.

Figure 3.67 illustrates a technique generally adopted by the author to arrive at a range of load and haul productivities necessary to effectively complete a production plan and provide equipment selection criteria. The three basic production phases in a deep open pit mine are indicated on Figure 3.67; also described in Table 3.69.

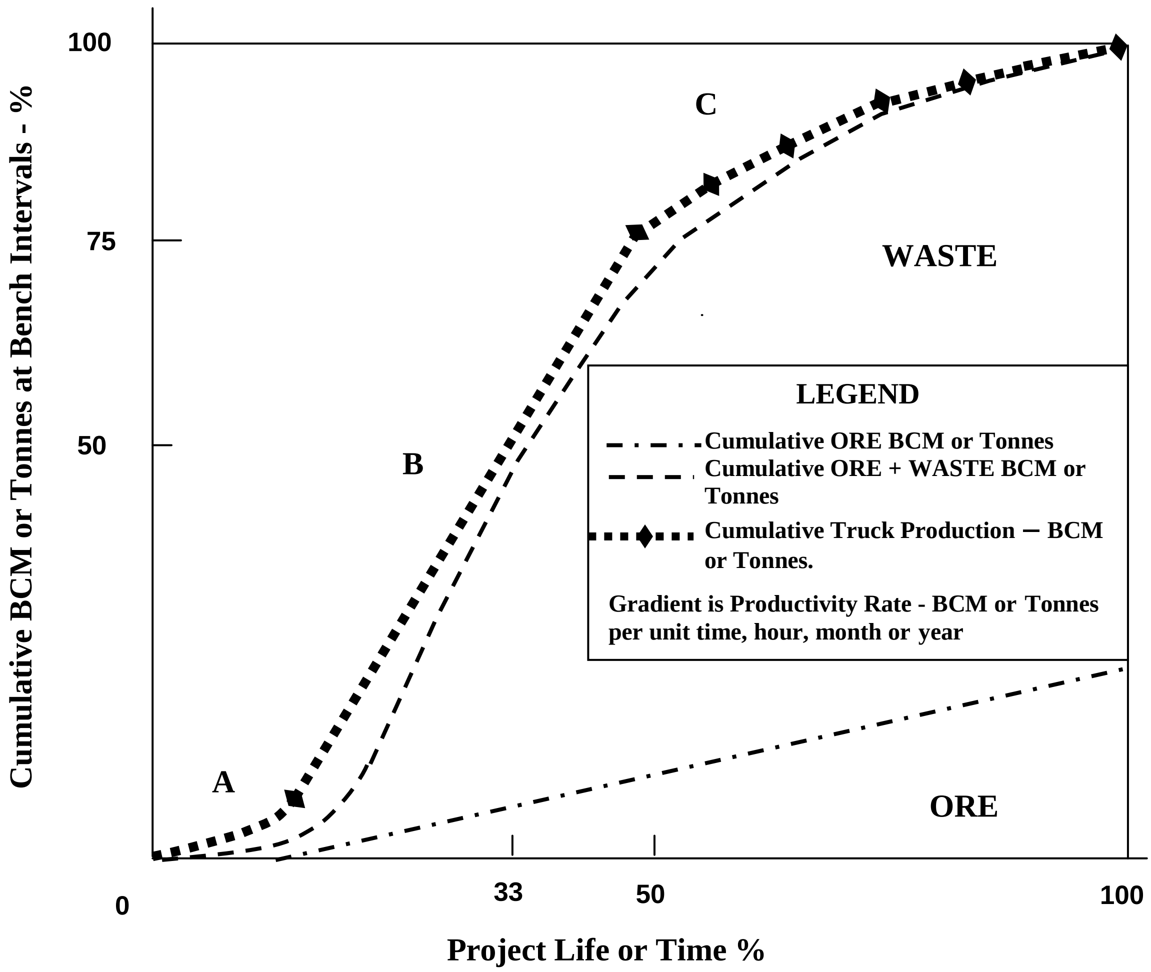


Figure 3.67 Conceptual Cumulative Production Plan And Matching Equipment Productivity

Phase A – as described in Table 3.69 – generally provides time to procure and establish main load and haul fleet on site and work up to expected productivity.

Phase B – a period of constant load and haul productivity sustained by planned addition of trucks to accommodate extending haul distances. Changing pit stratigraphy, with significant discrete changes to expected values for bank and loose density, is often a complication requiring strategic bucket selection for loading equipment and truck body design and volume selection for best practical and economic outcomes. Planning and equipment selection complications may require Phase B to be considered in several sub-phases as indicated in Table 3.69.

Phase C is essentially tuning the mining operation to best economics with existing equipment during a period of diminishing production requirements to mine closure.

It is obvious that load and haul equipment should primarily be selected to satisfy the necessary mining duties for the largest proportion of production volume that can be identified with discretely similar mining properties – preferably at constant productivity. The ultimate mining goal is to produce a stream of run-of-mine ore, or more generally mineral product that satisfies the needs of downstream sizing, processing and transport with ever-increasing concentration through to final mining commodity production. Generally, proceeding downstream from the mining operation, processing functions receive feed-rates, with reducing dispersion, trending towards the central limit, i.e., mean feed rate. Accordingly load and haul equipment selected for deep open pit mines, in fact for all mines, must satisfy the specific needs of downstream facilities including practically constant production rate of mineral product.

Mine planning activities will yield a suite of productivity requirements for mining yields at differing horizons through the depth of the open pit. Each discrete productivity requirement needs to be satisfied by a number of trucks. Each truck has a productivity of:

$$V_T \text{ tonnes or BCM per hour} = PL \bullet 60/T_T \qquad (9)$$

V_T = Productivity per truck tonnes per hour (TPH) or BCM per hour (BCM/H)

PL = truck payload in tonnes of BCM

T_T = Truck trip time – minutes

Phase Productivity V_{PH} TPH or BCM/H is an outcome of planning activities described above.

Basic number of trucks – N_B:

$$N_B = V_{PH}/V_T \qquad (10)$$

Equipment matching to a productivity requirement determined by planning is generally an iterative process. As will be seen below, probabilistic analysis of truck and loading equipment availability tends to discount expected production performance based on deterministic analysis. Accordingly, intuitive, experienced consideration of planning-derived productivity requirements may be prudently increased by a contingency allowance. That is, V_{PH} should include any necessary allowances to ensure that the basic number of trucks provides for all a-productive effects other than those that are covered by subsequent probabilistic analysis. It should be noted that, in default of providing contingencies to satisfy intuitive and experienced expectations, the same outcome will be achieved by iterative process.

By application of the above, or techniques of similar effect, basic load and haul fleet numbers are derived, specifically trucks. It should be noted that, for this truck fleet estimation, T_T is based on notional or first-estimate loading equipment characteristics. Number of loading equipment items can be estimated using equation (7) to determine the number of trucks that can be loaded by a single loading unit. Estimates of basic truck numbers are substantially independent of loading equipment options at this stage of the selection process. Particularly the resulting truck numbers are independent of the number of loading units.

Treatment of A-productive Factors

Having established the basic fleet numbers from equation (10), a number of a-productive effects need to be considered including the following five factors:

1. Mechanical non-availability to accommodate maintenance of equipment, specifically trucks.
2. Non-utilisation due to inefficient management of available equipment.
3. Operator inefficiency as discussed in Section 3.3.10 and illustrated by Figure 3.58.
4. Loading and hauling equipment mismatch, i.e., insufficient trucks so that loading equipment productivity is reduced below expectancy by having to wait for trucks; or excess trucks so that unit truck productivity reduces below expectancy by trucks having to wait to be loaded.
5. Reduced load and haul performance due to “bunching” caused by individually variable truck trip times and loading equipment bucket cycle time. That is, variable intrinsic performance of individual equipment items as discussed below - affecting both truck and loading equipment performance that results in the hypothetical “rhythmical” cyclical load and haul system deteriorating to a “bunched” system. There is a tendency for the time lost by individual equipment items to interfere with following items. Unless there is

provision and policy provides for trucks to pass, all units in the cyclical system will tend to gravitate to the performance of the slowest truck.

Discussing each of the five factors in terms of treatment:

Mechanical non-availability:

Maintenance activities effectively isolate trucks from productive operations. Assuming scheduled work hours are a practical maximum, replacing trucks sterilized by maintenance by adding to the fleet is the appropriate treatment. Mechanical availability is an independent random variable.

Non-utilisation:

Non-utilisation of load and haul equipment is also an independent random variable. In a greenfields project and where all production and maintenance planning is based on deterministic analysis then equipment utilisation can be viewed as a parallel to mechanical availability. For a brownfields extension of existing operations or equipment replacement selection process, care needs to taken to ensure that estimated productivity requirements do not already partly or totally compensate for non-utilisation. Adding trucks to compensate for non-utilisation assumes that a similarly inefficient operational model is retained for all trucks. That is, the operational circumstances and practices that result in non-utilisation are the same for every truck, say; but they may differ between types of equipment. So loading equipment can experience differing utilisation characteristics to trucks and other equipment categories.

Operator inefficiency:

There is empirical evidence that operator inefficiency is haul distance dependent with efficiency increasing with haul distance. Figure 3.58 indicates that, ignoring one-way hauls of less than 1,000 metres, expected operator efficiency is some 90% for short hauls increasing to 97% for longest hauls. The effect of operator inefficiency is to generally and modestly detune equipment performance. Operator inefficiency is an independent, random variable and accordingly could be treated stochastically. The nature of operator inefficiency is an outcome of the interaction between the operator and the equipment unit through the medium of the truck control system. Maximum performance is a result of best possible response. As skilled as operators may be, there will tend to be some small lapses in time response that manifest as operator inefficiency. It is the author's opinion that corrections for operator inefficiency should be made where the time lapses manifest, i.e., to truck trip time or loading equipment cycle or loading time. For the purposes of discussion and analysis herein it is assumed that allowance for operator inefficiency is made to truck trip times and loading times, i.e., as a contingency allowance included in target productivity used to determine basic truck numbers.

Fleet matching:

Fleet matching is generally treated deterministically in terms of expected (mean) performance values for both loading equipment and trucks. The perfect match point is hypothetical. It is where truck trip and loading times have an exact integer relationship: where there is no waiting time experienced by loading equipment between truck arrivals; also returning trucks do not experience waiting time before loading. Generally, the theoretical truck fleet match will not be an exact integer. There is practical necessity to round up to overtruck or down to undertruck. These issues are discussed in more detail below.

Bunching:

Productivity effects of "Bunching" are complex and vary substantially through the range from significantly overtrucked to undertrucked – or "over-shoveled". Discussion in Sections 3.5.3, 3.5.4 and 3.5.5 describe the implications of the various possible operating circumstances, resulting "bunching" effect and productivity implications.

Treatment of basic truck numbers based on productivity requirements to arrive at a fleet number that will compensate for maintenance non-availability; also to allow for non-utilisation has been traditionally deterministic. A mechanical availability of, say, 90% and an independent utilisation of, say, 90% results in a combined factor of 81%. As noted care needs to be taken to ensure that non-utilisation and perhaps mechanical non-availability is not double counted when using empirical data in cases such as "brownfields": development or replacement.

$$N_{FD} = N_B/(MA \bullet U) \qquad (11)$$

MA = mechanical availability as defined in Section 3.4.

U = utilisation as discussed in Section 3.4 and in this Section 3.5

N_{FD} = deterministic fleet number.

Example: If a basic truck fleet of 15 trucks (N_B) is required to meet the required productivity, application of equation (11) estimates fleet number as:

N_{FD} = 15/0.81 = 18.52 – say 19 trucks where MA = 0.9 and U = 0.9.

In practice this would be found to be an insufficient truck fleet to cover for MA and U at the assumed values. Equation (11) assumes that MA, also U or the combination of them are constant values consistent with empirical outcomes when, in fact, both MA, U and their combination are independent random variables that can be modelled by a binomial distribution. A probabilistic treatment provides more adequately for random service outages for maintenance and to allow for non-utilisation. References that support probabilistic treatment of truck number determination and truck hours analysis including (Bohnet, 1992, p1535), (Hays, 1990, p684), and (Ramani, 1990, p727).

The binomial probability function can be expressed as

$$P_{x;N} = {}^{N}C_x \bullet p_x \bullet q_{N-x} \qquad \text{(A)}$$

$P_{x;N}$ = Probability of exactly x experiences from a population of N possibilities

${}^{N}C_x$ = Combinations of N experiences taken x at a time

p_x = Probability of a single experience

q_{N-x} =Probability of a single non-experience.

Background and development of equation (A) can be found in standard mathematical references. (Chou, 1969), or (Devore, 1999). The above symbols are generally consistent with the referenced texts.

Ramani's notation is generally adopted for the following discussion. Rewriting equation (A) accordingly:

$$P_n = {}^{N}C_n \bullet (P_a)^n \bullet (P_{na})^{N-n} \qquad (12)$$

P_n = probability that exactly n units are available

${}^{N}C_n$ = combination of N units taken n at a time ($n \leq N$)

P_a = probability that a single unit is available

P_{na} = probability that a single unit is not available = $(1 - P_a)$.

Also:

$$P_{Ln} = {}_{x=n}\sum^{N}\ {}^{N}C_x \bullet (P_a)^x \bullet (P_{na})^{N-x} \qquad (13)$$

P_{Ln} = probability that at least n units will be available.

(Ramani, 1990, p727)

Equations (12) and (13) have been developed in a spreadsheet with worksheets for a range of discrete MA or MA . U factors. Tables 3.70. and 3.71, Volume 2, Appendices, are examples - based on MA . U = 0.8 - referred to in the following discussion. A copy of the spreadsheet file labeled “Table 3.70” is on the CD inside the back cover to this thesis volume. Readers can change the factor on any worksheet to any value ($0 \leq$ MA . U ≤ 1) of their choice.

Example: Assuming a basic truck fleet of 15 trucks (N_B) is required to meet the required productivity as before, application of equation (13), through the facility of Table 3.71, appended in Volume 2, estimates fleet number as in Table 3.72,.

The range of trucks in the fleet to ensure the basic number of units to meet productivity requirements in Table 3.72 illustrates the obvious need for best mechanical availability and utilisation to reduce the investment in trucks for any chosen level of probability.

Table 3.72 - Probabilistic Determination of Fleet Units
For a range of - MA . U
Basic fleet units for productivity = 15
NUMBER OF TRUCKS IN FLEET FOR 15 AVAILABLE & UTILISED

MA . U	$0.90 \leq P_{Ln} < 0.95$	$0.95 \leq P_{Ln} < 0.99$	$P_{Ln} \geq 0.99$
0.80	**22**	**23**	**25**
0.85	**20**	**21**	**23**
0.90	**18**	**19**	**21**
0.95	**17**	**18**	**19**

Conversely, Table 3.72 illustrates that failing to provide for achievable mechanical availability or utilisation, i.e., over-optimism at the time of equipment selection will manifest as apparent undertrucking even though deterministic calculations indicate that truck fleet numbers are adequate.

Truck and Shovel Operating Hours

Availability of loading and hauling equipment being binomially distributed, Tables 3.70 and 3.71, appended in Volume 2, can be used to estimate loading hours and truck hours for both single and multiple loading units as follows:

Example: A single loading unit requiring a basic 5 trucks for productivity with a fleet of 8 trucks.

Assume that operations are scheduled for 10 effective hours per shift. Using same availability and utilisation values for loading equipment and trucks – 0.8 combined - calculate effective truck hours and loading hours including load and haul efficiency.

Shovel hours available to load trucks = 0.8 . 10 = 8 hours

Truck hours from Table 3.70:

		Available Truck Hours		
8 trucks	0.168	= 0.168.8.10 = 13.44		
7 trucks	0.335	= 0.335.7.10 = 23.45	**Proportion of Time. Hours . Trucks**	
6 trucks	0.294	= 0.294.6.10 = 17.64	**Effective Truck Hours**	
5 trucks	0.147	= 0.147.5.10 = 7.35	0.944 . 5 . 8	37.8
4 trucks	0.046	= 0.046.4.10 = 1.84	0.046 . 4 . 8	1.5
3 trucks	0.009	= 0.009.3.10 = 0.27	0.009 . 3 . 8	0.2
2 trucks	0.001	= 0.001.2.10 = 0.02	0.001 . 2 . 8	-
		Truck Hours Available 64.01	**Total Truck Hours Utilised by Shovel**	**39.5 = 61.7%**

Shovel Hours:	**Available Hours**		**Effect. Hours**	**Effect. Productivity**
	0.944 . 8 =	7.56	. 1.0 = 7.56	
	0.046 . 8 =	0.37	. 0.8 = 0.29	
	0.009 . 8 =	0.07	. 0.6 = 0.04	
Total Shovel Hours		**8.00**	**= 7.89**	**= 98.6%**

Comments:

With a combined availability and utilisation of 0.8 this is the probability that the shovel will be available. For a shift of 10 scheduled hours, the shovel will probably only be available 8 hours. So trucks experience the average 8 hours of shovel availability as a shift averaging 8 hours duration. The single shovel can only load 5 trucks and then only when trucks are available and utilised. Additional available trucks will be on standby or in a queue so the shovel is overtrucked. When available trucks fall below 5 the shovel is undertrucked. So load and haul productivity reduces by a small percentage. It will be noted that the example of probabilistic calculation of available truck hours and shovel hours is trivial. Each truck is available 8 hours of scheduled 10 hours. So available truck hours of 8 trucks is 8 • 8 = 64 hours. Similarly the shovel is available 8 hours.

Example: **A single loading unit requiring a basic 5 trucks for productivity with a fleet of 7 trucks.**

Assume that operations are scheduled for 10 effective hours per shift. Using same availability and utilisation values for loading equipment and trucks – 0.8 combined - calculate effective truck hours and loading hours including load and haul efficiency.

Shovel hours available to load trucks = 0.8 • 10 = 8 hours

Truck hours from Table 3.70:

Proportion of Time		**• Hours • Trucks**		**Effective Truck Hours**
7 trucks	0.210			
6 trucks	0.367			
5 trucks	0.275	0.852	5 • 8	34.1
4 trucks	0.115	0.115	4 • 8	3.7
3 trucks	0.029	0.029	3 • 8	0.7
2 trucks	0.004	0.004	2 • 8	0.1
Total Truck Hours Utilised by Shovels				**38.6 = 68.9%**

Shovel Hours:	**Avail. Hours**	**Effect. Hours**	**Effect. Productivity**
0.852 • 8 =	6.82 • 1.0	= 6.82	
0.115 • 8 =	0.92 • 0.8	= 0.74	
0.029 • 8 =	0.23 • 0.6	= 0.14	
0.004 • 8 =	0.03 • 0.4	= 0.01	
Total Shovel Hours	**8.00**	**= 7.71**	**= 96.4%**

Comment:

Again, the single shovel can only load 5 trucks. Additional available trucks will be on standby, or in a queue so the shovel is overtrucked. But, with one less truck, load and haul productivity reduces by a further small percentage. So, as the truck fleet reduces towards the basic number of trucks, load and haul productivity continues to reduce and undertrucking of loading equipment occurs more frequently. This is not always an undesirable operational state, as trucking efficiency tends to increase with potentially lower unit costs. This issue is further discussed below and in Sections 5.5.2 through 5.5.4.

Example: Multiple loading units, 4 provided to effectively operate approximately 3, requiring a basic 15 trucks for productivity with a fleet of 23 trucks.

It is assumed that operations are scheduled for 10 effective hours per shift. Using the same availability and utilisation values for loading equipment and trucks, as in previous examples – 0.8 combined - calculate effective truck hours and loading hours including load and haul efficiency.

Truck hours from Table 3.70:				**Available**	**Utilised by Shovels**
23 trucks	0.006	23 • 0.006 • 10	= 1.38		
22 trucks	0.034	similarly	= 7.48		
21 trucks	0.093		= 19.53		
20 trucks	0.163		= 32.60	60.99	0.296 • 20 • 10 = 59.2
19 trucks	0.204		= 38.76		0.114 • 19 • 10 = 21.7
18 trucks	0.194		= 34.92		
17 trucks	0.145		= 24.65		
16 trucks	0.088		= 14.08		
15 trucks	0.044		= 6.60	119.01	0.410 • 15 • 10 = 61.5
14 trucks	0.018		= 2.52		0.151 • 10 • 10 = 15.1
13 trucks	0.006		= 0.78		0.003 • 10 • 10 = 0.3
12 trucks	0.002		= 0.24		
11 trucks	0			3.54	0.025 • 5 • 10 =1.2
Truck Hours Available (23 • 10 • 0.8 =184)				**183.54**	**Utilised 159.0 = 86.4%**

Shovel Hours Available to Load trucks:

Deterministic estimate = 0.8 • 10 • 4 = 32 hours

From Table 3.70		**Shovel Hours**		**Effective Shovel Hours**
4 shovels	0.410	0.410 • 4 • 10	= 16.40	40 • 0.296 • 1.00 = 11.84
				40 • 0.114 • 0.95 = 4.32
3 shovels	0.410	0.410 • 3 • 10	= 12.30	30 • 0.410 • 1.00 = 12.30
2 shovels	0.154	0.154 • 2 • 10	= 3.08	20 • 0.151 • 1.00 = 3.02
				20 • 0.003 • 1.00 = 0.06
1 shovel	0.025	0.025 • 1 • 10	= 0.25	10 • 0.025 • 1.00 = 0.25
0 shovels	0.001			
Total Available Shovel Hours			**= 32.03**	**Effect. Hours 31.79**
				Effective Productivity 99.3%

Comment:

With a number of loading units there is potential for truck loading at any time within a scheduled shift. Total effective shovel hours using Table 3.70, appended in Volume 2, indicates that there are probably 3.2 shovels available as expected from deterministic calculation of mechanical availability and utilisation. Total utilised truck hours indicate an average of nearly 5 trucks at the shovel as expected. The calculations indicate that occasionally the shovels are undertrucked. For some 30% of the time 4 shovels are available, they will probably be limited to 19 trucks. As shovels go out of service operating shovels, as expected, are probably overtrucked.

It will be noted that, for all examples, productive truck hours are less than available truck hours due to independence and randomness of loading and hauling operations.

Consideration was given to improving productive hours relative to available hours. Two methods are hypothesized as follows:

1. Reduce the number of trucks so running undertrucked with eventually no waiting time for trucks so improved hauling efficiency but at the cost of loading productivity.

2. Increasing the number of loading items selected at a reduced size and bucket capacity to load in an extra pass or more, modestly increase the truck trip time T_T significantly increase truck loading time T_L but increasing the probability of an exact number of loading equipment items being available to load trucks.

The first option only appeals where efficiency of truck operations is highest and unit costs are likely lowest, where the productivity reduction due to reduced loading efficiency is strategically acceptable. The second option only appeals where planning can accommodate the additional loading unit or units; where the increased unit cost of loading with smaller units is acceptably offset by the improved trucking efficiency and consequent reduced hauling unit cost. The selected configuration of loading items and trucks needs to be tailored to the specific mining operation and the necessary load and haul duties to be provided by the equipment fleet.

Two further recasts of the example for multiple loading items, with all assumed parameters the same, were calculated based on 3 shovels and 5 shovels. Appropriate adjustments were made to truck numbers accommodated by an exact number of shovels. The results are summarized in Table 3.73.

The data in Table 3.73 are working estimates by the author and should be considered as indicative but realistically comparative. The data does illustrate that, for the same overall production objective, increasing the number of loading units appears to increase hauling efficiency by more than the expected decrease due to extra loading pass or passes for smaller loading equipment.

Table 3.73 Summary of Multiple Shovel Cases

Loading Hours and Related Data - 23 Trucks Available Hours ≈184

4 Loading Units Taken As Benchmark

Number of Loading Units	Probable Truck Hours Utilised by Shovels	Indicated Load and Haul Productivity	Truck Productivity Factor – Based on Estimated T_T Change
3	**148.8**	**0.94**	**1.05**
4	**159.0**	**1.00**	**1.00**
5	**173.3**	**1.09**	**0.96**

Further analysis in conjunction with the FPC results discussed in Section 3.3.10 indicated an increase of some 1% to 3% in T_T due to increasing nominal passes from 4 to 5, with only small increase of generally < 0.5% due to nominal passes increasing from 5 to 6. To reduce number of passes, bucket capacity must increase. So loading unit size increases. Consequently, bucket cycle times and truck loading times are slightly increased. Consequently reducing bucket passes by one or more does not realise time saving of the total time for a bucket cycle. The relatively smaller increases in loading equipment cycle time for increased number of passes, at least partially, offsets the time for an extra pass or passes.

As discussed in Sections 3.2.8 and 3.3.9 there is also the significant benefit of reduced payload dispersion that results from increasing number of passes.

Interpretation

In this section the treatment of a-productive factors including mechanical availability and utilisation has been discussed in terms of probabilistic techniques. Application of these techniques provides realistic load and haul equipment fleet numbers to realise a necessary productivity to achieve planned production. The need for iterative process and consideration of all relevant options to arrive at the optimum selection of fleet numbers, including loading equipment and truck capacities, has been demonstrated.

The need for circumspection in endorsing the *de facto* industry standard of 4-pass loading (three-pass loading in some schools of thought) has again been illustrated – this time in the process of optimizing the number of shovels where multiple loading equipment items are necessary.

3.5.3 Fleet Matching - Overtrucking, & Undertrucking

"Perfect Match Point"

The hypothetical state of "perfect match" is where loading capability in terms of number of trucks loaded per unit of time exactly matches the number of trucks in the fleet. So potential loading productivity exactly matches potential truck productivity – where trucks do not have to wait to be loaded and loading equipment does not have to wait for trucks. The "perfect-match point" is illustrated by Figure 3.68.

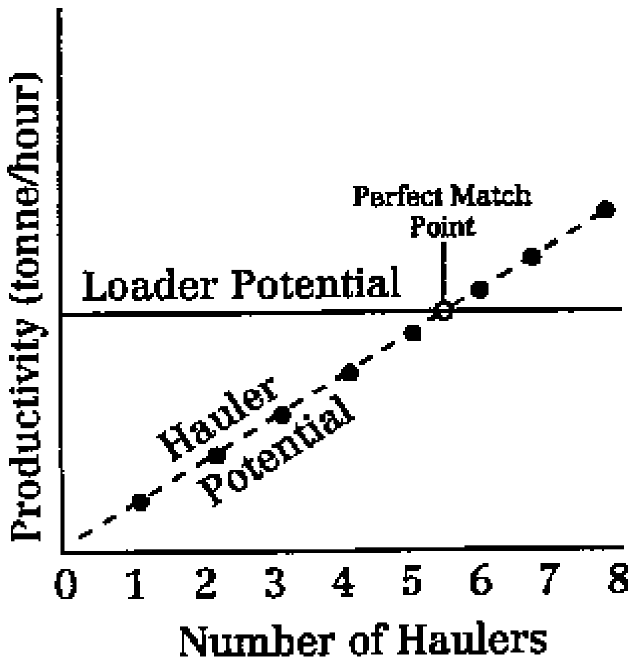


Figure 3.68 "The Perfect Match Point" (Gove, 1994)

The concept is based on deterministic application of mean values for truck loading and truck trip time. The state of "perfect match" is hypothetical because the parameters of truck loading and trip times and other relevant factors such as mechanical availability and utilisation of both loading and hauling equipment are all random variables. Accordingly, practical “perfect match” occurrences will be limited to single coincidental events or, at most short lived phenomena.

Fleet Matching

The term “Fleet Match” can be expressed algebraically as:

$$\text{Fleet match} = \frac{\text{(Loading + Exchange Time) x Number of Trucks}}{\text{Truck Trip Time}}$$

M_F = Fleet match – i.e., number of trucks per loading unit as above.

$$M_F = (T_L + T_S) \bullet N / T_T \qquad (14)$$

$M_F = 1$ is the state of “perfect match”.

$M_F > 1$ there are excess trucks, i.e., overtrucked – so loading equipment is fully utilised with trucks decreasingly utilised as M_F increases.

$M_F < 1$ there are insufficient trucks that are fully utilised but loading equipment is increasingly under utilised as MF decreases.

The M_F - productivity regimes are illustrated by Figure 3.69.

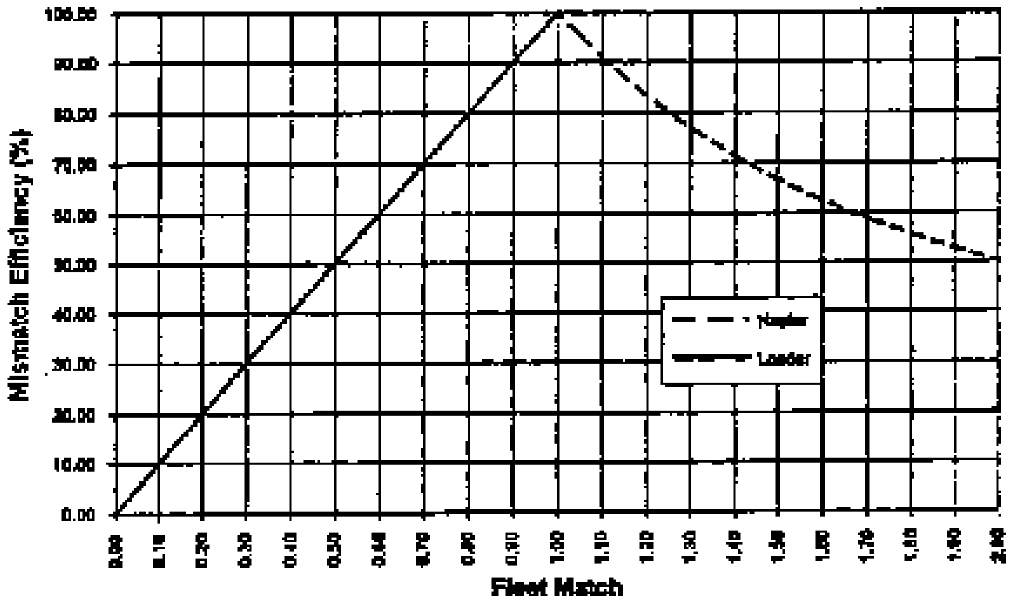


Figure 3.69 Mismatch Efficiency v. Fleet Match (Gove, 1994)

In application, equation (14) will rarely yield an integer result. Generally, the number of trucks – M_F - includes a part of a truck. Practically M_F must be rounded up or down to the nearest integer number – as demonstrated previously by equation (7).

Mismatch Treatment Options

Opinions amongst equipment-selection practitioners are mixed on the appropriate protocol for rounding of truck fleet match. The author has observed a traditional tendency of estimators to always round up. This is a conservative practice from a production perspective, but does not necessarily realise best unit costs.

Fourie and Dohm provide a "rule of thumb" that parts of trucks in the range:– $> 0 \leq 0.29$ should be rounded down and parts of trucks in the range:- $\geq 0.30 < 1.00$ should be rounded up. (Fourie, 1992). This arbitrary treatment may be adequate for preliminary estimates. But, as shown in 5.5.3, rounding to nearest integer number of trucks is a joint productivity-cost issue. Generally rounding down sacrifices loading equipment productivity for lowest unit cost of production. This is discussed in terms of cost effects in some detail in Sections 5.5.3 and 5.5.4.

It is the author's experience that, where production targets are firm, not to be exceeded, best load and haul operating rhythm, and optimum operating costs, are achieved by opting to round down to a modest undertrucking state. Where production must be maximized; and the potential cost penalty is justified by downstream processing cost benefits and marketing opportunities, then overtrucking will likely be the adopted option. From time to time, there will be changing economic and operating circumstances where either undertrucking or overtrucking loading equipment is the favoured operating state.

3.5.4 Bunching - Contributing Factors and Treatment

Nature of Bunching

The fifth of the a-productive factors identified in Section 3.5.2 is "bunching". In effect bunching is a result of general detuning of load and haul fleet performance – equivalent to an increase in truck trip time T_T. Alternatively, greater-than-expected waiting time W_L can be experienced by loading equipment if sufficiently undertrucked to ameliorate bunching. Particularly higher than expected bunching a-productivity can result from a closer-than-expected approach to exact fleet match between loading equipment and trucks.

Some authors have treated "fleet matching" and "bunching" concurrently or in close succession with the inference of interrelationship or firm connection. As discussed below, there is empirical evidence consistent with intuitive reasoning that the degree of bunching effect increases to a maximum at the "perfect match point" This implies that bunching can be treated by fleet mismatching. In fact, the two issues are quite separate in cause and effect. Fleet matching can be expressed mathematically in terms of mean values of operating parameters and treated deterministically using equation (14) or other related linear relationships that include cost considerations – as shown in Sections 5.5.3 and 5.5.4. A mismatch can be simply treated by adding or subtracting trucks. Productivity reduction due to bunching cannot be remedied by adding trucks. Productivity necessary to realise planned production must provide sufficiently for any bunching a-productivity that the load-and-haul fleet configuration attracts. So, it is necessary to establish a policy of fleet matching configuration to anticipate bunching effects likely to be experienced when load and haul equipment has been identified and fleet numbers established probabilistically as shown in Section 3.5.2.

In the following discussion, "Bunching" is described as an outcome of variability in operating parameters. Distribution modelling of the bunching phenomenon, or developing other mathematical process as a basis for analysis, is difficult; and difficulty increases with complexity of the operation.

Bunching manifests as a productivity inefficiency resulting from a-rhythmic loading and hauling operations that result in increased load and haul costs.

If operating sufficiently overtrucked, "bunching" results only in a general detune of truck fleet performance – similar to an increase in T_T. Trucking costs tend to increase moderately without change in productivity. If overtrucking is limited with number of trucks close to the "perfect match point", loading equipment will tend to experience increased waiting time due to bunching, so loss of productivity.

If undertrucked, loading equipment will be under-utilised manifesting as waiting time that accommodates part or all of the truck trip time extension due to bunching causes and effects. In the undertrucked state, loading equipment will experience a-productivity due to mismatch but will also experience reduced bunching a-productivity than would be experienced at the "match point". An extreme case can be hypothesized where undertrucking effect offsets bunching effect. In the significantly overtrucked state, loading equipment will not experience a-productivity due to mismatch. But increasing overtrucking will increase expected waiting time for trucks

– W_T – with significantly reduced truck efficiency. As overtrucking reduces, bunching a-productivity tends to increase as the truck fleet closes on the match point.

As described in Section 3.3.10, substantial empirical evidence collected by Caterpillar and described by Dan Gove and Bill Morgan, (Gove, 1994), has shown that:

- CV_T of truck trip times is generally in the range 0.10 to 0.20
- Bunching a-productivity is consistent with a discount factor at $M_F = 1$ of:

  Bunching Efficiency $\quad\quad \eta_B = (1 - CV_T) \quad\quad (15)$

- Effect of bunching reduces as $M_F >> 1$ or $<< 1$

A statistical-mathematical rationale for equation (15) is offered in Section 5.5.6.

Figure 3.70 illustrates the relationship between fleet match M_F and bunching efficiency η_B for three levels of effect used in Caterpillar's FPC simulation software as follows:

1. Minimum 90% corresponding to CV_T of 0.10 for T_T
2. Average 85% corresponding to CV_T of 0.15 for T_T
3. Maximum 80% corresponding to CV_T of 0.20 for T_T

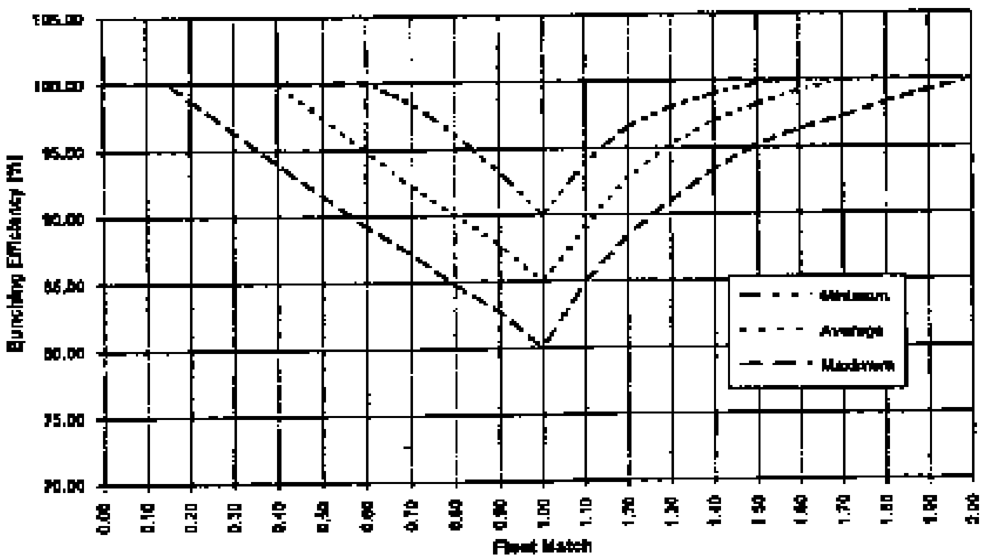


Figure 3.70 Bunching Efficiency v. Fleet Match (Gove, 1994)

Figure 3.71 illustrating Loader Efficiency η_L v. Fleet Match M_F and Figure 3.72, below, illustrating Truck Efficiency η_T v. Fleet Match M_F - indicate how M_F needs to be adjusted for any required - η_L - or - η_T – so indicating necessary adjustment to the truck fleet or loading equipment fleet for any required productivity.

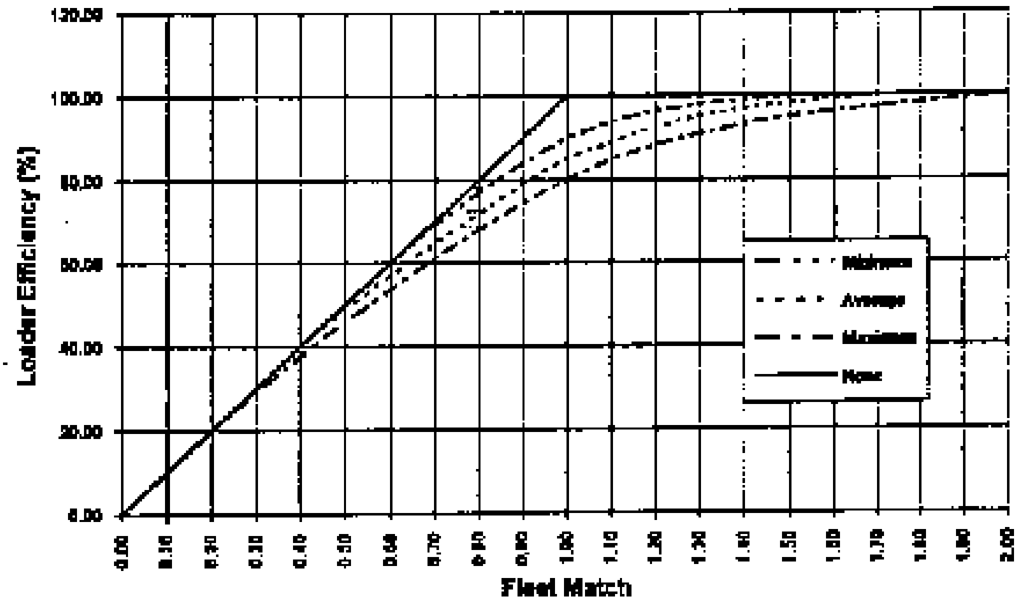


Figure 3.71 Loader Mismatch & Bunching v. Fleet Match (Gove, 1994)

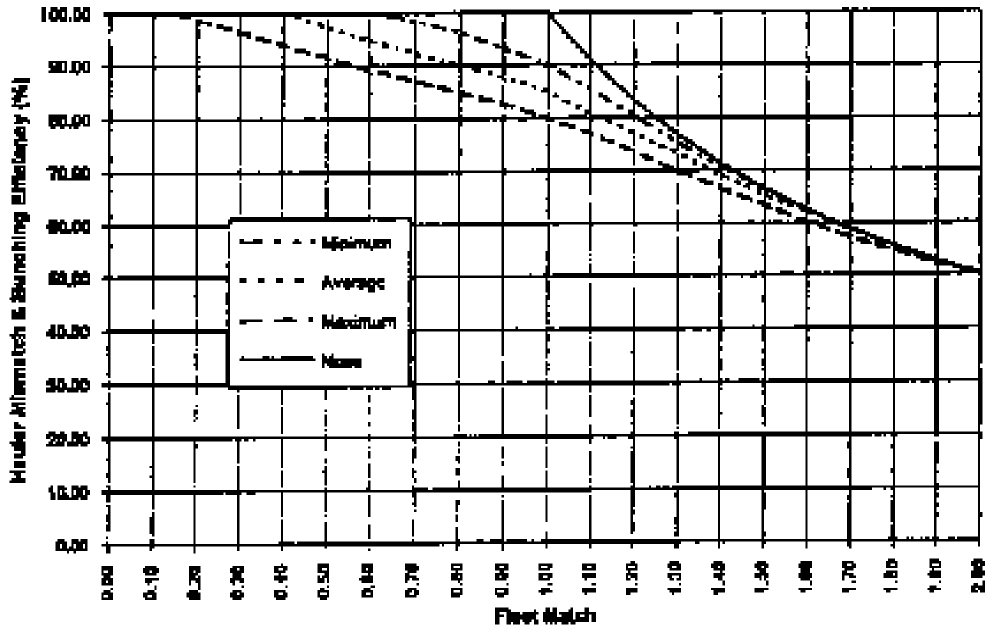


Figure 3.72 Hauler Mismatch & Bunching v. Fleet Match (Gove, 1994)

Productivity Treatment for Bunching

Two options identified for treating a-productive effect of bunching are:

1. Before proceeding to preliminary fleet selection and determination of number of trucks, increase target productivity by a contingency that will offset a-

productivity of bunching at the level expected to result from matching policy and the selection process method. Between 10% and 20% contingency at perfect match point provides a guide - correct as necessary by iterative analysis.

2. Review the general nature of the mine operation and planned pit configuration, and decide to overtruck or undertruck ("overshoveling") and degree to achieve an acceptable forecast for bunching a-productivity. Overtrucking is favoured by short hauls, small numbers of trucks in fleets and low number of passes [nominal 3 or 4 pass] to load trucks. Undertrucking (over–shoveling) is favoured by long hauls; large numbers of trucks in fleets and increased passes [nominal 5 or 6 pass] to load trucks. Set up the fleet match point either overtrucked or undertrucked so bunching effect is reduced. Proceed to fleet numbers and matching analysis. Rework in terms of costs – see Section 5.5.5 - as necessary to adjust for significant departure from the planned fleet match and expected bunching forecast.

Bunching Causes and Prevention Measures

Factors that contribute to bunching and resulting a-productivity are listed and discussed below. Some causes of bunching are controllable to some degree by planning and management initiatives to reduce bunching effects. Other causes such as natural variability in equipment performance can be considered as non-controllable. Obviously management of load and haul operations must aim for prevention where causes are controllable. Generally, preventative measures will improve operational "rhythm" – a concept raised in earlier discussions.

Causes of bunching include:

Intrinsics – generally non-controllable causes:

- Engine power variability.
- Variability of efficiency of power transmission of other driveline componentry.
- Tyre wear – so varying wheel diameter, increasing rim pull and tending to reduce rim speed of trucks.
- NMW variability that results in the necessity to determine individual payload for each truck to comply with specified maximum GMW – a prerequisite for steering and braking safety standards compliance.

- Debris adhering to truck chassis and structure.
- Carry-back in the truck body.
- Variable operator skill and efficiency – a result of varying natural skills, training levels together with effects of changing operating conditions including weather and staff turnover.

Non-intrinsics – generally controllable to some degree:

- Variability in surfaces of roads and manoeuvering areas.
- Changing competency of materials exposed by mining as a foundation for mine haul roads.
- Availability of road surfacing materials in required quantity and of acceptable quality.
- Other mining activities that interrupt or interfere with loading and hauling operations.
- Errors and omissions by supervision, management, oversights in pit development strategy, mine planning and scheduling.
- Non-optimum maintenance reliability and efficiency (bench marks for which are provided in Section 3.4 – specifically Table 3.67).

The above list is representative rather than exhaustive. Degree of effect on bunching by listed causes, and consequent a-productivity is wide ranging. The end effect observed empirically is an accumulation that does not always allow identification of specific causes to enable application of management remedies. The above items are discussed in more detail below.

Discussion On Some Variability Factors

Caterpillar claims that new and dealer-repaired or remanufactured engines will dynamometer-test to +/-3% of designed flywheel power. Data provided by WesTrac, Caterpillar's dealership in Western Australia, has been analysed. Descriptive statistics are provided in Table 3.74. CV of this small sample of off-site rebuilds only is 1.35% inferring a range of +/-3.7% - consistent with relatively low dispersion. The statistics are expressed both in terms of dynamometer tests in kW; also in indices based on specified engine power of 1300 kW. New engines ex factory and remanufactured engines can be expected to exhibit reduced variability – within the claimed range indicated

.

Table 3.74 Truck Variable Performance Factors

	Engine Power kW		Truck NMW	
	kW	Indices	Mean	Minima
Number of Records	32	32	24	24
Maximum Value	2388	1.038	166.13	164.30
Minimum Value	2221	0.966	157.65	155.70
Range	167	0.073	8.47	8.60
Range %	7.25	7.248	5.23	5.40
Average of Range	2304.50	1.002	161.89	160.00
Average of Population (Arithmetic Mean)	2304.13	1.002	161.91	159.31
Median	2303.50	1.002	161.86	159.60
Variance	963.73	0.000	4.83	4.54
Std Dev Sample	31.04	0.0135	2.20	2.13
Coefficient of Variation	0.0135	0.0135	0.0136	0.0134
Skewness	0.03	0.03	0.14	0.10
	To Right	To Right	To Right	To Right
Kurtosis	1.82	1.82	-0.51	-0.15
	Leptokurtic	Leptokurtic	Platykurtic	Platykurtic

NMW variability is illustrated by descriptive statistics of a small sample of data (provided by WesTrac) in Table 3.74. NMW of nominal 220-tonne payload trucks were weighed at three separate times. Descriptive statistics of means of NMW determinations for each truck in the fleet are shown in Table 3.74. The relatively low but significant variability of the small sample of NMW shown by CV of 1.4% and a range of 5.2% in Table 3.74 is some half the variability indicated by advice from Hays that NMW can vary +/-5%, (Hays, 1990, p675). Trucks operating in deep pits spend more than some 70% (as much as 75% - Table 3.63) of travel time fully loaded. So performance for that segment of a truck trip is subject to GMW. As NMW is in the order of 40% to 50% of GMW, effect of variability on measured NMW relative to GMW is proportionally reduced. The effect of variability of truck payload measured relative to GMW is similarly proportionally reduced. Joint distribution theory also indicates that expected values are simply cumulative but variance is an exponential function as follows:

$$E_{GMW} = E_{NMW} + E_{PL} \qquad (16)$$

E_{GMW} = Expected (mean) value of GMW.

E_{NMW} = Expected (mean) value of NMW.

E_{PL} = Expected (mean) value of PL.

$$\sigma_{GMW}^2 = \sigma_{NMW}^2 + \sigma_{PL}^2 \qquad (17)$$

$$\sigma_{GMW} = \sqrt{(\sigma_{NMW}^2 + \sigma_{PL}^2)} \qquad (17A)$$

σ_{GMW} = standard deviation of GMW

σ_{NMW} = standard deviation of NMW

σ_{PL} = standard deviation of PL

Equations (16), (17) and (17A) are valid for random variables. Equations (17) and (17A) are valid only for independent variables. (Chou, 1969, p178). NMW and PL are generally treated as independent random variables in the research.

Measuring relative variability in terms of CV, equation 17A shows that the relative variability - CV, of GMW will generally be significantly lower than the variability, i.e., CV's of either NMW or PL:

$\sigma_{GMW} > \sigma_{NMW}$; also $\sigma_{GMW} > \sigma_{PL}$

CV is a quotient of σ/mean of variable, always < 1. When comparing CV statistics with values < 1:

$CV_{GMW} < CV_{NMW}$; also $CV_{GMW} < CV_{PL}$

The cited relationships have application, utilized in Section 5.5.6, for the similar relationship that generally exists between truck trip times as an accumulation of components including loading time, dumping time, turn and spot, queuing time; and, importantly, waiting time.

The interpretation of the above discussion is that NMW variability is measurable and significant when standing alone. But the productivity effect of NMW variability is relatively reduced when combined with payload variability. But the effect does potentially influence truck payload capacity and measured payload variability in actual operations, albeit to relatively small degree. Efforts to reduce NMW variability are beneficial and considered to be generally justified.

Tyre-wear induces truck-performance variability due to decreased wheel radius and consequent increased rimpull. There is a complementary small reduction in top speed that can be assessed from tyre dimensions.

Standard tread depth of tyres for larger mining trucks is designated E4 – an indication of the wear potential of tyres. The ratio of E4 tread depth to tyre diameter is in the order of 6% for large mining truck tyres, so maximum possible diameter reduction from wear is limited to some 6%. Only a small proportion of tyres on large mining trucks wear out - tread worn away to a bald, slick tyre surface. Premature failure due to separation, sidewall cutting or rock penetration preempts tread wearing out for in the order of 90% of tyres at sites where operational conditions are severe (author's experience).

For the purposes of discussion, it can be assumed that average tread wear to failure is 50%. This corresponds to an average wheel radius decrease, and rimpull increase, of some 3%. So, up-ramp speed for deep pits will tend to decrease slightly. But this is dependent on the rimpull/speed curve for the particular truck and how it negotiates the ramp profile. All other segments of a truck haul that are not at full power will not be affected by tread wear. Average T_T time variations due to tyre wear in deep pits can be expected to range +/- 1% to 1.5% from mean tyre diameters. Accordingly a-productive effect due to tyre wear can be considered relatively small.

In Section 3.3.4 and, particularly, Figure 3.29, it was indicated that power transmission loss from engine flywheel to wheels is some 17% for mechanical drive trucks to 26% for DC electric wheel trucks with losses for AC electric-wheel trucks in the order of 20% approaching mechanical drives. These are nominal average power-loss percentages. The shape of the torque – speed curve for alternative drive systems is also a difference significant in the operational application of mining trucks. The practically flat torque curve of mechanical drive trucks has advantages in deep pit operations.

There is also significant variability in the transmission systems of mining trucks. The author's best estimate based on experience with dynamometer tests of mechanical transmissions and efficiency tests of electric motors, generators and alternators would be a variability range similar to engine variability, i.e., +/- 3% to 5% over component life.

It can be concluded from the discussion (albeit limited to a small number of factors) that the several factors contributing to bunching effects tend to be individually small. But collectively the factors accumulate to a significant a-productive bunching effect.

Consequently, controllable factors, the non-intrinsics and those intrinsics that can be managed to advantage, need to be identified and the necessary controls established and implemented. Prevention of bunching effects is an unrealistic expectancy. But minimization of bunching is a realistic ambition by addressing influencing factors and working towards reasonable operational “rhythm”.

In the process of equipment selection, when setting up operating criteria including productivity required to realise the mine production plan, some concept of bunching effect to be accommodated must be developed. The a-productive effect of bunching can be assessed; but this is dependent on the degree of control to be exercised over the causes of bunching effects that must be specified, justified as realistic and effectively established.

Bunching Comments

A-productive effects from bunching manifest as inefficiencies in load and haul performance. The nature and degree of reduced productivity is a complex outcome dependent on variability in performance of the equipment items that make up a load and haul fleet. There is empirical evidence of the degree of bunching effect that can be applied to discount productivity to provide realistic production results. Literature research for mathematical analytical solutions was unsuccessful. A typical comment in references (paraphrased): “It is difficult to obtain wait times for loading equipment and trucks as a function of the number of loading items and trucks” - (Ramani, 1990).

It is also difficult to model bunching effects stochastically. These difficulties have been addressed by using empirical data as the basis for simulation of truck waiting time and bunching effects as an alternative solution.

The criteria (illustrated by Figure 3.70) for bunching inefficiency, over a range of fleet match values, are based on Caterpillar’s research as presented by Gove and Morgan. These criteria are in general industry use for estimating bunching effects. If the option to adopt a specific selection of these criteria is exercised, they are embedded in results from FPC simulation applications that are generally accepted by the mining industry. So, it is concluded that bunching criteria, based on empirical observations and described above, are adequate for adoption by estimators and engineers for purposes of establishing production and productivity in the process of load and haul equipment selection for open pit mining.

Notwithstanding the perceived practical adequacy of the empirical-based approach discussed, Section 5.5.6 offers a rationale in statistical-mathematical terms for equation (15) as a basis for discussing cost effects of bunching.

3.6 TIME MANAGEMENT – DEFINITIONS

3.6.1 Introduction

Before commencing selection of equipment of suitable capacity, and in adequate numbers, to satisfy a mining production objective, all of the necessary inputs need to be assembled. Realistic production estimates and other operating parameters, and an understanding and setting up of a sound basis drawn from prior effective mine planning are essential prerequisites for effective equipment selection. The process and various inputs are more fully discussed in Section 4.1.1 to follow.

Setting up a defined Time Management System is an imperative management initiative to enable determination of time components relevant to productivity determinations. An essential input to establishing required equipment performance is a realistic estimation (particularly for a "greenfields" project) or determination from historical data, in-house or industry-wide, of the number of productive hours per year, that can be adopted as a robust basis for required productivity determinations.

3.6.2 Definitions

For the purposes of discussion a simplified time management system and inferred time definitions is provided by Figure 3.73. The following definitions are illustrated by Figure 3.73:

Potential Working Hours = Calendar Time = 24 hours everyday of the year – taken as 8,760 hours.

Mechanical Availability (MA): $= \dfrac{\textbf{\textit{Available Time}}}{\textbf{\textit{Calendar Time}}}$

$= \dfrac{\textbf{\textit{Calendar time – Downtime}}}{\textbf{Calendar time}}$

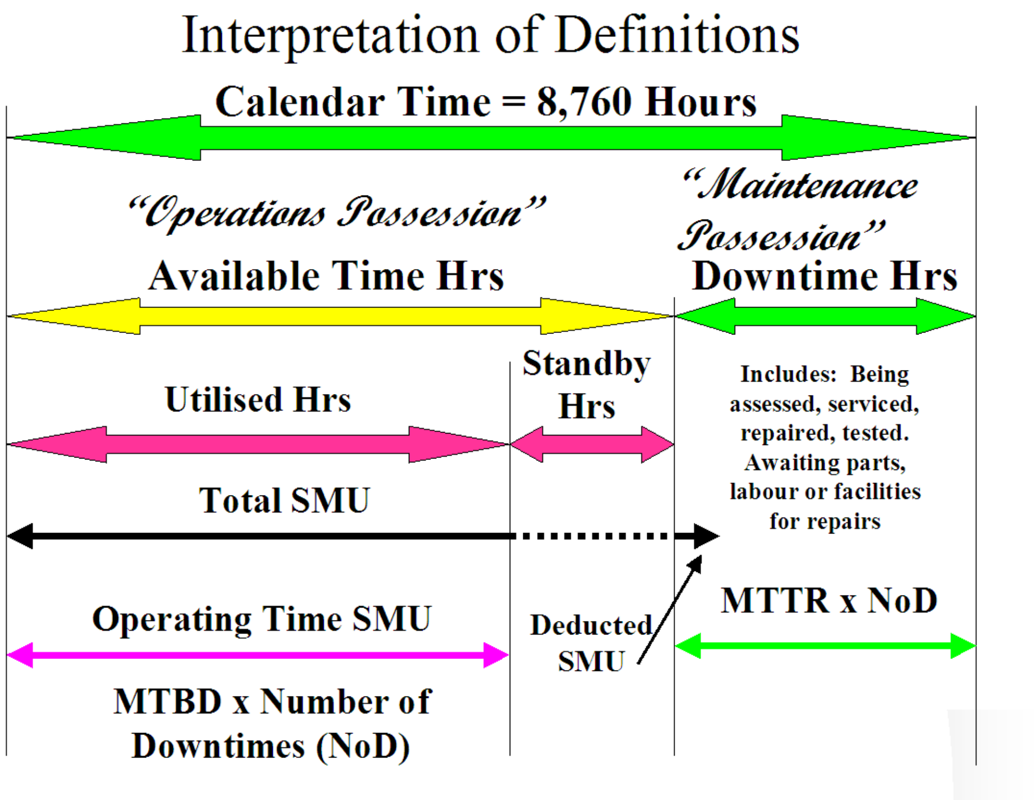


Figure 3.73 Interpretation of Definitions

It will be noted that the definition of MA is similar to the operational version (rather than maintenance version) in Section 3.4 – provided that:

- The "time available for work" in the MA definition in Section 3.4 is "calendar time", i.e., 8,760 hours (ignoring leap years of 8,784 hours); and
- The exclusions of daily fuel and lube top-ups from "downtime" are observed as described in Section 3.4 when calculating MTBS.

That is:

Downtime: from an operations perspective includes all planned and unplanned maintenance including refueling, lube top-ups, tests during normal operations and changing of tyres. Depending on the purposes for which the MA and related MTBS and MTTR are used, local policy may exclude some or all of these strictly non-operational items. Accordingly, policy must be established for inclusion or exclusion of refueling, lube top-ups and testing. Obviously, this potential downtime ambiguity needs to be addressed by due diligence investigation of other mine sites in the process of equipment selection. For the avoidance of doubt, the author places the boundary of downtime according to whichever function has “possession” of the equipment, maintenance or operations – as illustrated by Figure 3.73.

Available: simply – if equipment is not down it is operational.

Utilised Time: (Use of Availability) alternatively termed *Operating Time:* = Proportion of time actually used in operations

Standby Time: non-utilised available time.

Reliability – MTBD }These three items, referred to in Figure 3.73

Maintainability – MTTR }previous page, were described and discussed

Number of Downtimes (NoD) }in Section 3.4.

Service Meter Units – SMU The record in SMU of equipment operating time as the time the engine is running (for equipment powered by internal combustion engine) or by power metering of electric powered equipment.

SMU–determined operating time as recorded by on-board information management systems (such as Caterpillar’s VIMS or equivalent) is always greater than operating time measured by equipment activity from dispatch systems or operator time keeping records. Specifically for trucks, that the engine is running is not necessarily evidence that the truck is engaged in productive operations. The author is aware of a deep open pit operation using 225 tonne trucks where, for an annual average, effective operating hours were 92% of the SMU recorded for the same annual period.

The provided description of Time Definitions should be considered as introductory only.

3.6.3 Dispatch Systems

As the demand increases for more management control to realise productivity and cost expectancies for equipment of increasing scale the need for accurate, accessible data of ever-increasing detail becomes more cogent.

Table 3.75, Volume 2, Appendices, provides time definitions from a typical dispatch facility. The detail of time allocation is substantially increased from that used to compile Figure 3.73. Increased discrimination by defining more detailed time components provides accurate time bases for a large number of key performance indicators KPI to cover production and maintenance activities. Given appropriate additions to dispatch facility software, particularly the front-end reporting functions, even more detailed subdivision of time can be recorded. As time proportions of individual time components reduce in size, care must be taken to ensure that time data is sufficiently accurate to be meaningful. For detailed management studies, such as continuous improvement programmes (CIP), it may be necessary to undertake specific time / activity studies to audit dispatch system reports or even to supplement data available from a dispatch facility.

Amanda Croser describes continuous improvement (CI) management at a deep open pit gold mine. Priorities for CI application to service activities were identified on the basis of KPI. Croser indicates a detailed breakdown of mean truck fleet times. Daily service for fuelling and lube top up is included transparently in a group of "maintenance related" activities.

Table 3.76 Time Definitions & Breakdowns – Empirical Data

(Croser, 2004)

SOURCE									
WENCO Dispatch System					**Specific Job Studies**			**Specific Job Studies**	
			%			**Hours**			**Hours**
Total time		8,760	100.00		**Truck Average Service**	19.6		**Truck Activity Time**	11.7
Breakdown	Maintenance Related Delays	432	4.93		**Activity**	11.7		**Washing**	0.5
Scheduled Service		336	3.84		**Wait**	6.8		**Pre-inspect & Prep**	0.4
Scheduled Maintenance		38	0.43		**Lost**	1.1		**Servicing**	8.8
Daily Service		110	1.26					**Defects**	2.0
Tyre Change / Repair		64	0.73						
Accident Damage		65	0.74						
Other Downtime		37	0.42						
Crib	Lost & Delay Times	583	6.66						
Standby		394	4.50						
Operator Changeover		38	0.43						
Other delays		49	0.56						
Queue and Wait	**Operational Delays**	711	8.12						
Ready Time		5,903	67.38						

Table 3.76 illustrates Croser's technique of "drilling down" through time analysis levels to isolate specific detailed time components to assess potential for production / cost improvement and so assignment of priorities (Croser, 2004).

Clearly the degree of detail described by Amanda Croser is not necessary for establishing the basis of load and haul equipment selection. It is considered sufficient to estimate or establish from existing empirical data realistic average time assignments in annual hours for:

- Rostered (scheduled) time.
- Available time.
- Utilised time.
- Direct operating time.

This last item is useful if instantaneous digging rates are a productivity estimating parameter. If equipment productivity rates are recorded or estimated over a period of a shift or longer period then for production estimating it may be more appropriate to use utilised time as the basis for production estimating.

Obviously, care needs to be taken to understand current definitions of time components and reporting / estimating practice during due diligence investigation both internally and at any external mining operation.

Western Australian School of Mines

Selection Criteria For Loading and Hauling Equipment - Open Pit Mining Applications

Volume 2

Raymond J Hardy

This Thesis is presented for the Degree
Of
Doctor of Philosophy
Of
Curtin University of Technology

July 2007

DECLARATION

This thesis does not contain material previously accepted for the award of any other degree or diploma in any university. To the best of my knowledge and belief, where this thesis contains material previously published by any other person, due acknowledgement has been made.

………………………..

R J Hardy
9 July 2007

THESIS PRESENTATION

Scope of the research, breadth of topics and resulting extensive documentation required presentation of the thesis in two volumes.

Volume 1 **Selection Criteria Issues, Productivity and Analysis –** Chapters 1 to 3 included in Volume 1; and

Volume 2 **Selection Process, Costs, Conclusions and Recommendations –** Chapters 4 to 7 - plus References, Appendices and Supplementary Information – included in Volume 2.

ABSTRACT

Methods for estimating productivity and costs, and dependent equipment selection process, have needed to be increasingly reliable. Estimated productivity and costs must be as accurate as possible in reflecting actual productivity and costs experienced by mining operations to accommodate the long-term trend for diminishing commodity prices,

For loading and hauling equipment operating in open pit mines, some of the interrelated estimating criteria have been investigated for better understanding; and, consequently, more reliable estimates of production and costs, also more effective equipment selection process.

Analysis recognizes many of the interrelated criteria as random variables that can most effectively be reviewed, analyzed and compared in terms of statistical mathematical parameters.

Emphasized throughout is the need for management of the cyclical loading and hauling system using conventional shovels/excavators/loaders and mining trucks to sustain an acceptable “rhythm” for best practice productivity and most-competitive unit-production costs.

Outcomes of the research include an understanding that variability of attributes needs to be contained within acceptable limits. Attributes investigated include truck payloads, bucket loads, loader cycle time, truck loading time and truck cycle time.

Selection of “ultra-class” mining trucks ($\geq$ 290 -tonne payload) and suitable loading equipment is for specialist mining applications only. Where local operating environment and cost factors favourably supplement diminishing cost-benefits of truck scale, ultra-class trucks may be justified. Bigger is not always better – only where bigger can be shown to be better by reasons in addition to the modest cost benefits of ultra-class equipment.

Truck over-loading may, to a moderate degree, increase productivity, but only at increased unit cost. From a unit-cost perspective it is better to under-load than over-load mining trucks.

Where unit production cost is more important than absolute productivity, under-trucking is favoured compared with over-trucking loading equipment.

Bunching of mining trucks manifests as a queuing effect – a loss of effective truck hours. To offset the queuing effect, required productivity needs to be adjusted to anticipate "bunching inefficiency". The "basic number of trucks" delivered by deterministic estimating must provide for bunching inefficiency before application of simulation applications or stochastic analysis is used to determine the necessary number of trucks in the fleet.

In difficult digging conditions it is more important to retain truck operating rhythm than to focus on achieving target payload by indiscriminately adding loader passes. Where trucks are waiting to load, operational tempo should be restored by sacrificing one or more passes. Trucks should preferably be loaded by not more than the nominal (modal) number plus one pass.

The research has:

- Identified and investigated attributes that affect the dispersion of truck payloads, bucket loads, bucket-cycle time, loading time and truck-cycle time.
- The outcomes of the research indicate a need to correlate drilling and blasting quality control and truck payload dispersion. Further research can be expected to determine the interrelationship between accuracy of drilling and blasting attributes including accuracy of hole location and direction.
- Preliminary investigations indicate a relationship between drill-and-blast attributes through blasting quality control to bucket design, dimensions and shape; also discharge characteristics that affect bucket cycle time that needs further research.

ACKNOWLEDGEMENT

"No man is an Island, entire of itself." from *Devotions* by John Donne

Soon after setting out on the research journey recorded in this thesis the words of John Donne, above, were brought home with undeniable impact. In addition to the knowledge acquired during a working lifetime in the mining industry, civil construction and associated vocations the author still depended on many others to safely and satisfactorily reach journey's end. Generous support and contribution of many friends, associates and casual acquaintances in industry and academia that have assisted along the way are acknowledged with sincere thanks and great appreciation.

Every reasonable effort has been made to correctly acknowledge the owners of copyright material. The author would be pleased to hear from any copyright owner incorrectly, or not, acknowledged.

Support and assistance of the following are especially acknowledged:

Associate Professor Dr Emmanuel Chanda of the WA School of Mines (WASM) Mining Department, for his continued encouragement, guidance and supervision; also at WASM, Dr Peter Lilley (formerly Director of WASM, now with CSIRO, Perth), Mahinda Kuruppu and Clive Workman-Davies assisted from time to time.

Western Premier Coal Limited, Collie, WA – within the Wesfarmers Limited group of companies - provided funding for WASM research projects that partially supported the research work described by the thesis. That support is gratefully acknowledged.

At WesTrac, Caterpillar dealer located in South Guildford Western Australia (franchised some 80 years as the first Cat dealer outside USA, more recently expanding into New South Wales and China), personnel that have assisted with information and operating data are too numerous to individually acknowledge. To WesTrac staff in general my sincere thanks.

Individual acknowledgements must be afforded Jim Walker, CEO, for opening the door to the huge knowledge base acquired by WesTrac and possessed by its South Guildford staff; Fred Haynes for his interest, support and supply of weighing data in the early stages; Tony Power who kept the author in touch with technical advances and developments in the mining industry; Adrian Clement who gave his time generously, provided access to VIMS and Minestar data to support the research, also for demonstrating the exciting benefits and promise of these real-time data access systems and Peter Perich for showing the way to market equipment prices and comments on written down values of mining equipment. More recently Quentin Landre has supplied Caterpillar equipment specification material; also providing information to enable any WASM student to access specifications and performance data vital to their understanding of mobile mining equipment.

Particularly I am indebted to WesTrac for access to FPC simulation software and other mining equipment applications over the years; also for a lifetime supply of Performance Handbooks and related specification material.

To David Rea of Caterpillar, many thanks for the insight afforded the author on VIMS and related matters; also to Caterpillar Global Mining generally for access to technical material made available at recent Miners' Forum's in Western Australia.

Other OEM - dealership contacts also contributed including: from Minepro, P & H dealer, Noel Skilbeck and staff; from Komatsu, Alan Costa; from Liebherr, Henning Kloeverkorn, Merv. Laing and Duncan Hooper; from Terex Mining – O & K, Craig Watson and Andy McElroy; from Hitachi, Eric Green – to all many thanks and best wishes to those contacts who have moved on in their career paths.

Tony Cutler of OTRACO provided comprehensive information on large earthmover tyres, especially for ultra-class trucks; also Hans Weber of Michelin provided technical information on tyre application and management for large mining trucks.

To the many other personal and commercial contacts in and associated with the mining industry that have steadily increased the volume of knowledge and understanding available to the author, knowledge that was tapped to progress the research, the author extends heartfelt thanks and appreciation.

Last but not least the author acknowledges the contribution of his partner in life, Heide, who supported the research with an interesting mixture of tolerance and vigorous encouragement including many exhortations to "get on with it" – when the author was experiencing adequate strength of spirit but an inhibiting weakness of flesh.

Raymond J Hardy, July 2007.

VOLUME 1

TABLE OF CONTENTS

VOLUME 2

TABLE OF CONTENTS

APPENDICES

APPENDED TABLES

TABLES AND FILES ON CD (INSIDE BACK COVER)

SELECTED TABLESS

Listed below are "Selected Tables" that readers may find of practical use or may serve as a foundation for further research. For convenience they are provided on a CD inside the back cover of Volume 2.

Table Number	Table Title	Page Number For Hard Copy
3.4	Loader Efficiency - η_{LE}% - and Number of Truck-Loading Passes	Vol. 1, p 50
3.57	Productivity Criteria for Sacrificing Bucket Loads	Vol. 2, p 459
3.59	Cycle Time Calculator, 4 Parts	Vol. 2, pp 460 - 463
3.70 and 3.71	Probability Tables	Vol. 2, pp 464, 465
5.10	Cost Criteria for Sacrificing Bucket Loads	Vol. 2, p 467

SUPPLEMENTARY INFORMATION

Three papers published during the tenure of the research have been referred to in the text. Copies of files are on the CD inside the back cover of Volume 2.

Hardy#1, Raymond J, **(2003)** *Four-Pass Loading, Must Have Or Myth?* Fifth Large Open Pit Mining Conference, 2003, Kalgoorlie, Western Australia, The Australasian Institute of Mining and Metallurgy.

Hardy#2, Raymond J, **(2003)** *Outsource or Owner Operate,* Twelfth International Symposium on Mine Planning and Equipment Selection, Kalgoorlie, Western Australia, Australasian Institute of Mining and Metallurgy.

Hardy#3, Raymond J, **(2005)** *Outsource versus Owner Operate*, IIR Contract Mining Conference, 2005, Perth, Western Australia. (An update of a previous paper included in references as Hardy#2, 2003).

LIST OF TABLES

LIST OF FIGURES

CHAPTER 4

EQUIPMENT SELECTION

4.1 SELECTION IMPLEMENTATION

4.1.1 Introduction

4.1.1.1 Preamble

Chapter 4 describes review processes and activities based on selection criteria that facilitate equipment selection decisions. Except for additions to existing fleets (where uniformity of fleets is productivity and cost beneficial) mining equipment selection is generally comparative. A favoured option is determined in terms of the criteria examined in Chapter 3.

Research moved on to address preliminaries and final analysis culminating in equipment procurement. Final selection processes generally commence with compilations of comparative equipment specifications that, for each functional type of mining equipment have high probability of achieving required productivity performance and realization of planned production.

4.1.1.2 Specification Fundamentals

A “specification” provides a description of the service or product - the resource inputs and methods applied. Other parts of procurement documentation, including formally tendered contracts, providing information on administration, financial, commercial and legal matters are generally labelled “terms and conditions”.

Writing and compiling a specification requires the most comprehensive input from stakeholders within a procuring organisation. It is the outcome of all prerequisite planning; also production and operational investigation within the overall procurement process.

Specifications can be of several types each with a separate technique for specifying any individual equipment item, including:

1. Functional – usually with attached general performance undertaking or guarantee.

2. Performance – defined performance in quantity, quality and time.
3. Technical – detailed to the degree of non-ambiguity on the basic materials, physical form and quality of each individual item in the group of individual parts that make up an equipment unit.
4. Reference – to standard specification documentation - a secondary stage of Type 3. - with possibly addition of some part of techniques of Type 1. and/or Type 2.

Examples of specification techniques for specifying a mousetrap:

- Functional: - to attract, using cheese bait, and kill mice instantaneously.
- Performance: - to be baited with cheese and safely hand set to kill mice instantaneously with a minimum life of 500 operations.
- Technical: - A *pinus radiata* timber base with dimensions “x - y - z”, powered by a coil spring of “g” gauge steel wire preloaded to “k” kilos and with an operating time of “t” milliseconds etc.

Advantages of “performance” and “functional” specifications, individually or combined:

- Reduced time and effort to prepare.
- Avoids bias for/against one method or type of product/service.
- Encourages more innovative and economical responses.
- Allow bidders to offer alternative, low-cost solutions.
- Provides results-based responses.
- Relieves procurer of responsibility for product design.

Prerequisites for an effective specification:

- Developed from clear objectives that are the result of thorough investigation and due diligence.
- Express needs clearly, concisely and unambiguously.
- Not containing unnecessary details.
- Logically constructed to enable analysis by others.

- Maximize use of standards of construction, performance and acceptance criteria.
- State principle criteria for evaluation of quotations or tenders, supply and operational performance; and final acceptance.
- Special needs of each unique mining project.
- Non-discriminatory against any supplier/contractor.
- Provides opportunity for alternative or innovative quotations that satisfy project needs.

Compiling and writing a specification requires efficient due diligence and will generally be re-drafted many times to satisfy stakeholders.

A refined Specification is achieved by:

- Multi-stage tendering process – this is the normal situation where equipment selection process involves multiple and repetitive dialogue between suppliers and purchasers of mining equipment.
- Painstaking, repetitive editing and re-drafting.
- Realization that contract risk of an inadequate specification is generally high to extreme with unacceptable likelihood and consequences.

The above discussion on specifications was compiled with reference to “Contract Management in Australia” by John James (James, 1995).

4.1.1.3 Equipment Specifications

A reliable specification leaves no “hanging” questions. If for any sentence any of the simple questions Why? When? Where? What? and How? are valid, then the specification must be amended to satisfactorily answer the question(s).

Most mining equipment procurement involves a “Type 4.” specification, as well as some combination of “Types 1 to 3”. Suppliers (OEM and dealers) compile standard specification documentation, inclusive of a small list of separately priced optional items. Each optional item offered can be accepted or rejected by the purchaser. Optional on-board equipment operational and management systems are generally separately specified and priced. Suppliers will generally accommodate specific needs of purchasers for equipment specification and terms and conditions of tendering.

Any amended or additional specifications required by the purchaser can be negotiated with the dealer/OEM. The responsibility for any consequent modifications depends on the nature of each modification or additional equipment. For example supplying and location of portable fire extinguishers would certainly be a dealer responsibility. But supply and installation detail of fire suppression equipment in an engine compartment, can be expected to benefit from, and require OEM involvement to preserve warranty and any performance guarantees.

On arriving at a final specification satisfactory to the purchaser and acceptable to the dealer and OEM:

- Final pricing can be negotiated or tendered to the purchaser.
- Terms of payment can be determined.
- Delivery arrangements can be finalized.

Unqualified acceptance of quotation or tender by the buyer sets the terms of contract as negotiated and agreed. It is the mutual responsibility of the supplier (OEM/dealer) and purchaser to meet their individual obligations to satisfactorily conclude the supply contract.

4.1.1.4 Performance Guarantees

Provided the agreed specification does not preclude or conflict with a specified performance guarantee it can be expected to be enforceable and provide the protection, and risk amelioration the purchaser expects. It is the author's experience that if "performance guarantees are too good to be true" they usually are. Enforceability of unrealistic performance guarantees goes hand-in-hand with protracted dispute; and management distraction results from the dispute resolution process.

Purchasers should be duly diligent and make their own reasonable assessment of performance of equipment. During the process of equipment selection this assessment needs to be audited against internal experience, performance at other sites; and with circumspect consideration of performance information provided during the course of marketing activities of dealers and OEM.

It will be realised that OEM design and build mining equipment to suit a broad, but necessarily limited, range of ambient conditions. Optional configurations are often available to extend compatibility of equipment to extreme limits of ambient conditions. But departure from anticipated ambient operating conditions should be treated as an exception for application of performance guarantees when caused by:

- Latent conditions that could not be reasonably foreseen by the purchaser or supplier.
- Omissions, misinterpretation of information or insufficient prior collection of operating-conditions data.

It is the author's opinion that performance guarantees:

- Are no substitute for meticulous due diligence by the potential purchaser in the process of mining equipment selection and procurement.
- Serve the purpose of indicating good faith of the supplier in specification of, and expected performance from, individual equipment items – but always if actual ambient, physical operating conditions, operational functions and demands and management controls are within the ambit anticipated in establishing guaranteed performance.

4.1.1.5 Load and Haul Equipment Options

There is an overwhelming range of options on offer when selecting mining equipment. For loading and hauling equipment there are some six suppliers providing loading equipment options and some five suppliers of large mining trucks.

In discussing the need for, and justifying the application of simulation methods for estimating truck requirements for large open pits, Garston Blackwell (Blackwell, 1999) raises the need for a "realistic approach". Blackwell, in supporting the need for a realistic approach, advises: "Unfortunately there are many factors governing the tonnage hauled by open pit truck fleets that cannot be treated simply, and doing so will usually result in an operation that cannot meet production objectives." This implies support for, application of, probabilistic techniques for estimating "realistic" truck numbers as described in Section 3.5.

Blackwell further advises: "Assumptions made in calculating vehicle performance are optimistic." - for the following reasons:

- Empty trucks are always heavier than when manufactured because of mud and debris stuck to the truck body.

- Extra steel is added to bodies to reinforce (rapid) wearing surfaces and damage points.
- Engines, and to a lesser extent, transmissions do not always operate at rated capacity.
- Diesel fuel quality varies (and ambient operating temperature affects fuel burn).
- (Pay) loads carried are variable.
- Road conditions are variable and single lane traffic may be experienced over some in-pit haul segments.

Blackwell describes a case study where "the slowest vehicle (truck) travelling up-ramp loaded (was observed) at half the speed of the fastest truck". As startling as Blackwell's advice sounds, it is consistent with general observations. The "slowest-fastest" spread highlighted is a range of some +/- 33% of the mean speed assuming a symmetrical distribution. Discounting this effect on truck trip time T_T consistent with FPC output in Table 3.63, for medium to long hauls, realizes an indicated range for corresponding change of T_T ranging +/- 20% – 22%. This indicates a CV for T_T in the order of 7%.

As discussed in Section 3.3.10, Dan Gove and Bill Morgan (Gove, 1994) describe empirical evidence of CV for T_T in the range 0.10 to 0.20. This is consistent with the observed CV of truck trip time T_T distributions of 0.19 unfiltered to 0.12 filtered (only for anomalous values, not for non-intrinsic delays) for the case study discussed in Section 3.3.10 and shown in Table 3.60. Data from both Caterpillar and in Table 3.60 are empirical. Data from Table 3.63 is generated by Caterpillar's FPC application and not subject to non-intrinsic time losses and delays. The results are not necessarily comparable. But considering the accumulative effect of non-intrinsic delays on the FPC data would likely significantly reduce the difference. Certainly the research analysis appears to be consistent with Blackwell's observations for loaded trucks ascending ramps (Blackwell, 1999).

Selection of loading equipment also needs a realistic approach when applying productivity criteria. The open pit mining industry appears to have been sensitive to this need. As described in Section 3.2, bench marking of shovel performance indicates a tendency for the industry to respond to the many reducing factors and criteria by acceptance and a tendency to over-shovel resulting in loading equipment benchmarks of some half of intrinsic performance.

The range of options for trucks and loading equipment and comparative intrinsic performance criteria are further discussed in Section 4.1.5.

4.1.2 Key Considerations

4.1.2.1 All Mining Equipment

The following key considerations need to be addressed in commencing to implement mining equipment selection:

Locality and General Operating Conditions

- Location of proposed or existing operation and logistical scenario.
- Political situation and politico-risk profile.
- Meteorological environment for operations.
- Socio-economic and human resources circumstances.
- Environmental conditions.
- Special safety and health conditions.
- General risk profile of existing or future mining operations, especially hauling operations.

Project Characteristics

- Mining plan, production programme and productivity required.
- Blending requirements and any special delivery requirements for materials such as ore at a crushing facility.
- Special features of the mine plan including loaded down-ramp hauls, steep ramps, long hauls, road surfacing details, separate ore and waste fleets (mixed truck and loading equipment fleets) and trolley assist.
- Industrial background including working hours, rosters, statutory or agreed restrictions, existing industrial culture, quality and quantity of labour and special training requirements related thereto – for operations, maintenance

and interrelated mining functions including work practices of contractors on site.

- Corporate policy interrelated with the hauling operation.

4,1,2,2 ***Considerations - Trucks***

- Reliability of each truck option under consideration.
- Reliability of product support by suppliers.
- Available delivery compared with project requirements and any flexibility in those requirements.
- Tyre operating conditions relative to tyres available.
- Productivity issues including truck body type, capacity and (if any) liner-kit details requiring attention.
- Handling wet materials, highly abrasive and physically demanding materials including cherts and other siliceous materials.
- Truck performance expectancy based on simulation using deterministic data and calculation methods such as FPC (Caterpillar) and Talsim (Runge Mining) – comprehensive analysis of enough cases to cover the range of expected performances required.
- Determination of truck numbers including matching trucks to loading equipment.

The above lists are a composite from work by Bruce Gregory (Gregory#1, 2002), (Gregory#2, 2003) and the author's experience in mining development projects.

4.1.2.3 Considerations - Loading Equipment

Project Characteristics

Key considerations to be addressed to implement selection of mining loading equipment include:

- Geology of the stratigraphy hosting the ore and geometry of the ore deposits.
- Selectivity requirements.
- Production and productivity requirements for the various materials to be mined.

- Mining plan, production programme and productivity profile required.
- Handling wet materials, highly abrasive and physically demanding materials including cherts and other siliceous materials.
- Bench heights – including limitations to control dilution/ore loss for flat-dipping ore deposits.
- Corporate policy interrelated with the hauling operation.

Loading Equipment - Considerations

- Reliability of each loading option under consideration.
- Reliability of product support by suppliers.
- Available delivery compared with project requirements and any flexibility in those requirements.
- Productivity issues including bucket design, capacity and details of ground engaging tools requiring attention.
- Loader performance expectancy based on deterministic data and calculation methods – analysis of enough cases to cover the range of expected performances required.
- Determination of loading equipment numbers including matching with trucks.
- Expected digging characteristics of materials to be mined and all relevant parameters such as fragmentation and voids ratio.
- Minimum mining width.
- Loading equipment mobility requirements.
- Favoured loading equipment option or options including shovel or backhoe, electric or diesel-hydraulic and required dumping height to suit favoured trucks.

The above list is a composite from work by Bruce Gregory (Gregory#2, 2003) and the author’s experience.

4.1.3 Selection Process

4.1.3.1 Timing of Equipment Selection

When does mining equipment selection need to be addressed in the total development process? This question might well surface when the first indicatively economic drill intersections cause both internal excitement for, and external interest in, an exploration project.

From the earliest testing of economic feasibility, costs need to be estimated to enable determination of resource delineation and ore definition criteria such as cut-off grade; also to establish design criteria for optimal pit-design software. Initially costs for such activities are more often adapted from existing operations or previous feasibility studies by suitably adjusting for characteristics and conditions of the project; also for cost variation with time. For preliminary feasibility study purposes, costs so adapted are sufficiently accurate. Expected estimation accuracy at this early development stage is +/-30% or more.

Results from such preliminary studies are the basis for a decision to "mothball" the project (and wait for more favourable marketing environment), to abandon the prospect or to sell. Alternatively, preliminary study results may be sufficiently encouraging, providing opportunity to promote the project. Action taken may include increasing the area of interest, step-out and/or in-fill drilling; and more detailed, technical input through a further study iteration expected to produce more reliable feasibility study results.

At this second stage, where cost estimates need to be more refined, initial mining equipment selection, along with all other significantly increased metallurgical, marketing, environmental and engineering inputs needs to be implemented.

Figure 4.1 indicates the project stages where mining equipment selection is an activity in ever-increasing degrees of refinement through to awarding supply contracts or placing orders.

Technical facilitation of mining equipment selection is a linear process through deterministic and probabilistic calculation stages as implied in Section 3.5 above.

Typical simplified equipment selection process outlines are provided by Figure 4.2, for trucks and Figure 4.3, for loading equipment. These process outlines were prepared for an actual shovel and truck selection project. It will be noted that the simplified flow charts served as report forms and include cost estimations and comparisons – a management requirement when the form was designed.

In Section 3.4.2, product support by equipment suppliers was identified as the most important issue in mining equipment selection. Comparative cost differences tend to be subtle and consequently are often not clearly conclusive. So cost comparisons tend to be more confirmatory than a robust aid for decision. Obviously, costs are needed for budget purposes and for compiling of total mining costs. To reduce repetitive work, and for economy of technical resource utilization, cost estimations should be left as late as possible in the equipment selection process. This does not mean that all options are carried forward. The selection process through application of selection criteria and due diligence investigation of all interrelated issues should eliminate many of the options as unsuitable for the planned production programme and operational characteristics.

The initial five activities in both Figure 4.2, and Figure 4.3, for either truck or loading unit selection, can be generally described as "due diligence". This investigative stage was referred to, and its importance emphasized, initially in relation to "maintainability" in Section 3.4.2 and subsequently through Section 3.4 and in Section 4.1.2.

4.1.4 Selection Strategy

4.1.4.1 Effects of Non-optimum Selection – Trucks

Bruce Gregory compared the cost-impact due to non-optimum truck selection for a number of issues relative to a small degree of performance effect. His work was based on 2002 prices (Gregory#1, 2002). Table 4.1 updates Gregory's comparison to reflect recent increases in the price of fuel and tyres. Degrees of performance effect were amended and an indicated impact of poor product support by suppliers was quantified. The results in Table 4.1 are illustrated by Figure 4.4.

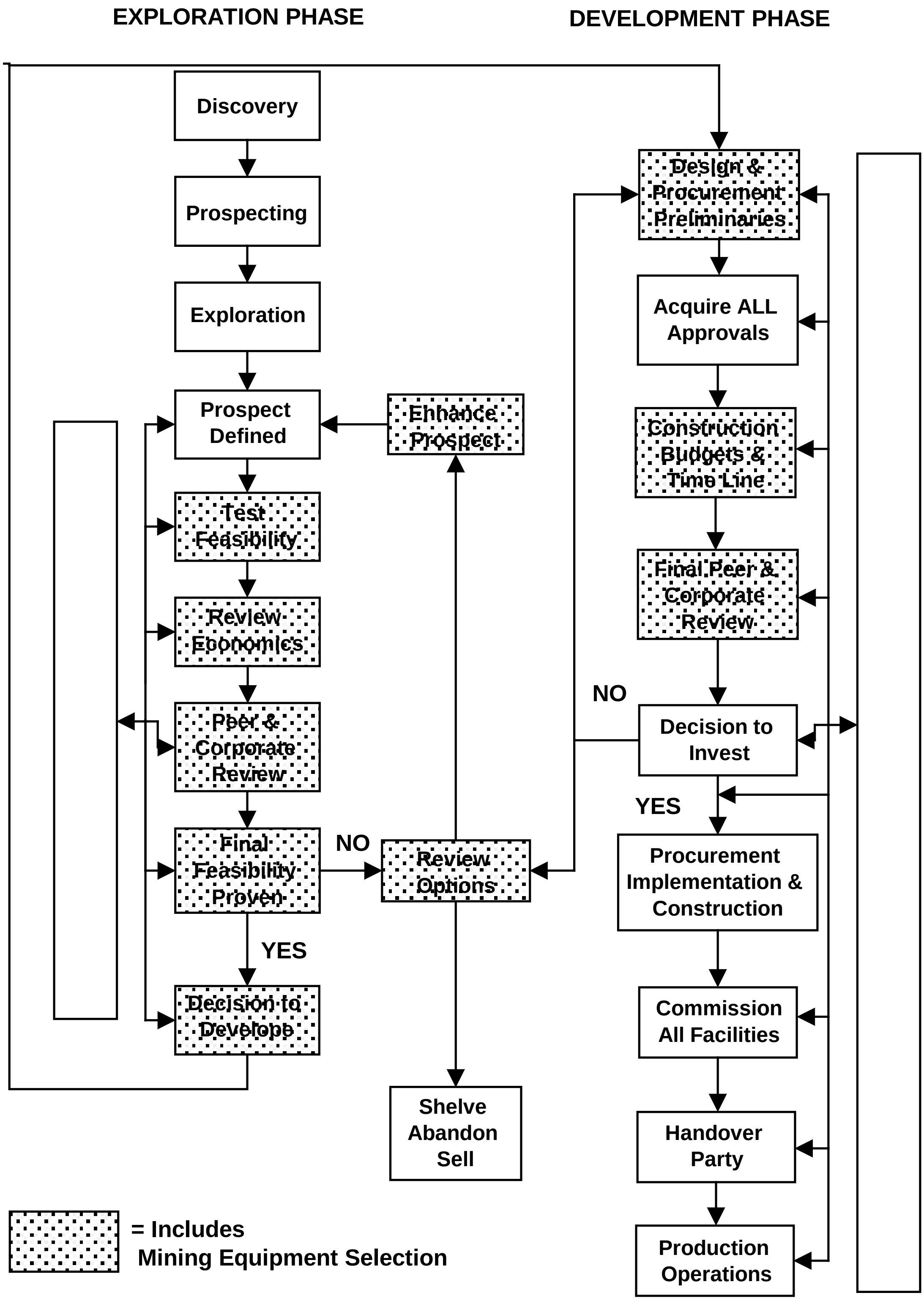


Figure 4.1 Generic Mining Development Outline

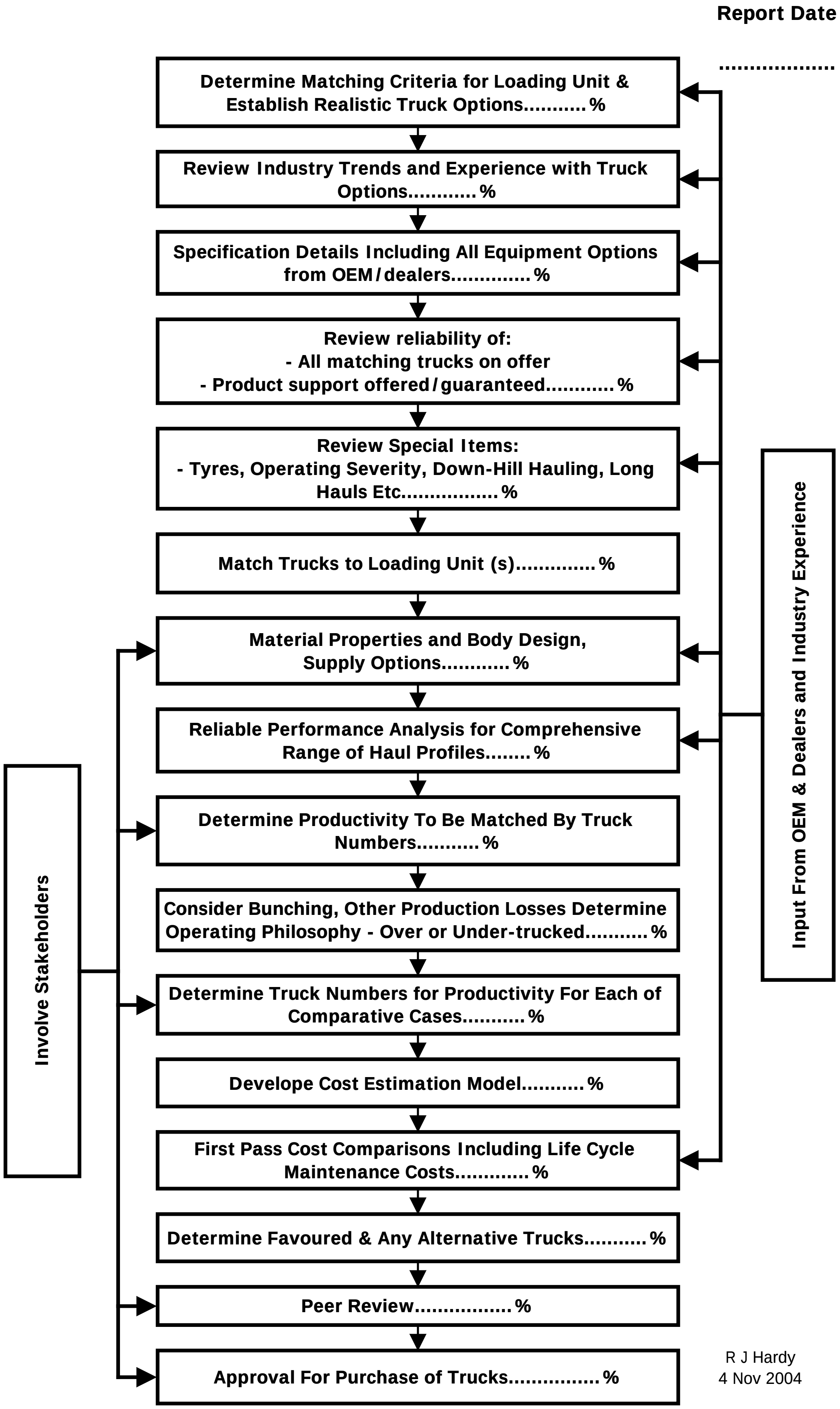


Figure 4.2 Truck Selection Process

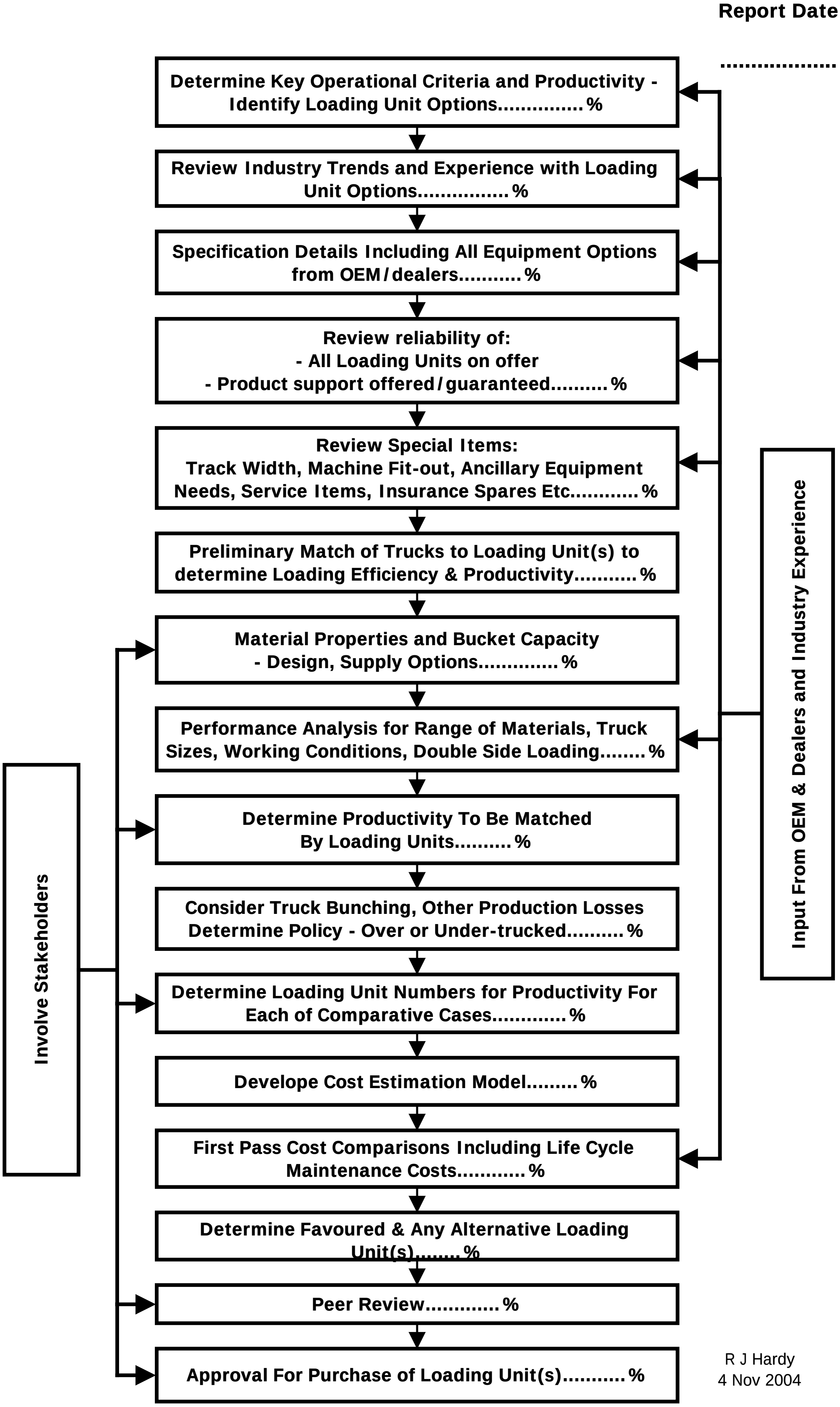


Figure 4.3 Loading Unit Selection Process

Table 4.1 - Potential Impacts of Non-optimum Truck Selection

Description of Issue	Potential Variance from Target	Potential Impact on Mining Costs	Potential Impact on Total Hauling Costs
	%	%	%
Purchase Price High	10	1	2
Maintenance Costs High	10	1	2
Truck Body Under Capacity	10	4	10
Low Average Truck Speed – Loaded	10	3	7
Reduced Tyre Life	30	3	6
Poor Truck or Supplier Reliability and Support	10*	4	10
Poor Loading Equipment - Truck Matching	10*	4	10

* - Resulting Production Loss

It will be noted that the greatest impacts result from causes that are generally within the control of mine operator or supplier. Inadequate product support from suppliers, poor truck reliability, mismatching of trucks and loading equipment and under-capacity truck bodies all have more significant cost impact than increases in capital or operating cost components or shortfall in intrinsic truck performance. These more significant issues have been addressed in some detail in Sections 3.3, 3.4 and 3.5. Obviously, product support by suppliers, reliability of truck options, and attention to body design, capacity and wear protection should be focal points during the investigative stage of truck selection.

4.1.4.2 Effects of Non-optimum Selection – Loading Equipment

Using a method similar to the truck analysis described in Section 4.1.4.1, Bruce Gregory has compared cost-impact due to non-optimum excavator selection in terms of a number of issues (Gregory#2, 2003).

Issues tested in terms of load and haul and total mining costs by Gregory included:

- Dilution and ore loss.
- Poor match of loading equipment with trucks.
- Loading inefficiency effect.
- Poor reliability of loading equipment including poor product support by supplier.

- Effect on load and haul costs due to non-optimum loading equipment capacity.

The above list was consolidated to four factors analyzed for effect on load and haul costs, total mining costs and ultimately profitability, viz.:

1. Dilution
2. Poor reliability (both equipment intrinsically and product support).
3. Match with trucks (non-optimum).
4. Inefficiency (inclusive of all causes).

Analysis in terms of profitability reduction ranked the factors in the order shown. The above four factors, especially dilution and equipment reliability, should obviously be the focus for the investigative stage of loading-equipment selection.

4.1.4.3 Loading Equipment Capacity and Load and Haul Costs

In the course of analyzing effect of non-optimum outcomes from testing loading equipment Bruce Gregory provides valuable insight into the relationship between loading equipment size for each truck payload capacity measured in terms of load and haul costs (Gregory#2. 2003).

Figure 4.5 is reproduced from Gregory's paper. Ordinate values are load and haul cost indices with lowest cost calculated during analysis set at 100%.

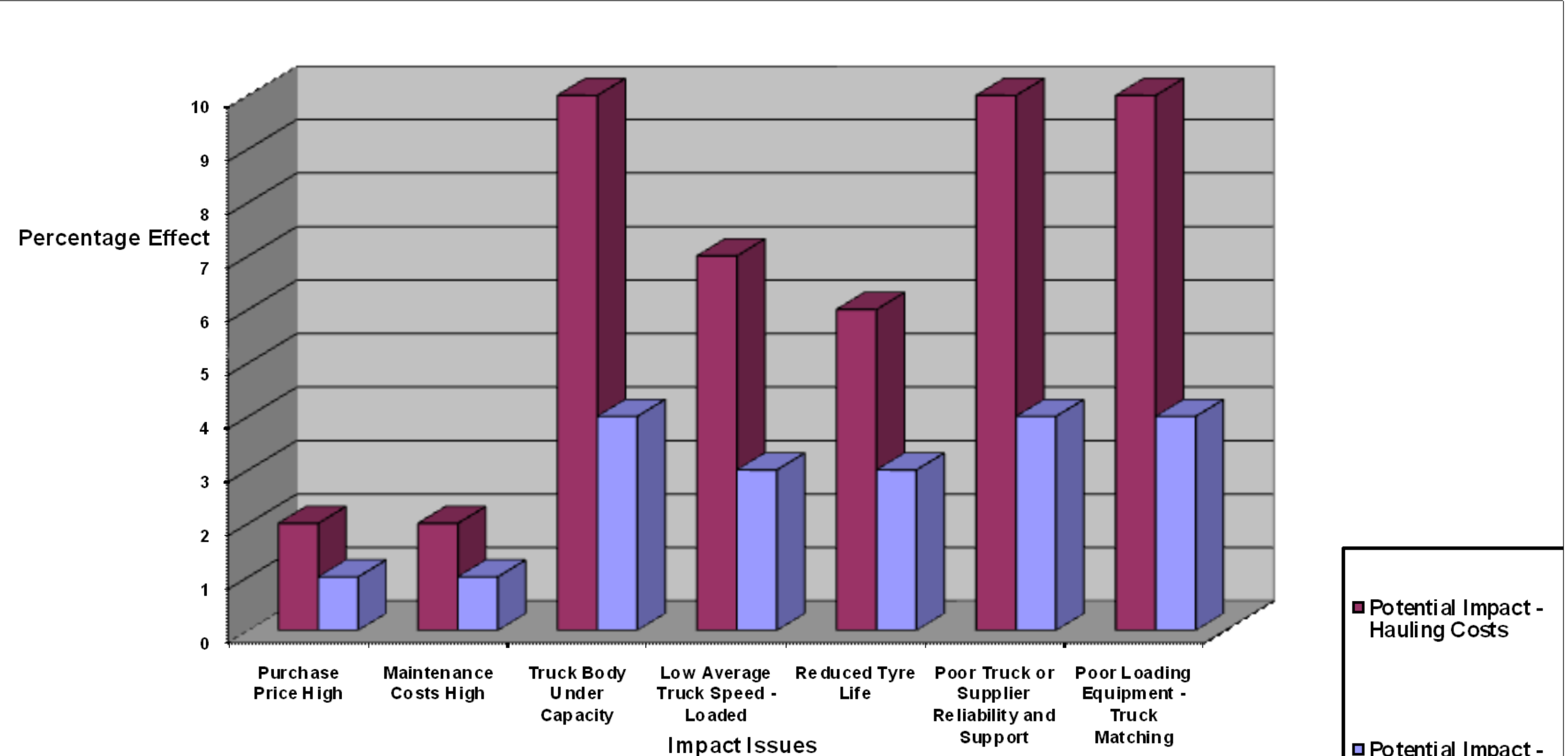


Figure 4.4. - Potential Impact of Non-optimum Truck Selection
Based on Gregory #1, 2002

From analysis and Figure 4.5 Gregory concluded that:

- Truck size is a significantly greater load-and-haul cost driver than loading equipment size.
- Lowest estimated load and haul costs were for the largest mining equipment analyzed.
- For a selected truck size, the largest loading equipment did not always result in the lowest load and haul index. Gregory explains this as mainly due to bucket – truck body matching.
- For any selected truck size, load and haul costs varied over a narrow range of 5% for different loading equipment sizes.

Further interesting interpretations of the results illustrated in Figure 4.5 include:

- As shown by Figure 2.2, (cost proportions updated to current prices in Section 5.1.3 –Table 5.1 and Figures 5.1 to 5.3) hauling cost is typically in the order of three times the cost of loading within total mining costs for a deep open pit. It is obvious that load and haul costs are not as sensitive to loading equipment size alternatives when compared over a range of trucks.
- Alternative loading equipment for any specific truck size will require different loading times involving different numbers of bucket passes. From Figure 4.5 load and haul costs are not particularly sensitive to changing size of loading equipment. This implies that load and haul costs are not particularly sensitive to number of bucket passes. This further supports the hypotheses in Sections 3.2.8 and 3.3.9 that increasing the number of bucket passes to realise the benefit of reducing the dispersion of truck payload distributions could be cost beneficial overall.
- The range of load and haul indices in Figure 4.5, for each truck size over the loading equipment options analyzed is small - some +/-2.5% for the smallest truck reducing to some +/-1.5% for the largest truck. Converting the comparison to total mining cost indices approximately halves these ranges – Figure 2.2 illustrates. If the comparison is considered in terms of total project costs or profitability, changing loading equipment size (measured by

operating weight) by, say, ± 50% for any selected truck fleet will have only small cost effect.

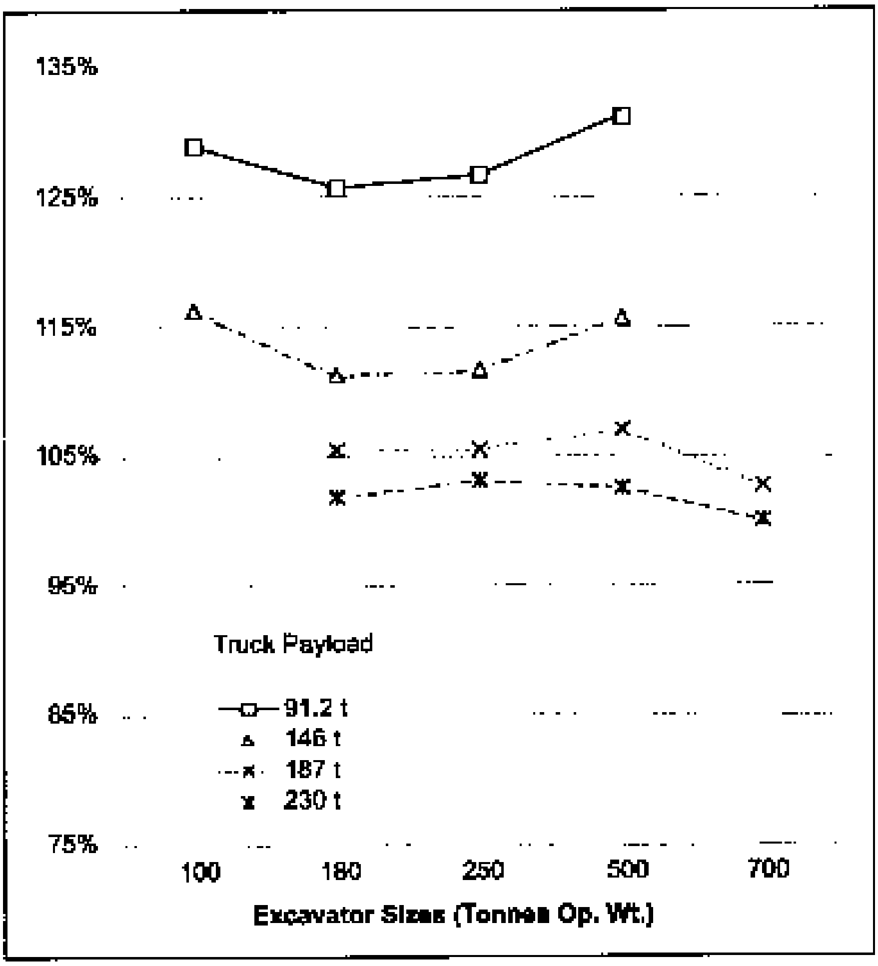


Figure 4.5

Relative Load and Haul Costs as Loading Equipment Scale Increases for a Range of Truck Payloads (Gregory#2, 2003)

If truck and loading equipment selections adopted are within reasonable practical limits, and generally satisfy productivity and other project requirements:

- Truck costs are the main determinant for load and haul costs, consequently most significantly influence total mining costs and have measurable effect on total project costs and profitability.
- Generally cost variability of loading equipment only imposes subtle effects on mining costs; but loading equipment provides specific technical capability

to, firstly ensure that trucks function as intended. Secondly loading equipment must accommodate all of the constraints to ensure that ore is extracted with minimum loss/dilution, in accordance with the mine plan and detailed pit plans based on stratigraphy of the ore and adequate geotechnical constraints to ensure safe operations.

- The comparative results illustrated by Figure 4.5 imply that, provided loading equipment satisfies requirements for selective mining and the condition and digging characteristics of material being loaded, an error of, say, one step in model range in selecting loading equipment size is, by a significant margin, not as significant as selecting a truck with non-optimum payload capacity.
- Finally, a decision for replacement of trucks with up-scaled truck models to benefit from reduced unit costs may not necessarily further significantly benefit from replacing loading equipment with up-scaled units.

Truck and loading equipment selection processes always need to apply detailed attention to the duties of each open pit mining operation.

4.1.5 Loading and Hauling Equipment Options

4.1.5.1 Mining Trucks

A sample of mining trucks offered by the principal truck manufacturers is analyzed in Table 4.2.

Figure 4.6 illustrates the comparative criteria that indicate relative performance of trucks, viz.,

- Engine power in kilowatts at the flywheel (i.e., nett of all engine auxiliaries, radiator fan, alternator, air service and air conditioning compressors) versus GMW – kW/tonne.
- Payload tonnes versus GMW tonnes.
- Engine power in kilowatts at the flywheel versus payload tonnes.

Where manufacturers offer a range of engines the highest specification configuration was chosen in compiling Table 4.2. Largest tyre options, if any, were also chosen.

Table 4.2 – Mining Truck Specifications

Manufacturer	Truck Model	Target Payload	GMW	Flywheel Power	Power/ GMW	Payload/GMW	Power/ Payload
Units		Tonnes	Tonnes	kW	kW/Tonne		kW/Tonne
Caterpillar	773E	54.3	99.3	501	5.05	0.55	9.23
	777D	90.7	163.3	699	4.28	0.56	7.70
	785C	140.0	249.5	1005	4.03	0.56	7.18
	789C	180.0	317.5	1335	4.20	0.57	7.42
	793C	218.0	383.7	1615	4.21	0.57	7.41
	797B	345.0	623.7	2513	4.03	0.55	7.28
Komatsu	HD465-7	55.0	98.8	533	5.39	0.56	9.69
	HD785-5	91.0	166.0	753	4.54	0.55	8.27
	HD985-5	105.0	178.8	753	4.21	0.59	7.17
	HD1500-5	140.7	249.5	1048	4.20	0.56	7.45
	*630E**	170.0	294.8	1388	4.71	0.58	8.16
	*730E**	186.0	324.3	1388	4.28	0.57	7.46
	*830E**	227.2	385.8	1761	4.56	0.59	7.75
	*930E***	290.4	499.0	1902	3.81	0.58	6.55
Liebherr	*T 252**	195.0	324.4	1510	4.65	0.60	7.74
	*T 262**	218.0	385.7	1510	3.91	0.57	6.93
	*Ti 272***	290.0	441.9	2014	4.56	0.66	6.94
	*T 282***	364.0	566.3	2610	4.61	0.64	7.17
Terex	*MT 3000**	108.3	206.3	839	4.07	0.52	7.75
	*MT 3300***	136.1	241.3	954	3.95	0.56	7.01
	*MT 3300***	136.0	248.9	1398	5.62	0.55	10.28
	*MT 3600B**	172.3	297.0	1398	4.71	0.58	8.11
	*MT 3700B**	185.9	317.0	1398	4.41	0.59	7.52
	*MT 4400**	229.9	390.9	1706	4.36	0.59	7.42
	*MT 4400AC***	217.7	390.1	1855	4.76	0.56	8.52
	*MT 5500B***	326.0	557.8	3415	6.12	0.58	10.48
Hitachi	EH 1700	94.3	170.1	836	4.91	0.55	8.87
	*EH 3500**	182.9	324.3	1411	4.35	0.56	7.71
	*EH 4500***	241.8	435.5	1973	4.53	0.56	8.16

*** Indicates DC wheel motors**
**** Indicates AC wheel motors**
Otherwise mechanical drive.

Data for Table 4.2 was extracted from manufacturer's specification brochures in early 2004. Specified payload was generally checked to correspond with GMW less NMW (i.e., tare). NMW includes a standard body liner kit where weight of the kit was supplied. The open pit mining industry seems to have tried and moved on from the light weight sacrifice-and-replace body concepts; where light-weight designs for bodies were fabricated from wear plate. It is the author's experience that body liners are generally installed in mining trucks. All GMW and payload data for Caterpillar trucks recorded herein are inclusive of adjustment for body liner kits.

This is not necessarily the case with all other manufacturers. In some cases, it was possible to assess the payload discount for body liners. In the majority of cases, this was not possible. Consequently due consideration needs to be given to the possibility that, except for Caterpillar trucks, payload-to-GMW may be overstated in the results in Table 4.2, Figure 4.6, and Figure 4.7. This possible error in the comparative data does not significantly affect general interpretations and conclusions drawn from them.

Figure 4.7 does not adequately illustrate the variability and range of the payload v. GMW ratio. Figure 4.7 is a scaled up illustration of payload tonnes versus GMW tonnes.

It will be noted that:

- With the exception of smaller mechanical-drive trucks, and largest AC electric drive trucks, the power kW v. GMW tonnes is in a reasonably narrow range of 4 to 5. So, comparative truck acceleration and speed performance will also have a limited range.
- The apparent advantageous power-to-weight ratio of medium to large electric drive trucks compared with mechanical drive trucks has to be tempered by the power-efficiency advantage from engine to rimpull at the road of mechanical drive trucks discussed in Section 3.3.4 and illustrated by Figure 3.29 – especially for DC drives.
- Power-to-payload ratio over the range of truck options naturally exhibits similar characteristics to power-to-GMW.
- Payload-to-GMW ratio indicates an absolute range from 0.52 to 0.66. The single value of 0.66 is for a Liebherr Ti 272, an advanced, monocoque-body design mentioned in 3.3.3 above and further discussed below. A core of ratio values from 0.52 to 0.59 represent the general realistic range for the standard configuration 2 x 4 wheel mining truck – with a realistic mean of 0.57.

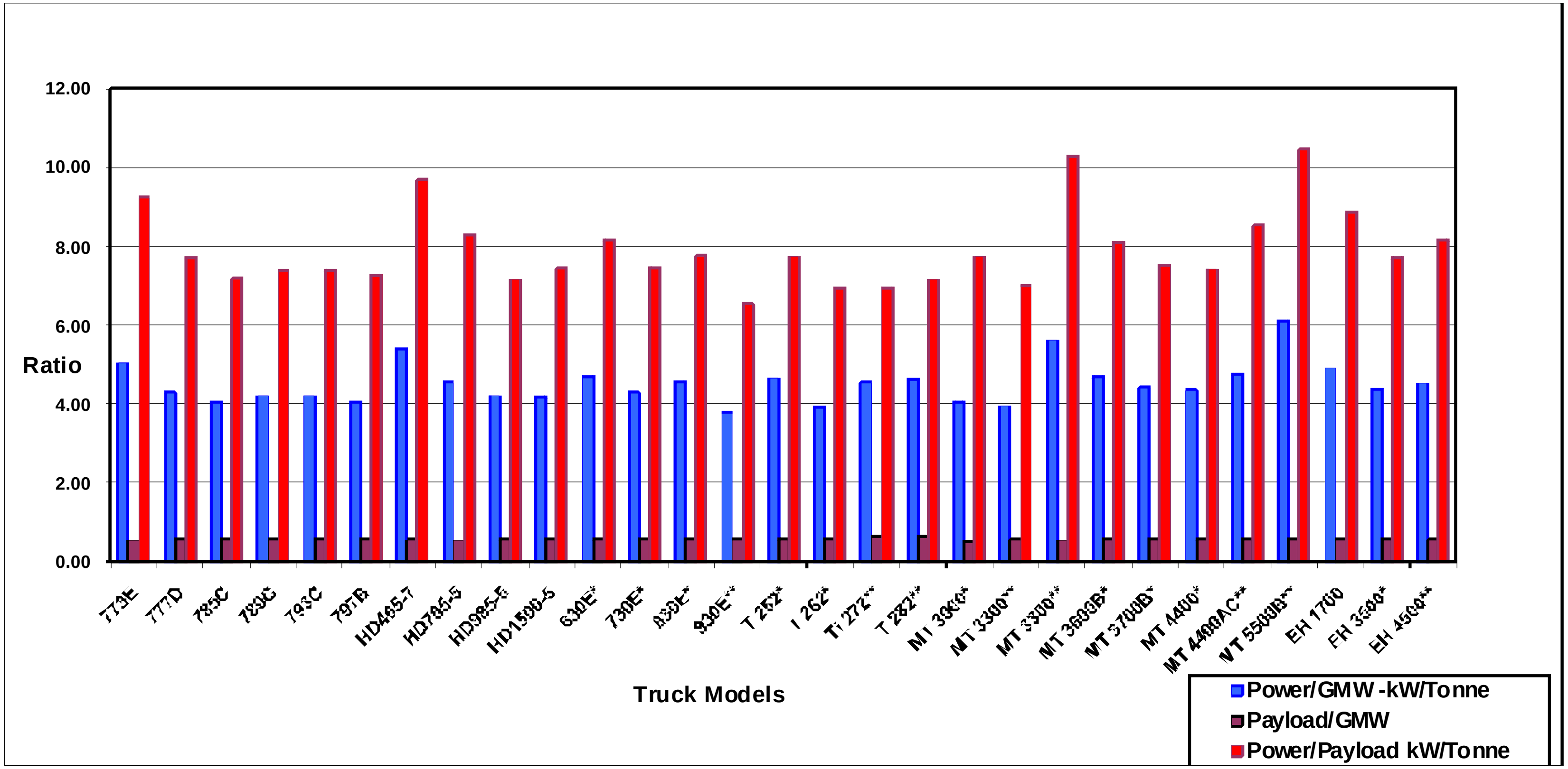


Figure 4.6 Mining Trucks - Comparison of Flywheel kW v. GMW, Payload V. GMW & kW v. Payload (From Table 4.2)

For Truck Models: * = DC Electric Drive, ** = AC Electric Drive

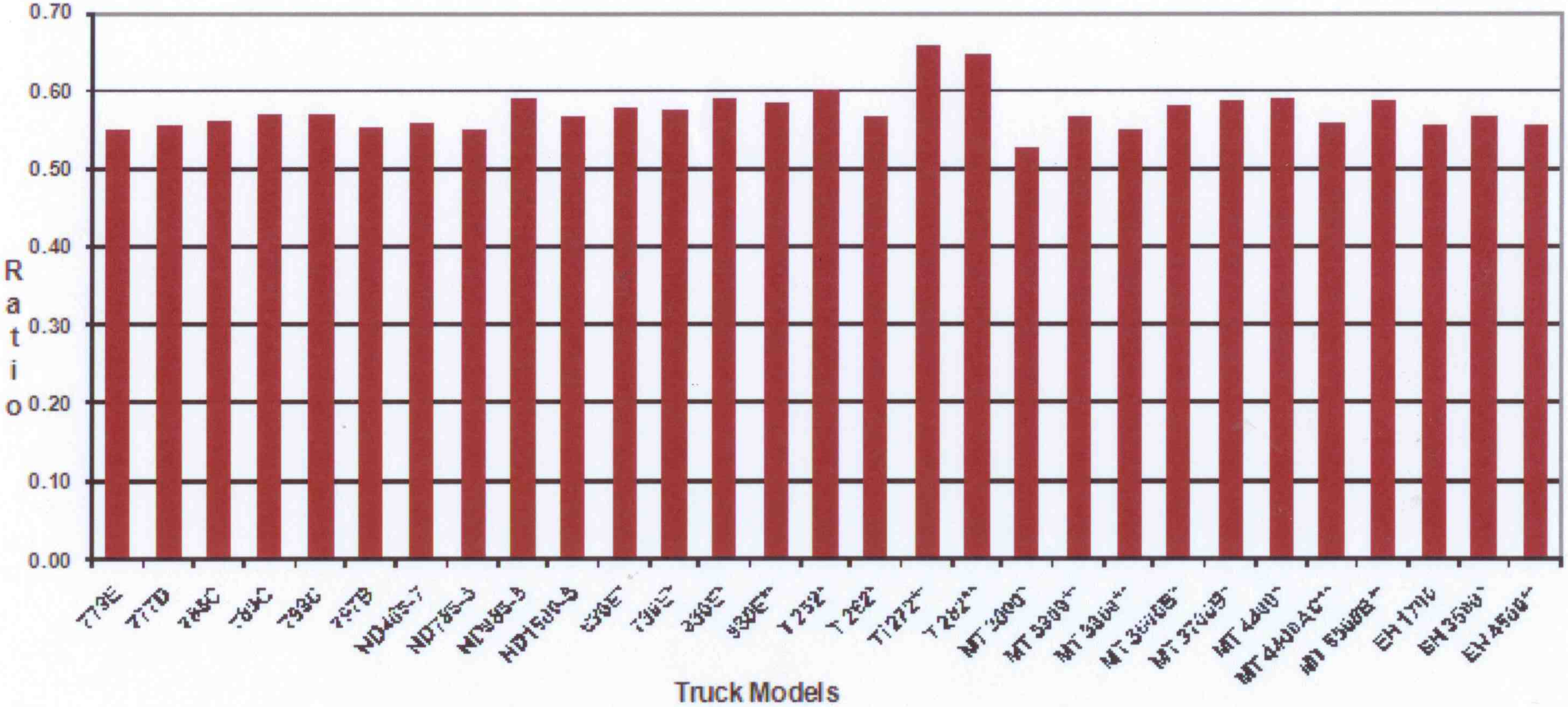


Figure 4.7 Mining Trucks – Payload v. GMW Ratio (From Table 4.2)

For Truck Models: * = DC Electric Drive, ** = AC Electric Drive

Adding power to a mining truck increases rimpull thus increasing acceleration and speed. But addition of power comes at a cost including:

- Measurable additional fuel burn and associated lube service costs.
- Potential tyre life reduction due to additional torque to produce the rimpull – the largest truck tyres exhibit tendency to shoulder separations associated with wall deformation resulting from acceleration torque and transverse distortion due to thrust loads at corners that are more critical as trucks increase in size (Cutler, 2002).
- A small increase in NMW in the order of 1 to 2 tonnes.
- Also, increased speed is accompanied by higher impact stresses through suspension and frame.

If existing truck designs cannot accommodate all resulting increased dynamic stresses, there may be need for structural upgrading and suspension up-rating, causing increased GMW.

Experience indicates that apparent significant increased capability in any one operating function tends to be offset, at least partially, by a decrease in performance and/or marginally increased cost. Apparent beneficial features of truck options revealed by due diligence and detailed investigation are generally partly or totally offset by compensating side effects that will be discovered by penetrating research.

In summary, due diligence and investigation of truck options are an absolute necessity for reliable equipment selection. Investigation must be exhaustive with the realisation that, for every apparent attractive feature of a specific truck, there will likely be offsetting features that necessarily need to be discovered.

4.1.5.2 Loading Equipment

A sample of electric rope shovels and hydraulic shovels offered by principal manufacturers are listed in Table 4.3. Figure 4.8 illustrates comparative criteria indicating relative performance of shovels, viz.,

- Operating weight (GMW) tonnes versus rated bucket cubic metres. Rated bucket cubic metres are struck for electric rope shovels and SAE heaped 2:1 for hydraulic shovels and excavators.

- Engine power in kilowatts at the flywheel (i.e., nett of all engine auxiliaries, radiator fan, alternator, air service and air conditioning compressors) versus rated bucket cubic metres.

Data for Table 4.3 was extracted from manufacturer's specification brochures in early 2004. The mid-range option was selected for operating weight where a range of track widths are offered. Bucket rated volume is based on a loose density of 1.8 tonnes per cubic metre.

Because of the significantly different configuration of the digging implement, bucket filling action and external electric power supply by trailing cable, comparison of rope shovels with hydraulic-powered loading equipment must be circumspect. Installed power for electric rope shovels was taken as the on-board transformer rating.

Table 4.3 Loading Equipment Specifications

Units		Cubic Metres	Tonnes	Tonnes	KW	Tonnes /CM	KW/CM
P & H	**4100XPB**	56	100	1428	2700	**25.50**	**48.21**
	4100A	47	85	1108	2250	**23.57**	**47.87**
	2800XPB	35	63	1033	1800	**29.51**	**51.43**
	2300XPB	25	45	651	1300	**26.04**	**52.00**
Liebherr	**R996**	34	61	656	2240	**19.29**	**65.88**
	R995	23	41	429	1600	**18.65**	**69.57**
	R994B	18	32	306	1120	**17.00**	**62.22**
Komatsu	**PC8000**	42	76	710	3000	**16.90**	**71.43**
	PC5500	26	47	515	1880	**19.81**	**72.31**
	PC4000	22	40	385	1400	**17.50**	**63.64**
	PC3000	15	27	255	940	**17.00**	**62.67**
	PC1800-6	11	20	184	676	**16.73**	**61.45**
O & K	**RH400**	47	85	1000	3280	**21.28**	**69.79**
	RH340	34	61	552	2240	**16.24**	**65.88**
	RH200	26	47	522	1880	**20.08**	**72.31**
	RH170	21	38	374	1492	**17.81**	**71.05**
	RH120E	15	27	283	1008	**18.87**	**67.20**
Hitachi	**EX8000**	40	72	780	2800	**19.50**	**70.00**
	EX5500	27	49	515	1870	**19.07**	**69.26**
	EX3600	21	38	350	1400	**16.67**	**66.67**
	EX2500	15	27	242	971	**16.13**	**64.73**
	EX1900	11	20	186	720	**16.91**	**65.45**

From Table 4.3 and Figure 4.8:

- For hydraulic shovels from a selection of the more prominent OEM, and a range of capacities, operating weight per rated cubic metre of bucket capacity is in a narrow range of 18 tonnes ±8% and power to cubic metre of bucket capacity of 67 kW, also ±8%.
- The sample of rope shovels is limited to one manufacturer and only four models. Operating weight to rated bucket capacity is some 26 tonnes ± 4% - a narrow range – evidence of generic design across the range.

The higher operating weight of electric rope shovels compared to hydraulic shovels is obvious from Table 4.3 and Figure 4.8. Additional mass results from increased size and consequent reduced mobility; and need for increased working area is offset by longer machine life; but with lower direct operating costs (excluding ownership costs), as further discussed in Section 5.2.4.

The above points indicate that intrinsic performance of available shovels of similar power and digging configuration and rated bucket capacity will be similar as will be their operating costs. Again, as with trucks, apparent differences in comparative productivity and unit operating cost of shovels in the process of equipment selection tend to be subtle rather than obvious. The decision to opt for specific loading equipment can be expected to be more dependent on cogent functional and operational necessities, and supplier support for equipment, than reliance on cost comparisons exhibiting small differences.

Conversely, if cost comparisons do indicate robust differences, checking and audit of cost compilations, both internal and independent, are likely warranted.

In summary, just as with trucks, due diligence and investigation of loading equipment options are an absolute requirement. But the investigation must be exhaustive with the realisation that, for every apparent attractive feature of a loading equipment option there will likely be offsetting features that necessarily must be discovered

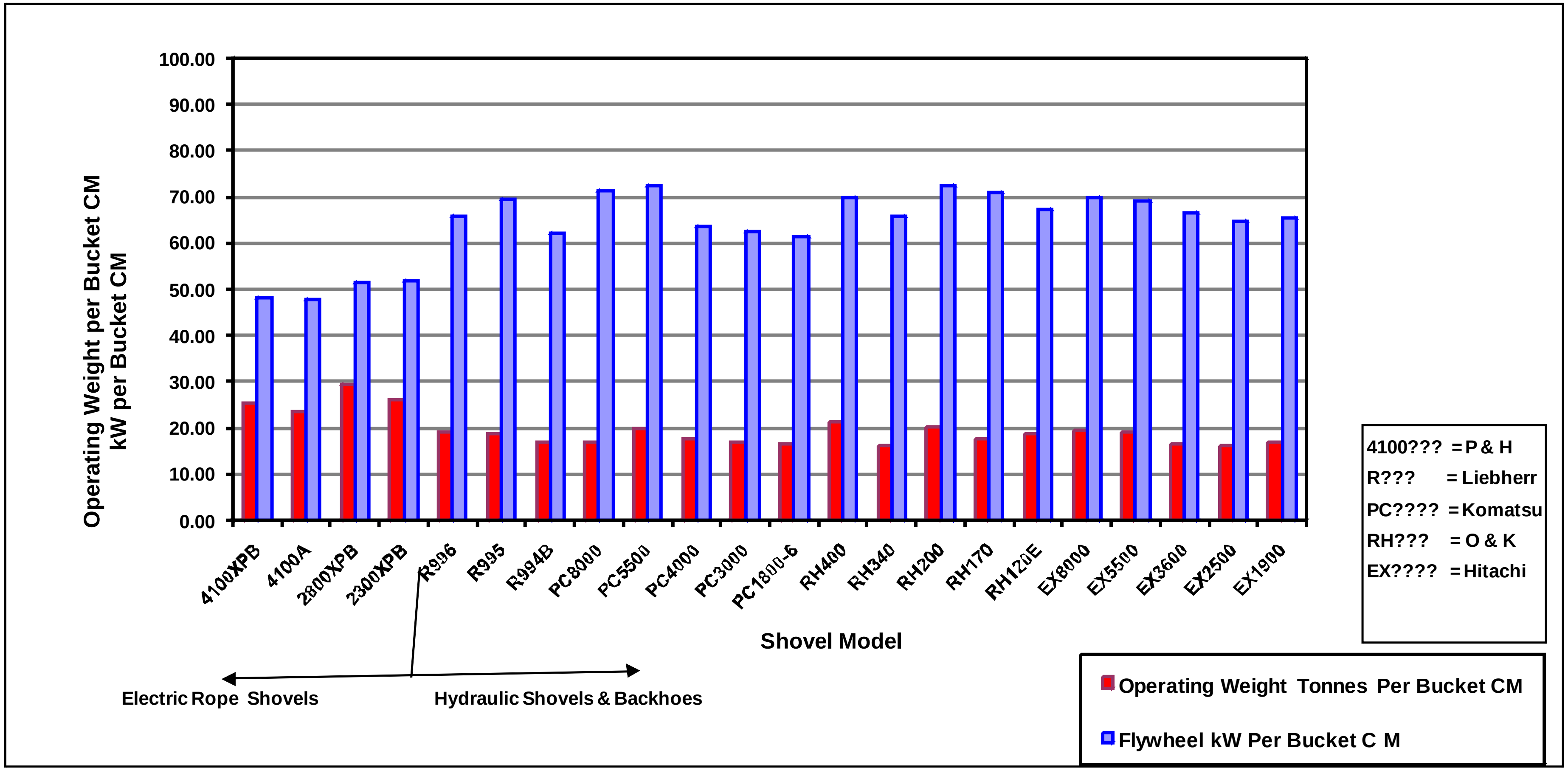


Figure 4.8 Loading Equipment - Comparison of Flywheel kW and Operating Weight Per Bucket Cubic Metre (From Table 4.3)

Wheel Loaders

Large wheel loaders still enjoy a small market share for loading large mining trucks. Examples of wheel loaders with mechanical drive and hydraulic-powered loading implements are available to 205 tonnes operating mass and nominal 36 tonnes bucket capacity with the market favourite at 190 tonnes operating weight and nominal 34 tonnes bucket capacity. High lift configurations for loading large mining trucks are some 1% heavier and nominal bucket capacity is reduced by some 5%. Electric wheel loaders with hydraulic powered loading implements are available to 245 tonnes with bucket capacity of 72 tonnes.

Wheel loaders have mobility, advantageous in multi-pit operations, to clean up small quantities, to finish up to a face allowing less mobile shovels, including electric rope shovels, to move to a new digging location avoiding delays in clearing an area for blasting. There are applications of large wheel loaders as the prime loading equipment in waste stripping and bulk mining operations. But, application of loaders for bulk of production is generally rare compared with rope shovels, hydraulic shovels and excavators.

For the purposes of research, analysis was confined to shovels.

4.2 LARGER MINING EQUIPMENT

4.2.1 Introduction

Diminishing prices for commodities, with the subsequent need to increase operating scale, and the demand for larger mining equipment to realize decreased costs, was discussed in Section 2.3.2.

Consequently the focus of the open pit mining industry has been on larger equipment with the objective of reducing unit-mining costs. More recently, the thrust for ever-increasing scale of mining equipment has been tempered by the realization that cost benefits of increasing scale tend to diminish as evidenced in technical literature.

“There are indications that some difficulties have accompanied the introduction of larger equipment into mines such as complexity, dilution, lost production and reduction of flexibility.” (Bozorgebrahimi, 2003).

Benefits by way of reduced unit cost per tonne mined from larger equipment are uncertain. There is suggestion that "the size of haul trucks and excavators has reached a feasibility threshold where the economies of scale have peaked" – attributed to the RAND Institute (Bozorgebrahimi, 2003).

Significant issues arising from increasing loading and hauling equipment scale are discussed and analyzed in Sections 4.2.2 to 4.2.5.

4.2.2 Diminishing Cost Benefit

The reducing cost-driven development of "ultra-class" (UC) mining trucks - 290 tonnes and higher payloads - was discussed in Section 2.1.3 and illustrated by Figure 2.1. Causes for diminishing haulage unit-cost-benefits with increasing truck size can be reasoned from discussions and comparative data tabulated and illustrated in Section 4.1.5. Similar ratios for power v. GMW, power v. payload; and payload v. GMW over the range of mining trucks, including UC trucks, from different OEM, illustrated by Figures 4.6, and 4.7, are a basis for understanding that performance of conventional mining trucks per tonne of payload tends to be practically uniform.

Cost drivers for operating mining trucks can be conveniently categorized as:

1. Owning costs, including capital amortization, insurance and interest cost of capital.
2. Fuel
3. Lubricants and filters.
4. Maintenance, including component rebuilds and labour to remove, repair and replace.
5. Service labour for fuelling, daily top ups, inspections, field checks and opportune minor repairs, supervision and management.
6. Tyres – replacement and management, including labour.
7. Operating labour.

Other authors and sources may use more detailed categories than the seven listed above. But for the purposes of the research the above seven categories are convenient and sufficient.

4.2.2.1 Fuel Consumption per Payload Tonne per Kilometre Constancy

Fuel consumption per tonne of payload, per kilometre of one-way haul, is practically independent of truck size and haul distance for open pits of comparative geometrical configuration. Fuel consumption per tonne - km over a range of truck payload capacities and haul distances is shown in Table 3.63, and illustrated by Figure 3.60.

The standard measure for fuel consumption of diesel engines is Brake Specific Fuel Consumption (BSF) expressed as litres-per-kW or grams-per-kW of fuel, with air at 25^0 standard temperature, 99 kPa pressure, diesel fuel with specific gravity 0.84 and viscosity compliant with API (American Petroleum Institute) standard gravity of 35. This is a hypothetical test condition where engines are dynamometer tested specifically for BSFC. In practical application, fuel is at a fuel tank temperature that is a derivative of ambient operating temperature and bypass fuel temperature returning to the tank. Resulting fuel temperature as delivered to the fuel pump tends to build up to an equilibrium level significantly higher than ambient temperature. At these higher temperatures, volume of fuel consumed for equivalent power is higher than the standard BSFC. Tuning the engine for rated performance becomes more difficult as operating temperature increases. Volumetric fuel consumption increases with operating ambient temperatures. OEM's offer a fuel-cooling option to reduce volumetric consumption for large mining trucks operating in high ambient conditions,

The term BSFC is historical rather than indicative of current practice in assessing power rating of engines. Early testing of fuel consumption of engines involved applying a friction brake to the engine output shaft, often on the periphery of the flywheel, running the engine at constant measured speed, measuring brake torque, deriving brake horsepower (BHP) and measuring fuel consumption – so BSFC. Currently BSFC is derived during dynamometer testing of engine power output. Engines being tested drive a hydraulic device that measures engine torque providing similar facility as the historical "brake".

Table 4.4 lists typical practical fuel consumptions in litres per kW for Caterpillar mining trucks inclusive of all practical operating allowances. These data are indicative only. Mining trucks from other manufacturers can be expected to exhibit similar specific fuel consumption performance. BSFC data in Table 4.4 show the benefit of longer stroke high displacement (HD) engines offered by Caterpillar as an option in the nominal 220 tonne and standard for UC trucks.

Table 4.4 Fuel Consumption of Caterpillar Mining Trucks
*** Includes 5% allowance for consumption in non-productive time**

Truck Model	BSFC		Recommended Practical Fuel Consumption		
	Gm/kW	Litres /kW	Engine kW	Deep Pit Load Factor	Typical Litres/Hr*
777D	**205**	**0.244**	746	0.45	**86**
785C	**204**	**0.243**	1082	0.45	**124**
789C	**204**	**0.243**	1417	0.45	**163**
793C	**214**	**0.255**	1716	0.45	**207**
793HD	**206**	**0.245**	1716	0.45	**199**
797B	**209**	**0.249**	2648	0.45	**311**

Life of engines is directly related to total fuel burn. Caterpillar has determined the engine life to total fuel burn relationship for their line of mining trucks. Table 4.5 provides the related data. Research has focused on “deep open pit applications”, equating to “severe service” in Table 4.5.

From Table 4.5 it can be seen that, by promoting an overly-optimistic engine life for the duty and service of an operation, actual engine life is unlikely to match expectancy and the risk of a shortfall in maintenance budget provisions will increase significantly. In summary, hauling work done is commensurate with cumulative fuel consumption that determines engine life; and, it is hypothesized, life of driveline components in general.

Driveline components transmit power to tyres where torque produces rimpull-driving force at the road. Each component can be expected to respond with life expectancy corresponding to individual total power transmission that can be measured as cumulative fuel burn.

This is the basis for concluding there is direct relationship between cumulative fuel consumption and lives of all driveline components including tyres, in respect of fair wear and tear of components.

The implications of this conclusion are revisited in Sections 5.2.4 and 5.2.5 for fuel and directly related haul cost drivers together with the relationships of cost drivers that are clearly not dependent on fuel consumption.

Table 4.5 Estimated Engine Life In Terms of Fuel Burn & Fuel Burn Rates for Mining Trucks

Caterpillar Truck Model	797		793C H/D		793C		789C		785C		777D	
Litres to Overhaul	3,785,400		2,786,000		2,460,518		2,490,800		1,892,700		1,137,000	
	Fuel Burn L/Hour	Engine Life Hours	Fuel Burn L/Hour	Engine Life Hours	Fuel Burn L/Hour	Engine Life Hours	Fuel Burn L/Hour	Engine Life Hours	Fuel Burn L/Hour	Engine Life Hours	Fuel Burn L/Hour	Engine Life Hours
	140	27,039	100	27,860	105	23,434	90	27,676	60	31,545	30	37,900
	150	25,236	110	25,327	115	21,396	95	26,219	65	29,118	35	32,486
	160	23,659	120	23,217	125	19,684	100	24,908	70	27,039	40	28,425
	170	22,267	130	21,431	135	18,226	105	23,722	75	25,236	45	25,267
	180	21,030	140	19,900	140	17,575	110	22,644	80	23,659	50	22,740
	190	19,923	145	19,214	145	16,969	115	21,659	85	22,267	55	20,673
	200	18,927	150	18,573	150	16,403	120	20,757	90	21,030	60	18,950
Fuel Burn for Selected Medium Service Engine Life	**210**	**18,000**	**155**	**18,000**	**154**	**16,000**	**125**	**20,000**	**95**	**20,000**	**65**	**17,500**
	220	17,206	160	17,413	160	15,378	130	19,160	100	18,927	70	16,243
	230	16,458	165	16,885	165	14,912	135	18,450	105	18,026	75	15,160
	240	15,773	170	16,388	170	14,474	140	17,791	110	17,206	80	14,213
	250	15,142	175	15,920	175	14,060	145	17,178	115	16,458	***85***	***13,376***
	260	14,559	185	15,059	185	13,300	155	16,070	120	15,773	90	12,633
Severe Service - Deep Open Pits	270	14,020	***195***	***14,287***	***195***	***12,618***	***165***	***15,096***	***125***	***15,142***	95	11,968
	280	***13,519***	200	13,930	205	12,003	175	14,233	130	14,559	100	11,370
	Indicative Fuel Burn for a Range of Duties											
Light Service	***152***	***24904***	***107***	***26037***	***107***	***22995***	***85***	***29304***	***66***	***28677***	***44***	***25841***
Medium Service	***214***	***17689***	***151***	***18450***	***151***	***16295***	***122***	***20416***	***94***	***20135***	***63***	***18048***
Severe Service	***274***	***13815***	***193***	***14435***	***193***	***12749***	***163***	***15281***	***122***	***15514***	***84***	***13536***

4.2.3 Is Bigger Better?

4.2.3.1 Ultra Class Mining Trucks - Application

Over the past 8 to 10 years demand for ever-increasing payload capacity of mining trucks has levelled out at the so-called “ultra class” (UC) truck in the order of 350 tonnes payload capacity. Specialised high-production applications that can justify UC trucks are limited, so numbers of currently operating UC trucks are small. Experience and operating history is accumulating slowly. Early experiences have been uncertain with a mix of success, unconvincing experience and disappointments. Uncertainty about the application and general benefits from UC trucks was raised in Section 4.2.1.

UC trucks were first offered in the late 1990's. Demand for UC trucks has increased in the past five years with recent operating experience leading to uncertainty and the question: "Is bigger better?" Articles in mining commercial literature, such as World Mining Equipment, September 2002 issue, featuring a number of reports of UC truck purchases and applications, have addressed this question in journalistic terms rather than technically analytical treatments.

Bristol Voss (2001) reporting on a seminar at Alberta University, Canada quoted Gerry Krause of Syncrude (tar sands mining in Canada) when he commented on some common concerns with UC trucks. Krause was reported as follows: "Is going from a fleet of eight small trucks to four large trucks the same as going from a fleet of 120 small trucks to 60 large trucks?" Further points attributed to Krause included:

- Out-of-service trucks have significantly more a-productive effect on small fleets.
- Larger trucks require wider haul roads with a total mining volume penalty in deep open pits.
- Strip coalmines might find large trucks economic where a kimberlite pipe diamond mine (≡ *"deep open pit"*) may not.
- Capitalization of the whole system of supporting facilities that must be compatible with larger trucks becomes more significant.
- Road construction and maintenance equipment must be upgraded for larger trucks.
- Tyre performance for UC trucks has been disappointing with the learning curve of operators and suppliers still trending upwards.
- Impacts on mine planning for UC trucks that need larger manœuvring areas.

(Voss, 2001)

General conclusions on UC trucks in summary:

- UC trucks are only acceptably effective for specialised, high-volume operations – deep open pits are unlikely to justify the adoption of UC trucks at this stage of development.

- UC trucks are not a panacea for reducing mine-haulage costs in any but a small number of specialised, large scale operating conditions.
- Replacing a fleet of smaller trucks with equivalent payload capacity of fewer UC trucks is unlikely to improve total production or reduce unit-operating costs – any cost benefit will be realized from reduced investment in infrastructure and associated ongoing costs.
- Mixed fleets of UC and smaller trucks are unlikely to deliver any significant unit cost benefit. It is more likely that the mix of performances will inhibit "rhythm" of hauling operations and excessive bunching inefficiency is likely an unacceptable risk.

In short: "greenfields", large-scale bulk mining operations without extended steep grades suit UC mining trucks. Otherwise, trucks in the nominal 220 tonne range are likely a reasonable practical limit to be exceeded only with due caution for the truck options currently available.

Significant issues for up-sizing mining trucks were also discussed by the author in a paper published in 2003. (Hardy#2, 2003). Engineering effort must address accommodation of larger mining trucks within detailed pit plans by increasing ramp dimensions and upgrading haul-road construction specifications to accommodate increased GMW. Also, servicing facilities, tooling and equipment require an exponential improvement as trucks approach UC specifications. The general effect of the management accommodations for increasing truck scale is illustrated by Figure 4.9, an amended version of Figure 2.1, referred to in Section 4.2.2.

The additional trend lines indicate possible outcomes from mine planning accommodations of larger trucks and a further accommodation for infrastructure upsizing. The tendency for the additional "accommodation" costs for UC mining trucks is diagrammatically indicated by the upward trends and the uncertainty of the relative haul costs of UC trucks by the query – "**?**".

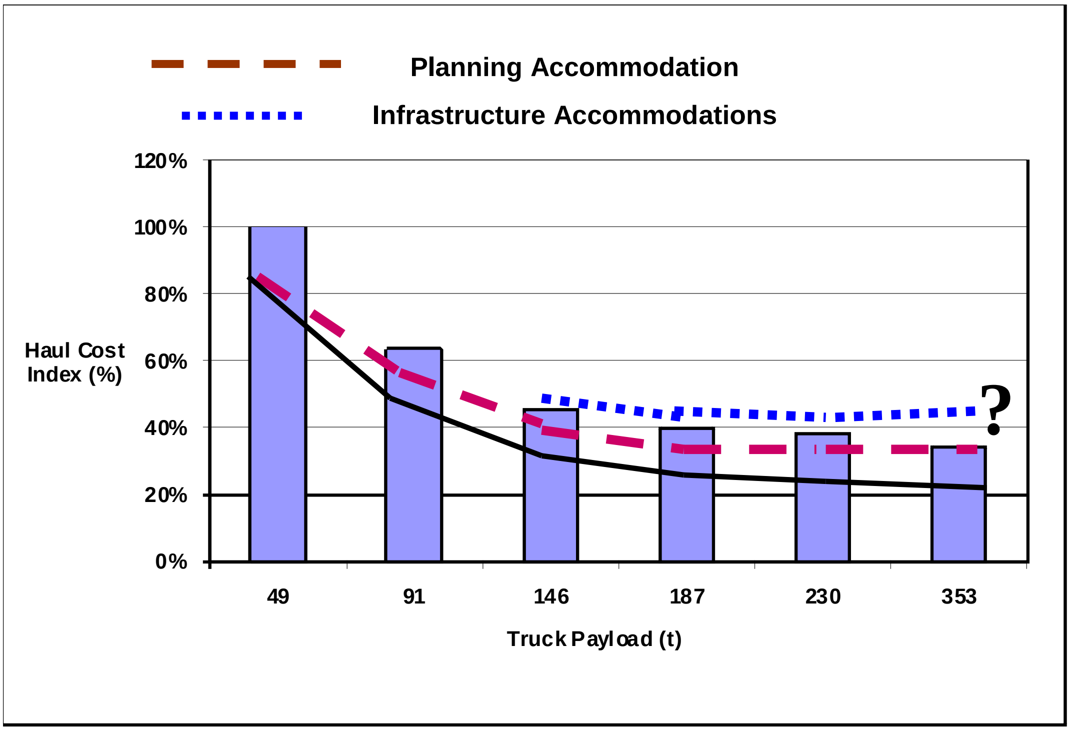


Figure 4.9 Hauling Cost Trend with Truck Scale (From Figure 2.1)

4.2.4 Future Mining Equipment Improvements

4,2,4,1 Mining Trucks

Payload/GMW ratio for the generic model range of mining trucks historically offered by OEM has been generally about 0.55. This was briefly discussed in Section 3.3. Following is a quote from a paper by the author published in 2003 (Hardy#2, 2003). "Similar power/GMW ratios, ensures that trucks of larger scale are backwards compatible in terms of rimpull/speed performance (with a few notable exceptions) to enable operation of mixed fleets with the aim of limiting queuing frustrations and resulting production loss. OEM's have tended to sustain generic designs that are essentially scaling up of trucks with similar configurations. Liebherr has broken away from generic constraint and produced an AC electric drive mining truck with a payload/GMW ratio of 0.66. Innovative features of this truck include: independent rear suspension, utilisation of the truck body as an adjunct structural element allowing reduction in frame (chassis) weight, and improved suspension characteristics. Like all innovative designs, the concept is taking some time to gain acceptance and work up to traditional levels of reliability. But the concept is a positive step towards the next generation of mining trucks.”

Table 4.6 Productivity and Cost Indices v. Payload GMW Ratio

Payload/GMW Ratio	Productivity Index	Unit Cost Index
0.55	1.000	1.000
0.57	1.036	0.965
0.59	1.073	0.932
0.61	1.109	0.902
0.63	1.145	0.873
0.65	1.182	0.846
0.67	1.218	0.821

Table 4.6 and Figure 4.10 simply illustrate productivity and cost benefits from improving payload/GMW ratio. Increasing payload/GMW ratio obviously has total conversion to unit cost benefit so making it the most attractive improvement avenue. Payload/GMW improvement on a smaller scale has been discussed earlier in Sections 3.3.3 and 3.3.5 where benefits from NMW savings from lighter-weight body-design initiatives were described as “subtle rather than startling”.

Lighter weight body designs have been accompanied by reduced body life and increased maintenance that tend to offset the “subtle” benefits of payload increase.

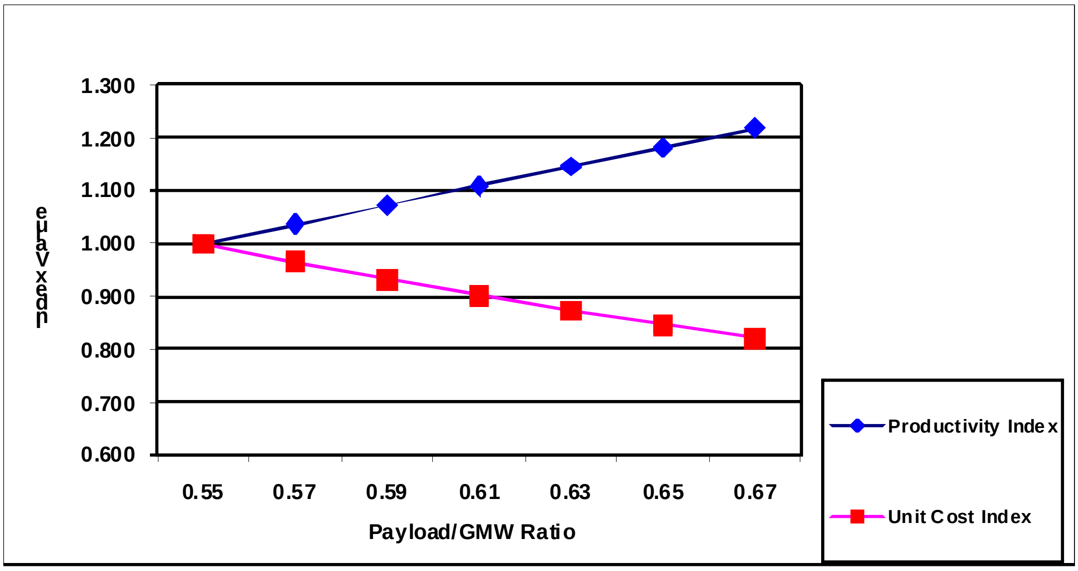


Figure 4.10 Productivity and Cost Indices v. Payload/GMW Ratio
(From Table 4.6)

Other potential improvements, such as increasing power for enhanced up-ramp speed, are subject to offsetting costs for fuel or power (trolley assist systems); increased driveline component weight so increased GMW, albeit small, and accelerated maintenance. Improvements involving increased operating speeds tend to have a flow-on, accelerated maintenance cost through the entire vehicle including frame and suspension. All of these offsetting limitations to improvement initiatives for increased productivity increase the attraction of direct improvement by significant design changes to mining truck configuration such as Liebherr's Ti272 described in Section 3.3.3.

Mining equipment of the future will be expected to continue to accommodate ever-reducing commodity prices - Figure 1.1 - and the consequent need for mining costs, including haulage, to reduce and sustain acceptable profit margins.

The author's opinion has not changed in respect of the potential for mining truck improvement to deliver reduced haulage costs since first writing and publishing in 2003. OEM's and the mining-truck supplier networks will continue to seek improved performance aiming for reduced unit cost. Every little cost-saving contribution is obviously beneficial. But mining truck development is at the stage where the more radical design options, departures from generic design limitations, are more likely to realize significant productivity and cost benefits.

4.2.4.2 Loading Equipment

Future focus for improved productivity and loading costs over the range of types and bucket capacities of loading equipment is not as clearly defined as for mining trucks.

The ubiquitous electric rope shovel and its forerunner the steam shovel, initially loading rail wagons in open pit mines, have a much longer history than mining trucks. For applications such as strip coal mining, open pit iron ore mining and similar bulk mining applications, where working areas are large, shovel relocations infrequent and large bucket capacity is attractive, electric rope shovels continue to enjoy market share. Improvements in rope shovels tend to be modest refinements and adoption of enhanced electric power and control technology as it develops.

More recently developed hydraulic excavators and mining shovels are in a continuous improvement phase with larger bucket capacities being offered to close the capacity gap to the larger electric rope shovels. Diesel-powered hydraulic systems are becoming more efficient so reducing fuel consumption.

The most attractive option for low capital and operating cost improvements is likely reducing bucket cycle time, especially swing times with bucket loaded and empty. These events are dependent on the cumulative installed power of slewing electric or hydraulic motors that is only a proportion of the total power available at bucket breakout in the digging segment of the bucket cycle. There are examples where both electric and hydraulic shovel upgrades have included additional slewing motor/transmission groups to retain bucket cycle time for larger bucket capacity that in turn attracts structural/mechanical upgrade resulting in a heavier digging implement group. It is a small design step to further enhance slewing power realizing faster cycle time within practical limits. Operator comfort at faster slewing speeds may become an issue. One solution could be for the operator’s cab to travel opposite to the slew slowly over a small tangential distance to reduce effective slewing acceleration and G force on the operator. Higher braking duty could be facilitated by power regeneration, electrically or inert-gas-over-hydraulic accumulation. The electrical and hydraulic engineering technology is available for increased slewing acceleration and rotating speed.

4.2.5 Hypothetical 500 Tonne Payload Truck

During reading and collection of thoughts and ideas that have been expanded in the research described herein, constancy and predictability of existing generic mining truck designs became obvious. This initiated the idea that the profile of a 500 tonne payload truck could be predicted by extrapolating characteristic criteria for mining trucks.

The generic nature of current truck designs offered by OEM is shown to be consistent and predictable by the following analysis.

Table 4.7 shows how equipment characteristics vary over the payload range of Caterpillar mining trucks.

Table 4.7 Comparative Indices - Caterpillar Mining Trucks Plus Hypothetical 500 Tonne Payload Truck

Caterpillar Truck Model Nominal Payload	Tonnes	777D 90.7	785C 140	789C 180	793C 218	797B 345	"799?" *500*
GMW	Tonnes	163.3	249.5	317.5	383.7	623.7	***891.3***
Index		1.00	1.53	1.94	2.35	3.82	***5.46***
Payload/GMW	Ratio	0.56	0.56	0.57	0.57	0.55	***0.56***
Engine Power	kW	699	1005	1335	1615	2513	***3661***
Index		1.00	1.44	1.91	2.31	3.60	***5.24***
Engine kW/Tonne Payload	kW/Tonne	7.70	7.18	7.42	7.41	7.28	***7.32***
	Index	1.00	0.93	0.96	0.96	0.95	***0.95***
Purchase Price	Au$M	1.80	2.73	3.21	4.27	6.45	***9.55***
Index		1.00	1.52	1.78	2.37	3.58	***5.30***
Purchase Price/Payload Tonne	$ x 10^3 /Tonne	19.85	19.50	17.83	19.59	18.70	***19.09***
	Index	1.00	0.98	0.90	0.99	0.94	***0.96***
Typical Fuel Consumption – Table 4.4	Litres/Hr	86	124	163	199	311	***451***
Index		1.00	1.44	1.89	2.31	3.62	***5.24***
	L/Payload Tonne	0.95	0.89	0.90	0.91	0.90	***0.90***

Data for Table 4.7 was derived from Tables 4.2 and 4.4. Trend lines for each of the four characteristics, GMW, Engine Power, Purchase Price and Fuel Consumption have been established:

- Initially as quotients (Payload/GMW ratio, and Fuel Consumption per Payload Tonne - km); or
- Indices on the basis of the Cat 777D assigned an index of 1 for Engine Power – kW and Purchase Price $M, both per Payload Tonne.

Further trend lines were established for each of the above four characteristics as indices sensitive to payload of each truck based on the Cat 777D with an index of 1.

Figure 4.11 compares the initial indices over the range of payloads. The obvious practically constant values of these indices over the range are a product of the nature of generic design and the static, dynamic and thermodynamic relationships that obey basic physical laws. Also, the cost of construction of equipment tends to have a linear relationship with NMW – so the conforming trend of purchase price with payload.

Dotted trend lines and open markers at each data point show extrapolated values for a hypothetical 500 tonne payload truck. Consistent indices over the generic design range provide a reasonable basis for projection of values of characteristics of a hypothetical 500 tonne payload truck.

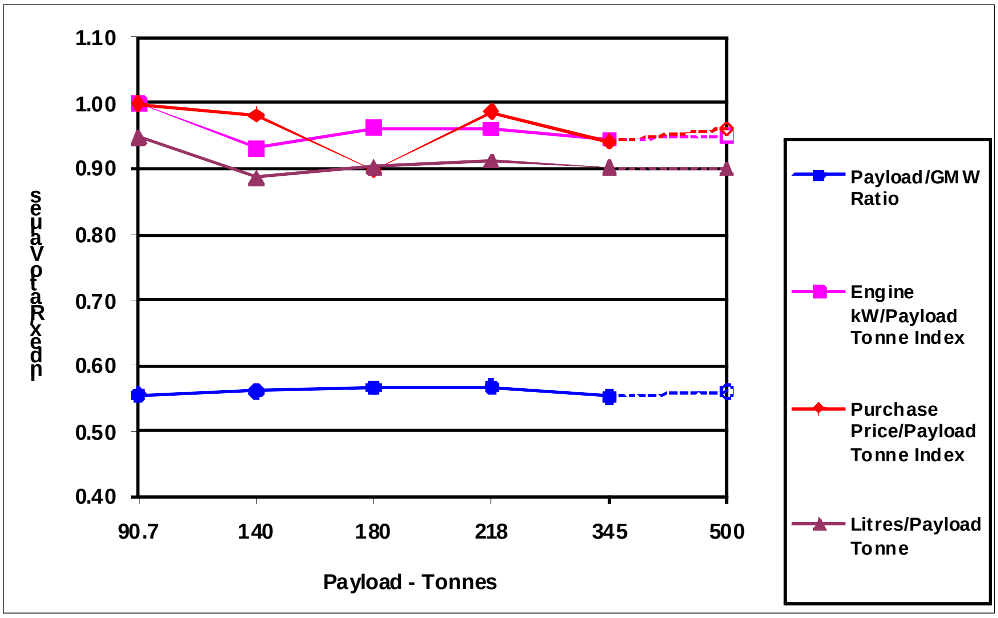


Figure 4.11 Comparative Indices – Caterpillar Mining Trucks plus Hypothetical 500 Tonne Truck (From Table 4.7)

Figure 4.12 compares the four selected characteristics that are sensitive to nominal target payload in terms of indices. The curved shape of the trend lines in Figure 4.12 could be misleading. If the Payload - Tonnes axis (abscissa) is drawn to scale of payload tonnes the trend lines become practically straight lines.

The predictability of these four characteristics; and further inferences that can be based in them, provides the basis for projecting characteristics of a 500 tonne payload truck conforming to generic design criteria as follows:

Nominal Target Payload	500 tonnes
GMW	890 tonnes
NMW	390 tonnes
Gross Engine Power	3700 kW
Indicated Purchase Price	Au$9.5M (2006 Au$1.00 – US$0.73)

Performance	Similar to 793 D and 797B – depending on transmission range configuration.
Tyres – Maximum Static Load	150 tonnes (6 wheels)
TKPH	Est. 2,500 (Tonne-Kilometre-Per-Hour)
Fuel Burn	450 litres/hour (deep pit, severe service)

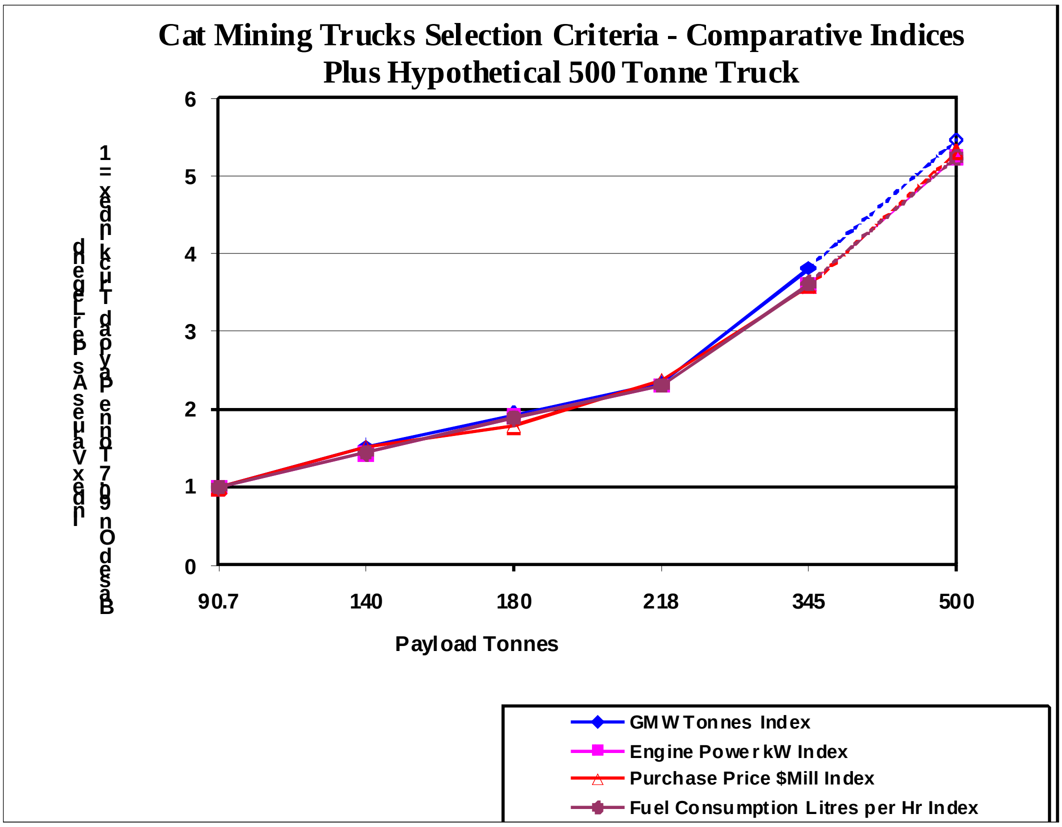


Figure 4.12 Cat Mining Trucks Selection Criteria – Comparative Indices plus Hypothetical 500 Tonne Truck (From Table 4.7)

The above hypothetical discussion is based on Caterpillar mining trucks. Similar analysis and extrapolations applied to ranges of mining trucks offered by other OEM could be expected to yield similar results. It was pointed out earlier in this thesis that the differences in performance and cost for trucks from all manufacturers tend to be more subtle than startling.

Obvious current technical impediments to the above hypothetical concept include:

- Tyres: - development of 63-inch-rim tyres for UC trucks is still continuing. Adding the next payload advance would convolute tyre-performance development and supply problems that currently are still being resolved. Using 8 wheels with castor steering (existing technology) and possibly four-wheel drive (AC electric drive) could be an option.

- Limited availability of high-speed – 1800 RPM – engines for mobile plant in the required power range.
- Non-availability of mechanical transmissions; but AC traction systems from electric powered railways could likely meet the power specification and be adapted – similar to historical development of DC electric wheel motors for mining trucks.
- Extrapolation of structural and mechanical designs would likely be challenging – just as it has been for OEM in developing large through to UC trucks – especially for drive wheel groups and truck bodies – to retain generically similar performance and acceptable productivity.

From an applied research viewpoint the above discussion and predictions for a 500 tonne payload truck work are hypothetical - really “crystal-ball gazing”. But the comparative tabulation and illustrations do reinforce, for trucks, that variable cost components for hauling and loading equipment, when compared in terms of unit costs, i.e., independent of truck payload or bucket load can be expected to be predictable and practically constant.

4.2.6 Trolley Assist

Historically justification of adoption of trolley-assist mining-haulage systems needed a significant gap between reconciled costs of diesel fuel and mains electric power for haulage. Two recent events have closed the gap:

1. Development of more efficient AC power wheel motors in large and UC mining trucks with retro-application to smaller capacity mining trucks.
2. Practically doubling of nett-of-tax diesel fuel price to large consumers over a period of some four years to 2006, a continuing escalating trend.

Recent feasibility studies for deep open pits of medium to large production scale have indicated that trolley-assist haulage is potentially cost beneficial.

It seems likely that future improvements to AC electric wheel drive systems have potential to favour trolley assist, not only for enhanced up-ramp performance in deep open pits, but also to reduce consumption of expensive diesel fuel.

Historically, inclusion of trolley-assist of DC drive trucks as an option in feasibility studies for deep open pits generally revealed unacceptable risk in operation due to rejects (overloaded trucks). A third lane on deep-pit ramps was required for rejected loaded trucks travelling at lower speed. Cost of extra pit volume, together with capitalization and on-going costs of the electrical reticulation system and trolley equipment onboard trucks, at that time more-than-offset the apparent cost benefits from increased productivity of trucks. Recent developments, including improved power curve for AC drives, anecdotal evidence of improved reliability, improved understanding of the need for controlled payload dispersion – to reduce over loading and rejects - have elevated trolley assist to “potentially cost-beneficial”.

In the economic climate prevailing in the Australian mineral industry, trolley assist has potential to extend mine life and recovered ore or product reserves for deep open pits of the future. Certainly trolley assist should be an option carried forward in feasibility studies until eliminated by economic and/or practical considerations.

4.2.7 Dispatch Systems and Autonomous Load and Haul Equipment

4.2.7.1 Dispatch System Investment

Investigation of benefits from a site-specific dynamically-programmed dispatch system (DS) should be integral with load-and-haul equipment selection. Benefits from DS for moderate to large scale open pit mines generally justify investment. Cost-benefit from efficient operation of truck haulage is an imperative as scale of operations and equipment increases – Section 4.2.2. Initially located within sight of open pit operations, DS have relocated to increasingly remote locations. Current practice includes off-site locations in cities, in corporate offices or special purpose facilities, centralizing dispatch control of a number of mines. It is technically possible for dispatch staff to work from home linked by video and teleconferencing to a DS; and to all relevant management links.

4.2.7.2 Autonomous Mining Equipment

Remote dispatch control of load and haul equipment already exists. Underground mining applications have adopted remote control by operators of mobile equipment as a safety initiative. Development from remote control of equipment to autonomous load and haul applications has been rapid with planning studies currently in progress for completely autonomic systems of mining trucks, loading and support equipment. Recent developments and trials of autonomous load-and-haul equipment, particularly trucks are obviously socio-industrially and politically sensitive. Details of proposed autonomous open pit systems were not available to the author. With performance and cost information generally limited to hearsay at time of completion of the thesis, this topic has not been researched. Further comments are provided in Section 6.1.6 and 8.2.

CHAPTER 5

LOAD AND HAUL COSTS

5.1 Basis of Cost Analyses and Discussion

5.1.1 Costs - A Snapshot In Time

Load and haul costs are the outcome of market-driven cost of goods and services that are time-variable inputs to the cumulative cost of operating mining equipment. Cost of operating loading equipment and mining trucks is time dependent. Unit load-and-haul-cost outcomes are “a snapshot in time”. Cost of equipment items per operating hour is dependent on both variable cost of goods and services and the, variable operating duty of the mining equipment. For mining trucks and matching loading equipment, operating duty is a function of open pit (or generally mine) geometry, geographic location, meteorological operating environment and other factors including industrial, human resources and socio-economic environments all of which vary with time to some degree. Analysis of productivity and costs in this Chapter 5 are in terms of indices to facilitate comparisons, the time-dependence of which are minimized.

5.1.2 Relative Importance of Cost Estimates

Experience with developing estimating costs for feasibility studies over the wide range of precision and confidence levels; also for operating budget purposes, provides understanding of the difficulty of comparing estimated costs for different mining operations. Some degree of reconciliation is possible but the ultimate test for mining operations is not on unit costs of operational functions or activities but how the operation stacks up in the commodity market place. It is somewhat redundant to compare estimated costs in detail for a project where the indicated total production cost of a commodity falls in the lower quartile of producers worldwide. If the feasibility study does not support development, there must be other impediments such as poor business address with substantial sovereign risk or funding difficulties. Drilling down to equipment selection criteria and focusing on cost comparisons, it is the author’s experience that:

- Comparison of operating costs between mining operations involves complex reconciliation with an outcome that always carries significant residual risk unless the purpose and use that costs are transported between projects is tolerant of substantial error between the estimates utilized and the ultimate feasibility estimates or actual costs experienced.

- Differences in estimated or actual operating costs between items of loading or hauling equipment with reasonably comparable performance, all other cost influences being small, tend to be subtle rather than startling – as indicated earlier in the thesis. It is conclusion that, for realization of expected productivity and operating costs, non-pecuniary attributes such as reliability and product support from suppliers need to be afforded substantial weight.

The above understandings lead to the conclusion that comparisons of cost estimates for equipment selection purposes should be treated circumspectly, given limited weight and should not be the sole basis for decisions. The obvious limitations of cost estimates lead to the further conclusion that analysis is most conveniently served by expressing and comparing costs in terms of comparative indices that are less time dependent.

5.1.3 Basis of Cost Indices

For the purposes of discussions herein load and haul costs will be assumed to be typically 50% of total mining costs as shown in Figure 2.2. Bruce Gregory established the cost proportions in Figure 2.2 in 2002 (Gregory, 2002). Recently there has been substantial volatility in fuel and tyre prices with significant increases. Fuel-driven cost increases flow on to all manufactured goods, especially tyres. A recent worldwide shortage of large earthmover tyres, which industry sources advise is expected to last for several years, has elevated prices. Haul cost proportions developed by Bruce Gregory (Gregory, 2002) have been adopted as a suitable basis to factor up for comparative analysis.

All analysis to follow is in terms of cost indices. In order for the indices developed and adopted for analysis to be reasonably indicative of current costs, Gregory’s 2002 haul cost proportions have been adjusted. Table 5.1 lists the original data and compares original and adjusted proportions. Figures 5.1, 5.2, and 5.3, illustrate successive adjustments to haul-cost proportions for increased cost of fuel and tyres.

Table 5.1 - Haulage Cost Proportions

Cost Item	Typical 2002 (Gregory B, 2002)	Adjusted for Fuel 35 to 55 cents/litre	Further Adjusted Tyres +50%
	%	%	%
Ownership	**26**	**23**	**22**
Fuel	**24**	**33**	**31**
Maintenance Parts	**19**	**17**	**16**
Operator Wages	**14**	**12**	**12**
Tyres	**13**	**11**	**16**
Maintenance Wages	**3**	**3**	**2**
Lubrication	**1**	**1**	**1**
Total	**100**	**100**	**100**

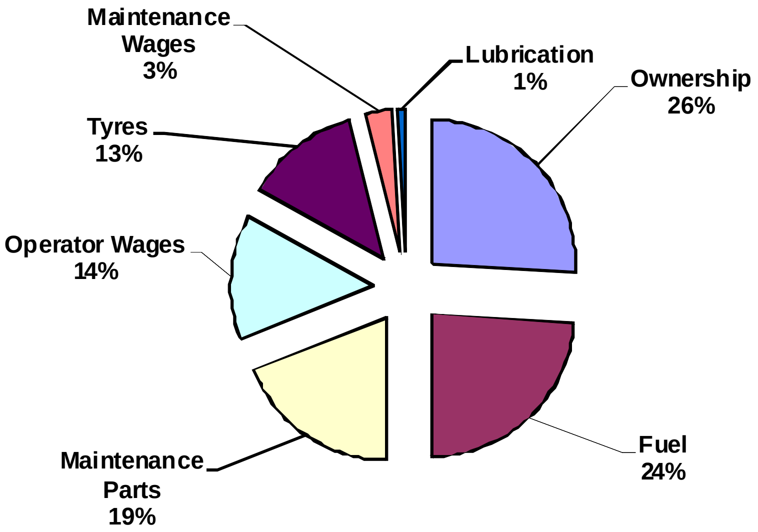


Figure 5.1 Haulage Cost Proportions (Gregory#1, 2002)

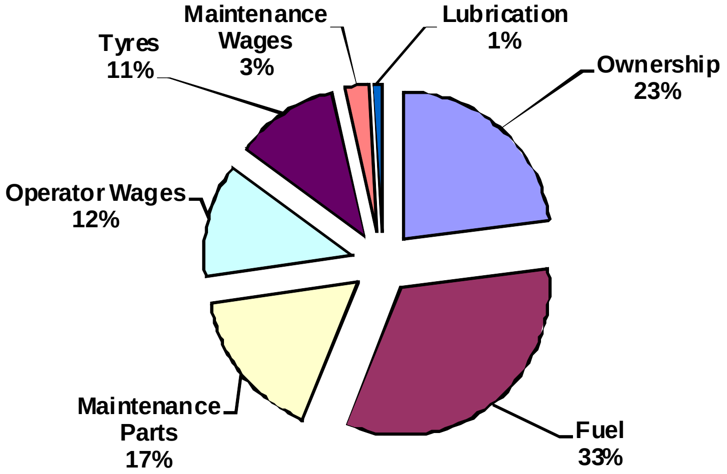


Figure 5.2 Haulage Cost Proportions –
Adjusted for Fuel 35 to 55 cents/litre - from Table 5.1

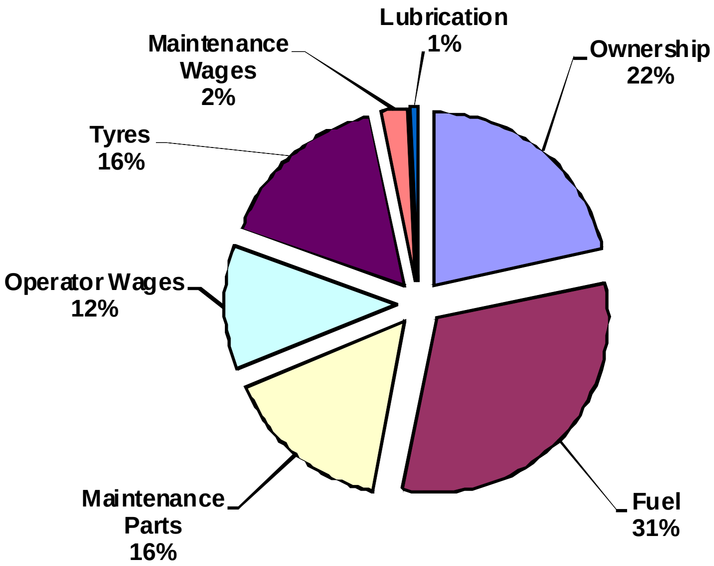


Figure 5.3 Haulage Cost Proportions

Adjusted for +50% Tyre Cost - from Table 5.1

Results of analysis based on the above haulage cost proportions an acceptable basis for comparing haulage costs in terms of indices.

5.2 Cost Indices and Proportions of Haul and Total Mining Costs

5.2.1 Fixed and Variable Costs

In the context of loading and hauling cost indices as adopted herein, the terms "fixed" and "variable" mean:

- Fixed costs - those cost proportions of hourly operating or unit production costs that are not dependent on the quantity of production – including operating labour, service and maintenance labour on site for fuelling, daily checks and top-ups and for removal and replacement of components. Fixed costs also include finance costs, interest on capital cost of equipment and insurances.

- Variable costs - those cost proportions that are dependent on hours productively operated or actual production achieved – by definition all other costs listed in Section 4.2.2 that are not fixed including fuel, lubricants and related consumables, maintenance both preventative and breakdown, component rebuilds and tyres. Capital redemption by way of an amortization charge is considered as a variable cost for the purpose of comparing load-and-haul costs.

The view taken herein of capital recovery costs for mining equipment is that capital is redeemed as a charge per hour operated or per tonne produced as the case may be. Recovery is over the assumed life of the equipment item with due allowance for residual value, if any. Due deduction is made from the capital cost for consumable items that are generally delivered with new equipment such as tyres on trucks and ground engaging tools (GET) on shovels.

5.2.2 Capital Redemption

There is a gray area with deductions from capital costs for tyres on new trucks and GET on shovels to established initial value for write-off. The line between capitalized equipment cost and consumable cost is arbitrary. A clinical definition of what constitutes capital value of an item of equipment could be based on whether the component subject of a capital reduction has less life than assumed for capital amortization – write-off – of the equipment item. For engines, transmissions, other driveline componentry, lightweight truck bodies, shovel buckets, power transmission componentry, hydraulic system components and electrical componentry that all have a life-to-component-replacement, which is expensed at the time of replacement, there could be a case for commensurate deductions for these components from initial book value.

In short, and hypothetically, for the purposes of write-off, trucks and loading equipment might be virtually stripped down to the frame. Basic componentry expected to survive the full write-off period (conceptual equipment life) would be depreciated according to tax statutes. All other components could be expensed over expected individual component life, i.e., recovered by cost-provision accounting or directly expensed as consumables. The effect would be accelerated write-off and increased operating margin retention in the first one or two production years of a project. A side benefit would be a closer correlation between market value (MV) and written down value (WDV) – avoiding the potential need to bring a WDV loss to account at equipment disposal.

Valid tax treatment of equipment write-off and acceptable accounting practice has not been considered in the above discussion. Obviously, the direction of any further research in respect of capital recovery requires consideration of legality and corporate benefit of any accelerated (or deferred) asset write-off that might result.

5.2.3 Capital Recovery Risk

The following notes are from a paper published by the author in 2005 updating an earlier paper on outsourcing or owner operating of mining operations (Hardy#3, 2005).

Owner operators and contract miners treat ownership cost of mining equipment differently. A contractor writes off straight line, over an estimated life in hours, the purchase price of equipment less residual value (and less any consumable deductions such as tyres) based on expected market value (if any) at end of write-off period. Theoretically, the point is reached where the written down value (WDV) equals the residual value (RD), which in turn (hopefully) is not more than market value (MV).

For all mobile equipment, throughout its life MV is accepted as less than straight-line WDV by a margin ranging from small to, say, 30% of original purchase value.

Caterpillar equipment appears to enjoy favourable resale value so a tendency for reduced WDV – MV margins over machine life. Readily saleable equipment such as off-highway trucks, graders, small loaders and excavators and sundry small mobile plant generally realize minimum sale or trade-in value in the order of 15 %. Other items including large excavators and loaders tend to realize lower values, (*Perich, 2003*).

So there is a risk for mine operators that, before reaching expected life, there could be a gap between WDV and MV. Usually offsetting this risk to a significant degree for mining contractors is the benefit of provision for component replacement and associated maintenance and repair that accumulates because of an average hourly maintenance cost for equipment items built into the scheduled prices. Provision accounting has a two-fold benefit. It is a source of accumulated funds to be drawn down when required for major planned maintenance, such as component rebuild/replacement. Although not so intended, a provision account may also offset any gap between MV and WDV in the event of premature disposal.

Owners generally write off at the allowable rate for tax purposes; and account for depreciation accordingly. This avoids any accumulation (or over drawing) in the tax equalization account that has to be brought to account as a profit (or loss) at some future time. Tax write-off allowable for equipment has recently been amended to the lesser of the life of the project or the term listed below:

Write offs under current tax rules: (available by search for TR 2000/18 C4 on the Australian Tax Office website).

Equipment Description:	**Life: Years**
Dozers	9
Electric Rope Shovels	25
Graders	10
Hydraulic Excavators	10
Lighting Plant	10
Off-highway Trucks	10
Scrapers	7
Wheel Loaders	8

In continuous mining operations (nominally 24 hours per day every day) main production equipment accumulates between 5,000 to 6,500 productive operating hours and up to 7,200 service meter units (SMU) per year, with larger equipment generally reporting at the higher end of the range. Except for large electric shovels, and to lesser degree hydraulic shovels, contractors will rarely estimate write-off life of any item of even large equipment at more than 50,000 hours. This corresponds to a maximum life of some eight years. Owner miners tend to write off at lower rates than a contractor. Any difference between WDV and MV at time of sale or trade can be carried forward by owners as a loss for tax purposes. But the net-of-tax loss to the equipment owner will report in the year it occurs. Therefore there is potential for owner miners to be reporting lower cash costs, and lower total costs with the certainty that, at some future time, any accumulated depreciation cost loss will be brought to account by a discrete loss and consequent reduction to future profit.

Figure 5.4 generally illustrates the analysis of capital-recovery risk. A reduced rate of capital recovery is indicated in Figure 5.4 for owners to reflect the different treatment of depreciation cost of assets. Owner miners will likely have a greater gap between WDV and MV over much of the life of each item of equipment.

Owner miners must take the risk of any gap between MV and WDV in the event of a premature shutdown of mining operations. In the balance, such risk for an owner (in control of future continuation of mining operations) is likely to be accepted without need for special treatment. In contrast, contractors are unlikely to accept the same level of risk. Owner miners need to identify and manage risk to future profits by not fully amortizing equipment before sale or trade in the normal course of operations.

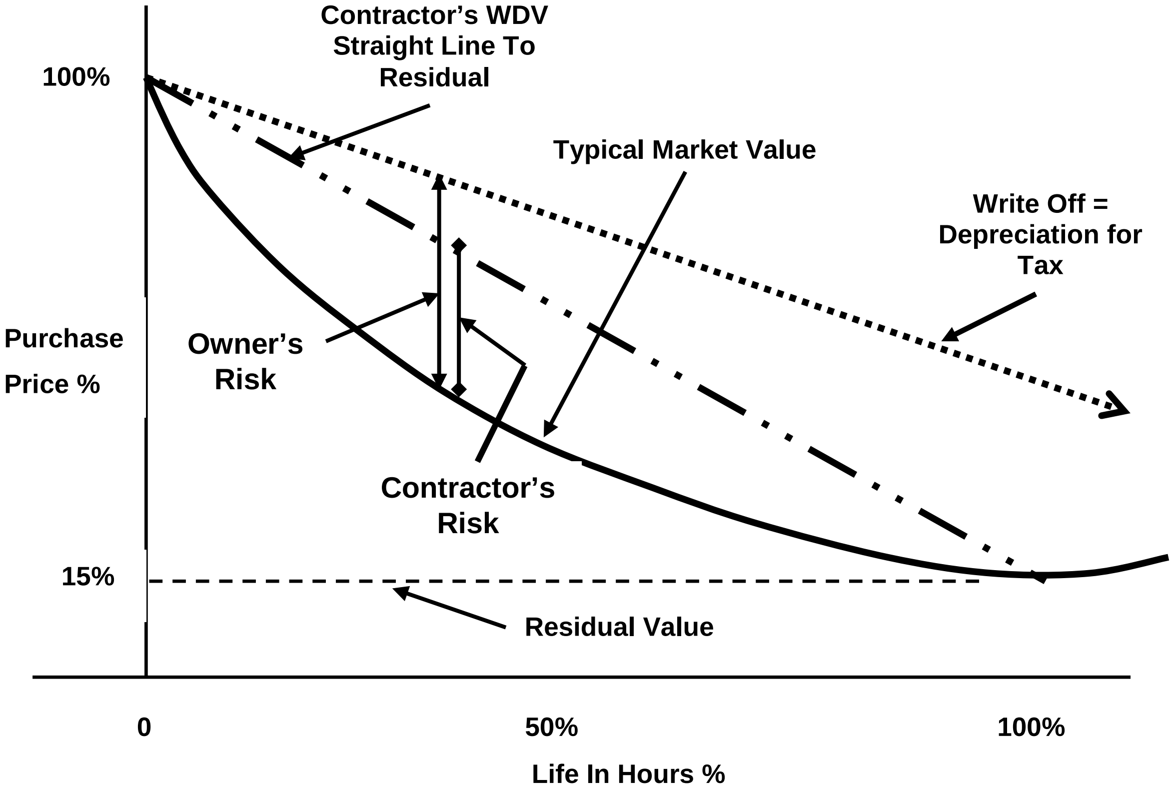


Figure 5.4 Capital Recovery Risk

Figure 5.4 shows that a mining contractor carries a significant capital-recovery risk early in the life of a contract – as does an owner. But as mining contracts generally provide for termination at the convenience of the owner, a contractor appears to be more exposed to significant capital-recovery risk than does the owner.

Traditionally, in "hard-money" schedule-of-rates mining contracts, unit prices have been averaged over life of the contract. This provides an accumulating maintenance provision to be drawn down in the latter stages of the contract – as illustrated by Figure 5.5.

From Figure 5.5 maintenance provision is accumulated at high rates early in the contract tending to offset the capital-recovery risk. Theoretically, the MV reflects the current intrinsic value that is less than the WDV by the decreased value of the machine componentry at that stage of machine life. The decreased componentry value is covered by the maintenance provision.

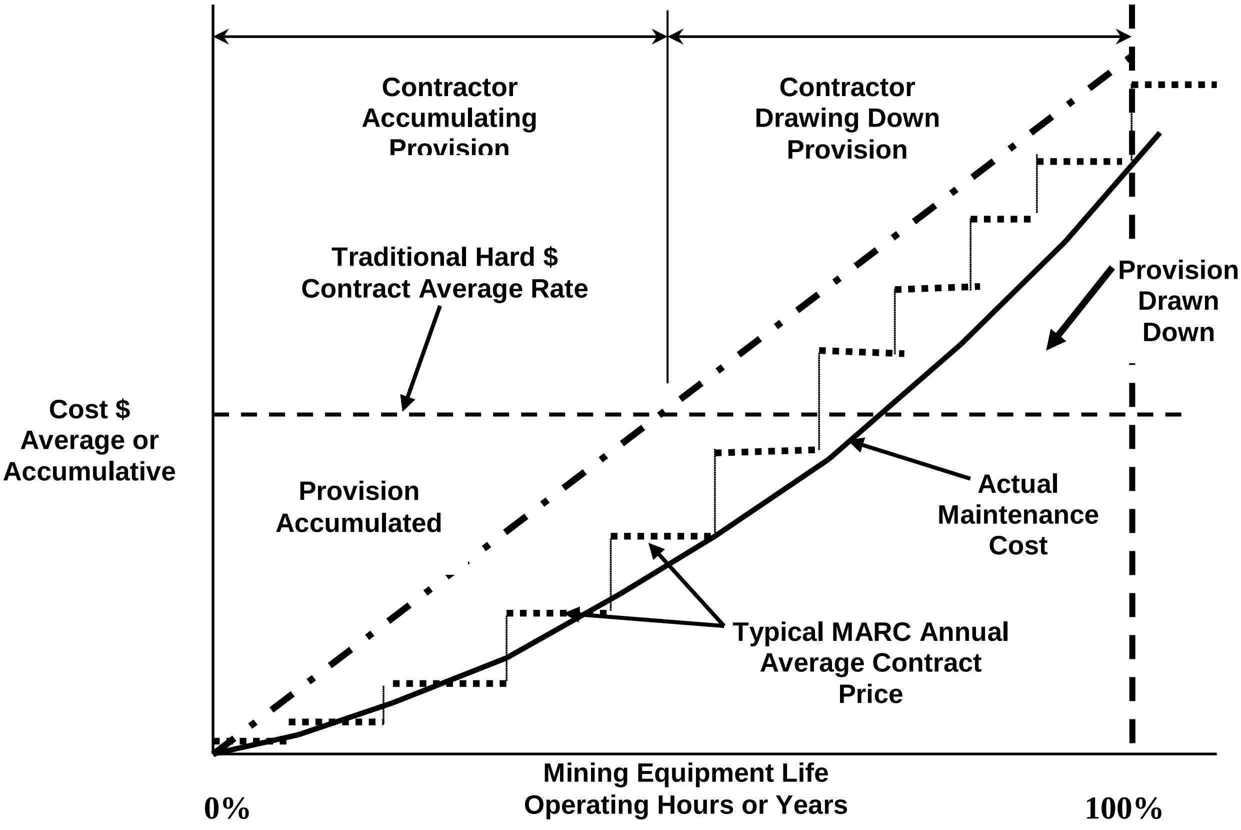


Figure 5.5 Mining Equipment Maintenance Cost (Hardy#2, 2003)

Purchase Capital = WDV + Amortization (Depreciation)

MV + Maintenance Provision = WDV

The above equations describe a perfect world where:

MV = Purchase Capital - Amortization – Maintenance Provision

This balance will rarely, if ever, be experienced. Premature failure of componentry that accelerates draw down of provision, or the absence of provision accounting, exceeding maintenance budgets, will upset the cost balance implied by the above equations.

Most mining operations, especially owner miners, continue on in blissful ignorance of any gap between MV and WDV until the moment of truth at trade-in or disposal when a discrete loss write-off has to be accepted by owner or contractor. New, or replacement-equipment purchase negotiations occasionally take an interesting direction. The outcome of inadequate asset-value accounting is introduced as a legitimate argument for price-reduction consideration by potential customers of

OEM and dealers. Any gap between WDV and MV tends to be tabled by owners or contractors during purchase negotiations. It is great credit to the parties involved in new equipment deals that such invalid arguments are generally absorbed with aplomb; eventually compromise is reached and a deal is done to the mutual satisfaction of the parties.

Figure 5.5 illustrates that, any style of contract where remuneration terms reflect actual maintenance costs incurred rather than average (including provision) costs, effectively denies a mining contractor the *de facto* cover for capital-recovery risk in the event of premature determination of the contract. Where a contractor establishes maintenance provisions internally, it is generally viewed as a debt to an equipment item. From this viewpoint, it is artificial to consider the same funding as cover for capital recovery risk. But at the time of contract determination and equipment disposal the MV will tend to reflect "as-is, where-is" value and any provision is then available to cover the WDV - MV gap. If the contractor transfers the equipment to alternative work, the provision covers future maintenance liability as intended. If, through the remuneration process, a mining contractor does not receive maintenance provision compensation then the capital-recovery exposure at contract termination-of-convenience is a significant risk, likely more so than for an owner.

Nomination of a maintenance sub-contractor, novation or voluntary execution of a MARC maintenance facility by a mining contractor has similar effect. MARC maintenance facilities generally average prices annually with costs of component rebuilds being included in cost estimates at pre-determined (sometimes warranted) times. In effect, MARC pricing compiled cumulatively tends to parallel actual cumulative costs. Figure 5.5 depicts cumulative trends. The step representation for a MARC contract cost is a mix of constant annual charges with annual adjustments to restore the cumulative MARC cost. In actual practice, practically constant monthly payment for MARC services in each year should, more correctly, be depicted to accumulate as a sequence of inclined trend lines.

5.2.4 Operating Cost Indices

Cost indices for haul operating states of production, waiting, standby without operator and for extra operating time (improved efficiency within scheduled shift hours) are listed in Table 5.2. These indices were derived from the proportions in Figure 5.3.

Table 5.2 - Cost Indices - Mining Trucks

Proportions	Operating %	Waiting %	Extra Operating %
Ownership	22	2	22
Fuel	31	1.5	31
Lubrication	1		1
Maintenance	16	0.8	16
Tyres	16		16
Service Labour	2	2	
Operating Labour	12	12	
TOTALS	100	18.3	86
Indices	1.00	0.183	0.86

Cost indices for loading equipment, projected for both hydraulic shovels and rope shovels, are included in Table 5.3.

Analysis and discussion in Section 5.3 and following sections are based on the indices in Tables 5.2 and 5.3.

Table 5.3 - Cost Indices – Loading Equipment

Proportions	Operating %		Waiting %		Extra Operating %	
	Hydraulic	**Rope**	**Hydraulic**	**Rope**	**Hydraulic**	**Rope**
Ownership	22	30	2	3	22	30
Fuel/Power	22	15	2	1.5	22	15
Lubrication	-	-			-	-
Maintenance	40	44	2	2	40	44
Tracks/Undercarriage	-	-				
Service Labour	10	6	10	6		
Operating Labour	6	5	6	5		
TOTALS	100	100	22	17.5	84	89
Indices	1.00	1.00	0.22	0.175	0.84	0.89

5.2.5 Fuel Consumption and Cost Indices

Outputs from FPC analysis of the of the Caterpillar Mining truck range for three hypothetical courses for open pits 50m, 250m and 500m deep were discussed in Section 3.3.10. Table 3.64 compares indices based on truck performance measured by total haul cycle time for a payload range varying between 80% to 120% of target mean payload for the three largest Caterpillar trucks (180, 218 and 345 tonnes nominal payload).

Table 3.65 provides a similar comparison of indices of comparative productivity measured in terms of the productivity payload index and performance index. This provides a productivity index relative to mean performance/mean payload combined index set at 1.00. Figures 3.62, 3.63, and 3.64, compare sets of performance and productivity indices over a payload range of ± 20% for the three haul courses analyzed with Caterpillar's FPC application.

Similar analysis was performed on fuel consumption results from FPC for the three courses for each of the three largest Caterpillar mining trucks. This analysis provided a set of comparable fuel consumption and cost indices (fuel consumption and fuel cost index values will be identical as fuel consumption and fuel cost are related by a constant fuel price).

Table 5.4 Fuel Consumption/Cost Indices

	Truck Payload %								
Potential Truck Trip Times	**80**	**85**	**90**	**95**	**100**	**105**	**110**	**115**	**120**
Course 1									
Fleet 4 Cat 789C	0.896	0.915	0.931	0.953	1.000	1.019	1.038	1.060	1.076
Fleet 5 Cat 793C	0.911	0.930	0.951	0.970	1.000	1.022	1.062	1.079	1.098
Fleet 6 Cat 797B	0.925	0.946	0.963	0.981	1.000	1.023	1.042	1.080	1.103
Course 2									
Fleet 4 Cat 789C	0.878	0.896	0.912	0.936	1.000	1.022	1.043	1.064	1.081
Fleet 5 Cat 793C	0.895	0.917	0.947	0.965	1.000	1.026	1.078	1.096	1.114
Fleet 6 Cat 797B	0.912	0.932	0.955	0.981	1.000	1.030	1.050	1.101	1.129
Course 3									
Fleet 4 Cat 789C	0.874	0.891	0.906	0.931	1.000	1.022	1.043	1.064	1.082
Fleet 5 Cat 793C	0.892	0.914	0.944	0.963	1.000	1.025	1.082	1.100	1.117
Fleet 6 Cat 797B	0.909	0.932	0.955	0.982	1.000	1.028	1.051	1.106	1.131
Across-model Means									
Course 1	0.910	0.930	0.948	0.968	1.000	1.021	1.047	1.073	1.092
Course 2	0.895	0.915	0.938	0.961	1.000	1.026	1.057	1.087	1.108
Course 3	0.892	0.912	0.935	0.958	1.000	1.025	1.059	1.090	1.110

Table 5.4 summarizes the results. Review of the data in Table 5.4 reveals similar index trends over the three-truck payload range. To simplify illustration, fuel consumption/cost indices were averaged across the three-truck range of 180, 218 and 345 tonne nominal payloads – termed "Across-model Means" in Table 5.4.

Across-model mean fuel consumption/cost indices are illustrated by Figure 5.6.

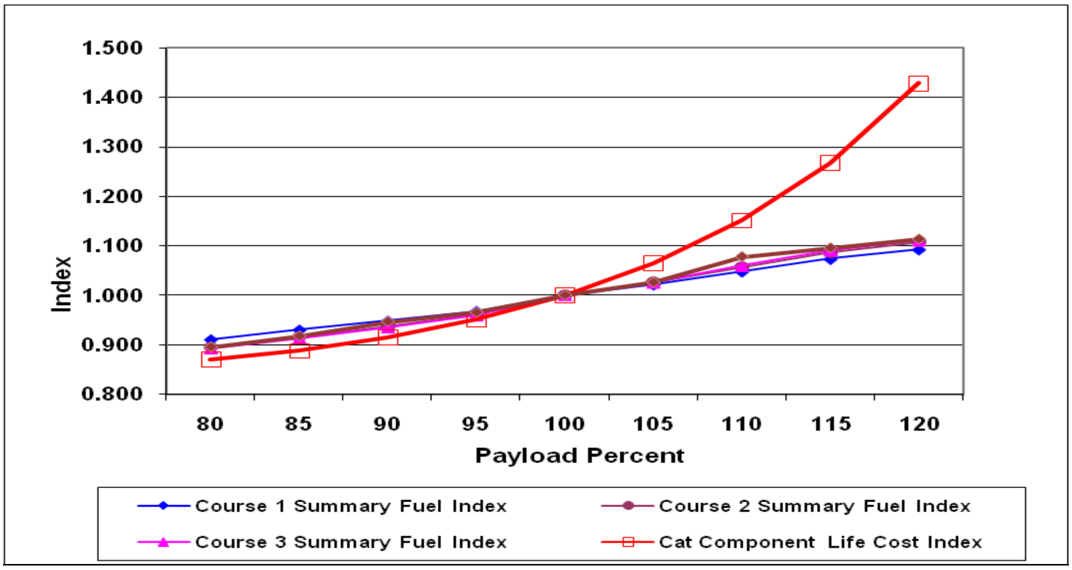


Figure 5.6 Fuel/Component Cost Index Comparisons – Payload 80% to 120%
From Table 5.4

The relationship between driveline component-life cost and payload variability is discussed in some detail in Section 5.3. A Component Life Cost Index, developed by analysis in Section 5.3, is also shown in Figure 5.6. There is obvious lack of correlation between component life index and fuel index trends, which are direct derivations of fuel burn results from FPC and the component life index. This is inconsistent with Caterpillar advice that the component life index is related to fuel burn. This issue is discussed further in Section 5.3.

5.3 Costs – Truck Overloading and Payload Distribution Effects

5.3.1 Introduction

Effects of payload overload and excessive dispersion of payload distributions were analyzed and discussed in Sections 3.2.8 and 3.3.9. Driveline-component-life penalties were indicated for truck overloading; also for excessive payload dispersion.

Consequences from overloading Caterpillar mining trucks over a range of +/- 20% from target-mean payload were analyzed in Section 3.3.10 using Caterpillar's FPC simulation software. Results are summarized in Table 3.64, for truck performance indices; and in Table 3.65, for truck productivity indices.

It was implied in Section 3.3.10 that apparent nett productivity gains from additional payload is partially offset by reduced truck performance, i.e., increased truck travel time T_V. But, it was also implied that apparent productivity gains from over loading are even further offset by additional costs. The apparent nett productivity gain at normal operating costs - direct operating costs per unit time reduced to unit costs per production unit (tonne or BCM) - attracts further potential cost penalties from:

- Reduced driveline component life due to overload.
- Nett dispersion effect of payload distribution in excess of permissible standards such as Caterpillar's 10/10/20 policy.

These potential cost penalties are analyzed and discussed below.

5.3.2 Component Life and Payload Relationship

Figure 5.7 illustrates Caterpillar's 10/10/20 Policy (published in April 2002). Concurrently Caterpillar updated previously-issued information on the relationship between component life and truck payload interpreted from empirical data.

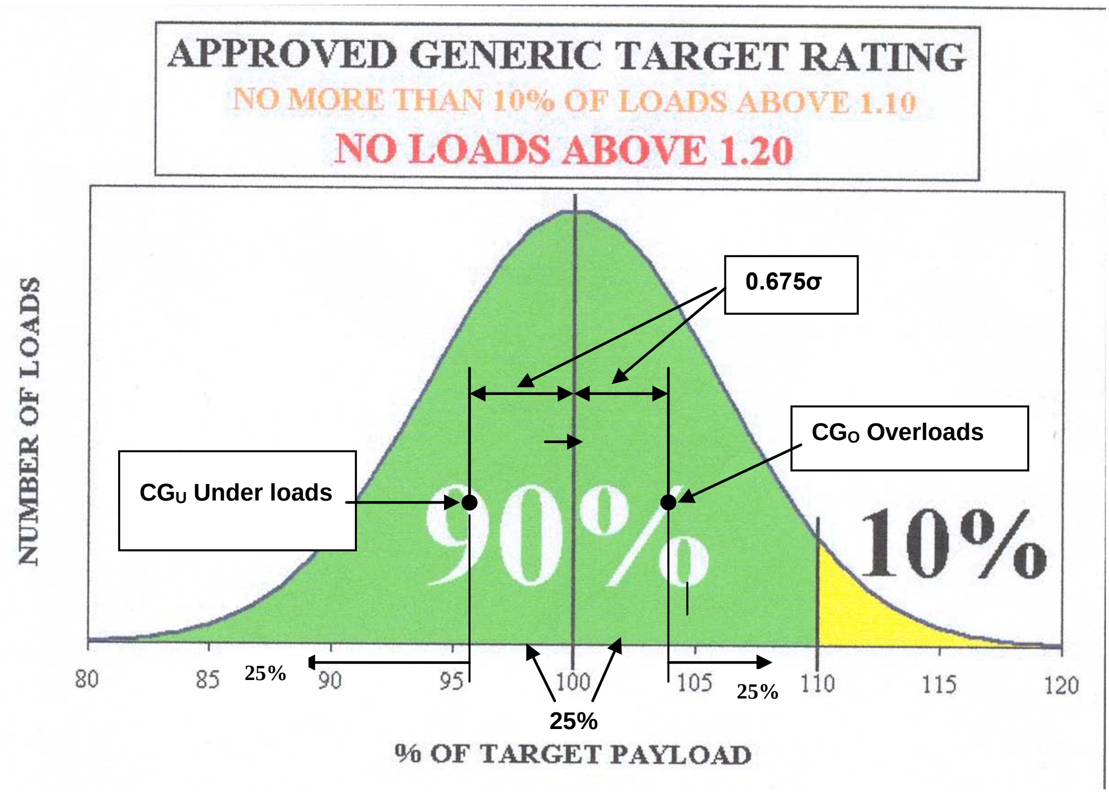


Figure 5.7 Approved Generic Target Payload Rating –
(Caterpillar Inc. 10/10/20 Policy, April 2002)

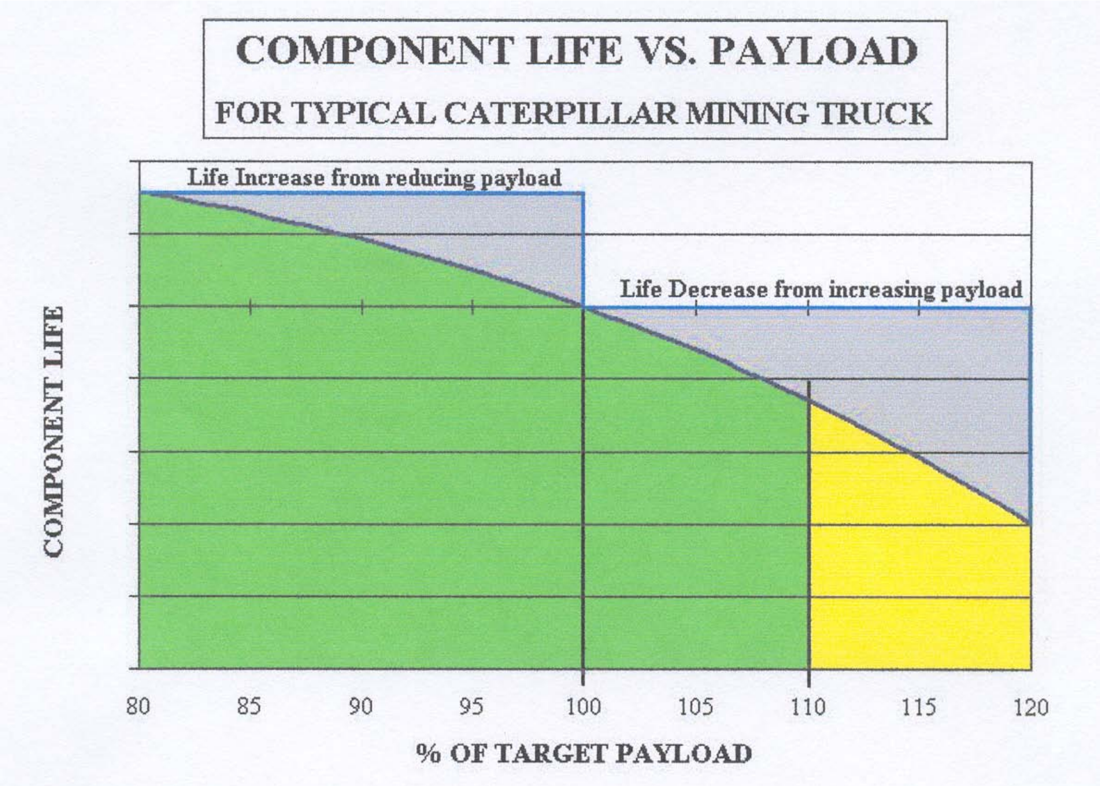


Figure 5.8 Component Life Versus Payload –
(Caterpillar Inc.10/10/20 Policy, April 2002)

Figure 8 illustrates the Component Life v. Payload relationship indicating that, for selected degrees of overload, component life is reduced more than additional component life gained by a similar degree of under loading.

Empirical fuel burn provides a means for explaining the component life vs. payload relationship, especially engine life. Component life for transmissions and "lower power train" is based on torque and speed – i.e., power transmitted - that can be directly related to fuel burn via BSFC. Mining trucks offered by OEM other than Caterpillar respond similarly for equivalent operating conditions. Although the current analysis is based on Caterpillar trucks, software and data, outcomes and interpretations can be applied generically to mining trucks from other OEM.

5.3.3 Haul Cost Indices for Payload Distribution Range

The component life v. payload trend line in Figure 5.8 implies a non-linear continuous functional relationship. A scale for component life is not provided. As a trial for purposes of analysis and discussion it was inferred that:

- 80% of target payload corresponds to 115% component life.
- 120% of target payload corresponds to 70 % of target life.

With the obvious, trivial case of 100% of target payload corresponding to 100% component life, the three data sets enable solution of a quadratic equation as a basis for interpolation of component life at any selected percentage of target payload. (A quadratic equation was chosen as sufficiently precise. A polynomial with higher power terms was considered but discarded as an unnecessary complication.

The generic form of the implied quadratic equation is shown as equation (G) below:

$$CL = a \bullet PL^2 + b \bullet PL + c \qquad (G)$$

CL = Component Life %

PL = Payload % of Target

a, b, c are constants.

Solving for a, b and c yielded

$$CL = -0.01875 \bullet PL^2 + 2.62497 \bullet PL + 25.00128 \qquad (1)$$

Table 5.5 - Haul Cost Indices For Payload Distribution Range

Payload %	70	75	80	85	90	95	100	105	110	115	120	125	130
Component Life %	116.88	116.41	115.00	112.66	109.38	105.16	100.00	93.91	86.88	78.91	70.00	60.16	49.38
Component Life Index	1.169	1.164	1.150	1.127	1.094	1.052	1.000	0.939	0.869	0.789	0.700	0.602	0.494
Component Cost Index	0.856	0.859	0.870	0.888	0.914	0.951	1.000	1.065	1.151	1.267	1.429	1.662	2.025
Haul Cost Index	0.908	0.910	0.917	0.928	0.945	0.969	1.000	1.042	1.097	1.171	1.274	1.424	1.656

Percentages for component life in Table 5.5 were derived from equation (1). Also shown in Table 5.5 are:

- Component Life Indices (100% reduced to 1.00)
- Component Haul Cost Indices (100% reduced to 1.00)

Haul Cost indices were derived by applying to the proportions for fuel, lubrication, maintenance and tyres (totaling 64%) from Table 5.2, the component cost indices in Table 5.5. The calculation used was 0.36 + 0.64 x component cost index for each payload percentage in Table 5.5. Figure 5.9 - data from Table 5.5 - illustrates index trends for component life; component cost and component-life haul cost.

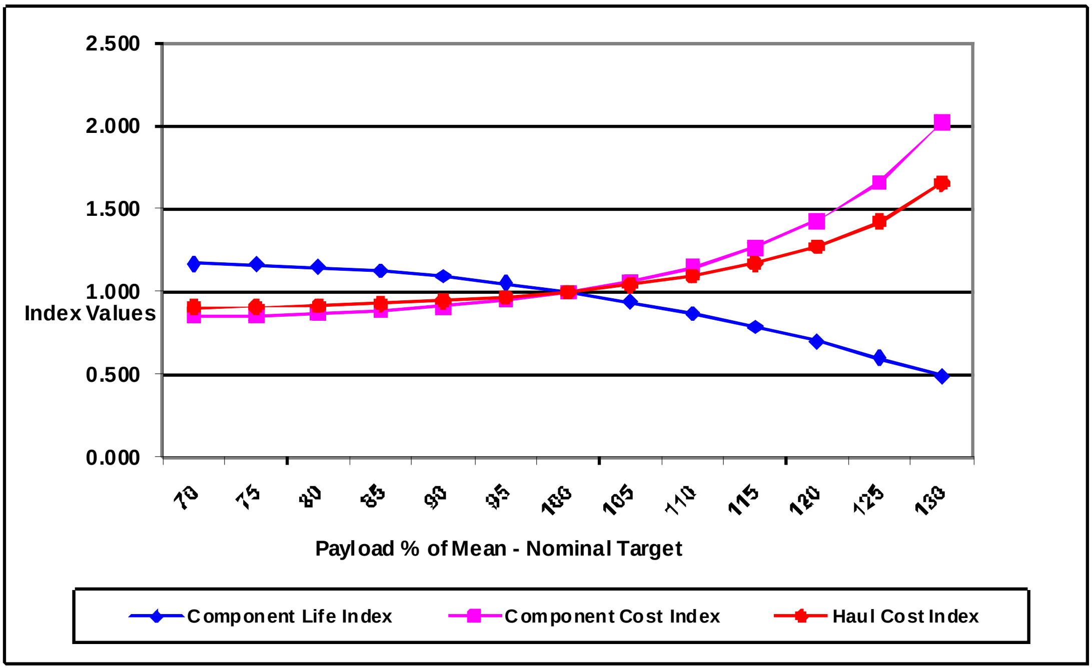


Figure 5.9 Component Life, Cost and Haul Indices v. Payload Distribution
From Table 5.5

The above analysis is based on:

- Component life values inferred from Figure 5.8.
- Component-life haul-cost indices include fuel, lubrication and tyres in the group of affected cost components.

In review of the second of the above points, it is reasonable to expect an increase or decrease in these identified cost components as payload varies from target mean. But variation of these cost components may differ from the component life v. payload relationship. That is, the variation in fuel, lubrication and tyres may be more or less than that of driveline component life. The known relationship between fuel burn and engine life, intuitively, can be extended to driveline life. Also, tyre life is significantly reduced by increased truck payload as discussed in Section 3.3.8. So, it is considered reasonable, for the purposes of analysis and discussion, to proceed on the basis of the initial inference and the assumptions described above.

5.3.4 Truck Overloading, Component Life and Haul Cost

Implications of Figure 5.8 for overloaded trucks were investigated.

Variable Mean Overloading effects were extracted from Table 5.5 and posted to Table 5.6 over a range of mean payload to +30% in 5% steps. Figure 5.10 - data from

Table 5.6, illustrates Haul Cost Indices that allow for component life reduction with increase in mean payload.

Table 5.6 - Haul Cost Index Trends - Variable Dispersion Range and Mean Overloading

Variable Mean Overloading							
Mean Payload Over Target Mean	Mean	+5%	+10%	+15%	+20%	+25%	+30%
Haul Cost Index	1.000	1.042	1.097	1.171	1.274	1.424	1.656
Variable Dispersion Range							
Payload Dispersion Range	Mean	+/-5%	+/-10%	+/-15%	+/-20%	+/-25%	+/-30%
Haul Cost Index	1.0000	1.0003	1.0010	1.0023	1.0041	1.0064	1.0093

Productivity indices for the payload range +/- 20% of mean value were analyzed in Section 3.3.10. For example, Table 3.65, indicates a productivity index of 1.051 for an overload of 10% (that is an increase in payload index of 1.10 nett of the performance reduction due to overload) on a Cat 793 for Course 2 as analyzed by FPC – described in Section 3.3.10.

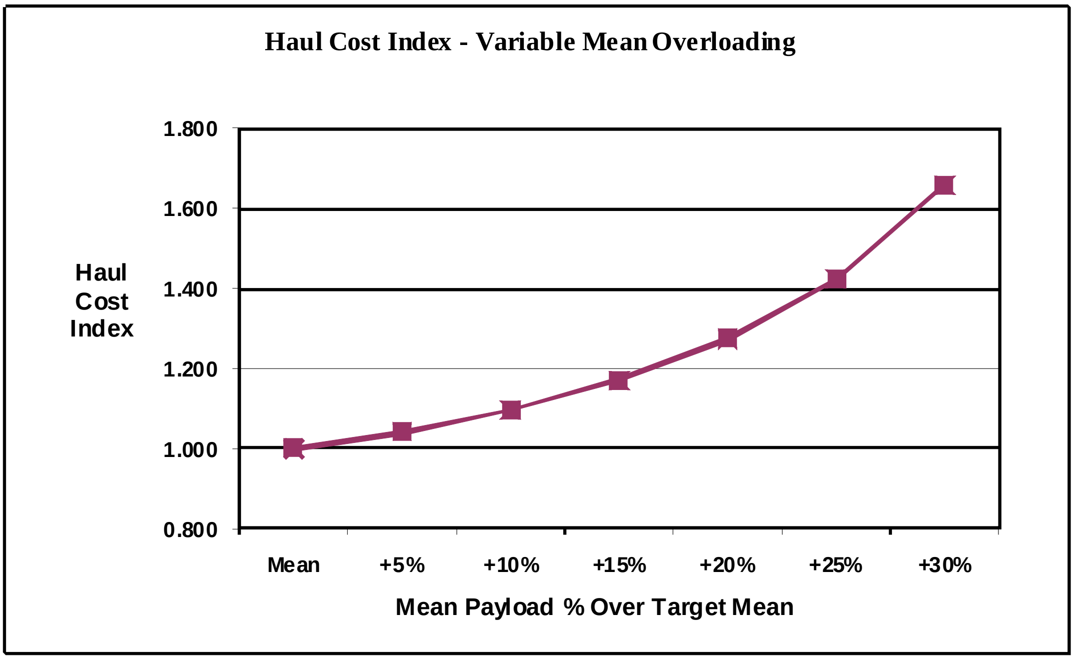


Figure 5.10 Haul Cost Index – Variable Mean Over Loading - From Table 5.6

Table 5.6, illustrated by Figure 5.10, indicates a haul cost index of 1.097 - allowing for driveline component life reduction at 10% overload. The quotient of 1.097/1.051 – i.e., the Haul Cost Index for 10% overload (due to component life reduction) divided by the Nett 10% Overload Productivity (after allowance for reduced truck performance) – provides the Accumulated Haul Cost Index for 10% over target payload.

It was recognized that range-effect due to payload dispersion on productivity or component life, i.e., losing more when overloaded than gained when under-loaded and vice versa, may or may not be relevant. Even if relevant, such effect may not be significant. Relevance and significance of range-effect payload dispersion is discussed in Section 5.3.6.

/Table 5.7 Accumulated Cost Indices for Variable Truck Payload

	Truck Payload %								
Potential Truck Trip Times	**80**	**85**	**90**	**95**	**100**	**105**	**110**	**115**	**120**
Course 1									
Fleet 4 Cat 789C	1.085	1.047	1.019	1.007	1.000	0.998	1.008	1.039	1.092
Fleet 5 Cat 793C	1.094	1.054	1.025	1.006	1.000	1.010	1.025	1.052	1.103
Fleet 6 Cat 797B	1.100	1.056	1.026	1.008	1.000	1.004	1.025	1.062	1.113
Course 2									
Fleet 4 Cat 789C	1.051	1.019	1.000	1.001	1.000	0.999	1.012	1.047	1.107
Fleet 5 Cat 793C	1.062	1.035	1.012	0.999	1.000	1.025	1.043	1.074	1.128
Fleet 6 Cat 797B	1.066	1.031	1.011	1.002	1.000	1.012	1.042	1.091	1.145
Course 3									
Fleet 4 Cat 789C	1.036	1.007	0.992	0.999	1.000	1.000	1.013	1.050	1.112
Fleet 5 Cat 793C	1.049	1.026	1.005	0.995	1.000	1.031	1.051	1.082	1.138
Fleet 6 Cat 797B	1.056	1.022	1.005	0.999	1.000	1.015	1.051	1.105	1.161

Accumulated Haul Cost indices are provided by FPC analysis described in Section 3.3.10. Analysis provided indices for the three largest Caterpillar trucks (180, 218 and 345 tonne nominal payloads), for three haul courses representing open pits of increasing depth. Results are recorded in Table 5.7, and illustrated by Figure 5.11.

5.3.5 Interpretation and Comments

For a selected overload or under load relative to the mean target payload, data In Table 5.7 and Figure 5.11 provides, an indication of the nett productivity and haul cost index taking reduced or increased truck performance into account. In addition, The Accumulated Haul Cost Index allows for reduced or increased component life due to truck overloading or under loading.

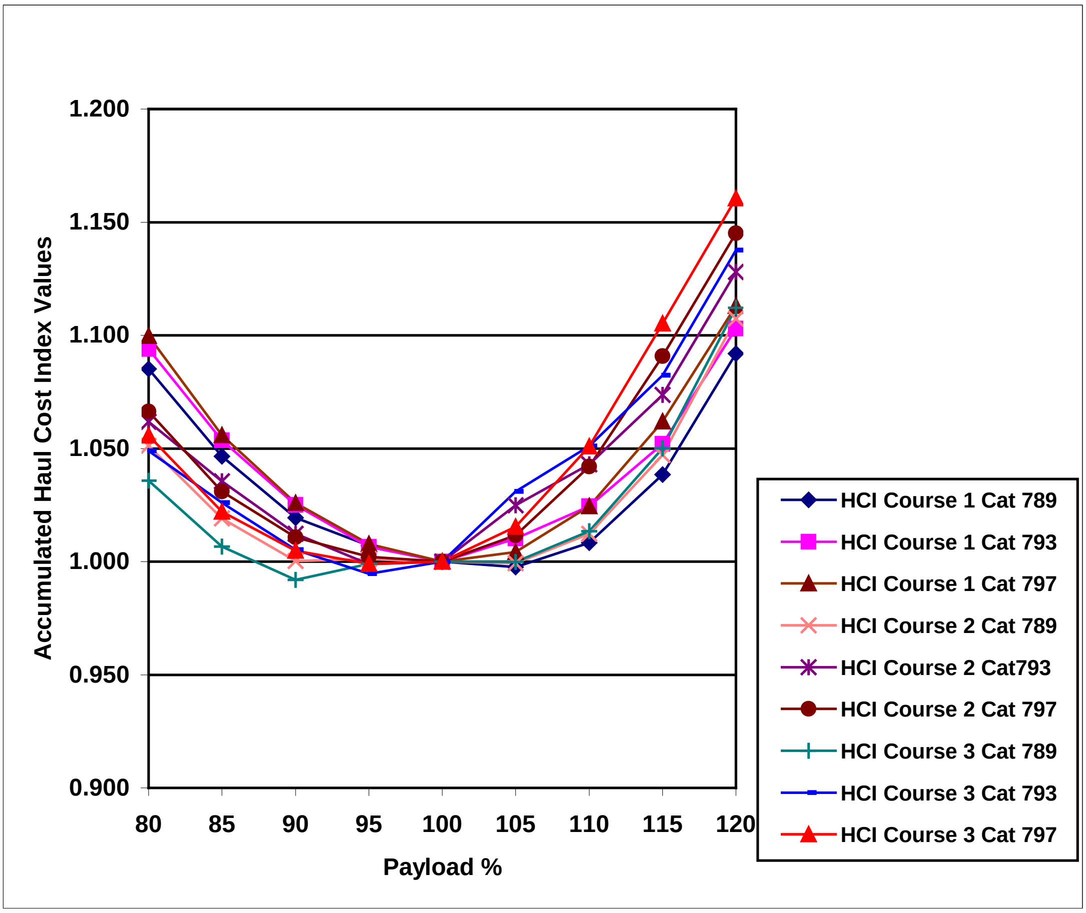


Figure 5.11 Accumulated Haul Cost Indices vs. Variable Truck Payload
From Table 5.7

The results of analysis in Section 5.3.4 are based on inferences and assumptions, identified throughout the discussion. To some degree the results may be considered hypothetical. But further inferences and assumptions listed below are consistent with operational experience and empirical observations. Figure 5.11 indicates that:

- Significant departure from mean target payload, say beyond the range, +2% to –5% can be expected to attract significant increase in haul costs.
- It is generally a lesser haul-cost penalty to under-load compared with an equivalent overload.
- As the haul distance increases, haul costs tend to be more sensitive to an overload than a comparative under-load.
- As trucks increase in size haul costs appear to be more sensitive to overload than an equivalent degree of under-load.

5.3.6 Payload Dispersion, Component Life and Haul Cost Index

It can be further inferred from Figure 5.8 that varying payload dispersion, i.e., increasing range of payload distributions, is accompanied by varying component life so varying haul cost index.

Considering each observation in a sample of truck payloads there will be an associated elemental component life depending on where in the normally distributed sample the individual payload observation falls. The aggregate of payloads will have an accumulated component life that will be positively skewed (to the right) as indicated by the component life trend line in Figure 5.8.

This implies that payload distributions will have escalating nett decrease in component life as the payload distribution range increases. That is, a wider payload distribution range will mean escalated component life decrease.

Analysis was performed to test the significance of the nett dispersion effect over different payload ranges using the indices in Table 5.5. Ranges +/- 5% to +/- 30% of target payload were analyzed in 5% steps.

A normal distribution probability model was adopted for sample distributions of payload and related cost indices (justified by the substantial evidence described in Sections 3.2.8 and 3.3.9. It was assumed that the Dispersion Haul Cost Index is the nett of the sum of over loads and under loads. The centroid (CG_U) of the under loads is at the 25 percentile point on the payload distribution and the centroid (CG_O) of the sum of overloads is at the 75 percentile point. These percentile points, 25 and 75, correspond to $\pm$ 0.675σ from the mean. For simplicity, the normal distribution model was truncated with range assumed at ±3σ. The error from this assumption is small, and, in the context of results, insignificant.

The arrangements of the above assumptions are sketched on Figure 5.7 – Caterpillar 10/10/20 Policy – (with due appreciation to Caterpillar).
For example the ± 30% range of payload CG% is:

$$0.675\sigma \bullet 30\%/3\sigma = \pm 6.75\%$$

For ± 30% range of payload – CG positions are:

Overload - CG_O Position = 106.75% of mean target payload.

Under load - CG_U Position = 93.25% of mean target payload.

Table 5.8 - Component Life/Cost and Haul Cost Indices v. Payload Dispersion

Payload Range	-30%	-25%	-20%	-15%	-10%	-5%	Mean	+5%	+10%	+15%	+20%	+25%	+30%	Nett Haul Cost Index
+/- 30%														
Payload CG %	93.25						100.00						106.75	
Component Life Index	1.067						1.000						0.916	
Component Cost Index	0.937						1.000						1.092	
Haul Cost Index	0.960						1.000						1.059	1.0093
+/- 25%														
Payload CG %		94.38					100.00					105.63		
Component Life Index		1.057					1.000					0.931		
Component Cost Index		0.946					1.000					1.074		
Haul Cost Index		0.965					1.000					1.048		1.0064
+/- 20%														
Payload CG %			95.50				100.00				104.50			
Component Life Index			1.047				1.000				0.946			
Component Cost Index			0.955				1.000				1.058			
Haul Cost Index			0.971				1.000				1.037			1.0041
+/- 15%														
Payload CG %				96.63			100.00			103.38				
Component Life Index				1.036			1.000			0.960				
Component Cost Index				0.965			1.000			1.042				
Haul Cost Index				0.978			1.000			1.027				1.0023
+/- 10%														
Payload CG %					97.75		100.00		102.25					
Component Life Index					1.024		1.000		0.974					
Component Cost Index					0.976		1.000		1.027					
Haul Cost Index					0.985		1.000		1.017					1.0010
+/- 5%														
Payload CG %						98.88	100.00	101.13						
Component Life Index						1.012	1.000	0.987						
Component Cost Index						0.988	1.000	1.013						
Haul Cost Index						0.992	1.000	1.008						1.0003

Corresponding Component Life, Component Cost and Haul Cost indices were drawn down from Table 5.5, as shown in Table 5.8.

The Dispersion Haul Cost Indices for the six cases were analyzed, summarized in Table 5.6 – entitled "Variable Dispersion Range" and illustrated by Figure 5.8. Dispersion (Nett) Haul Cost Indices are small – less than 1% haul cost penalty at the widest distribution range of ±30%.

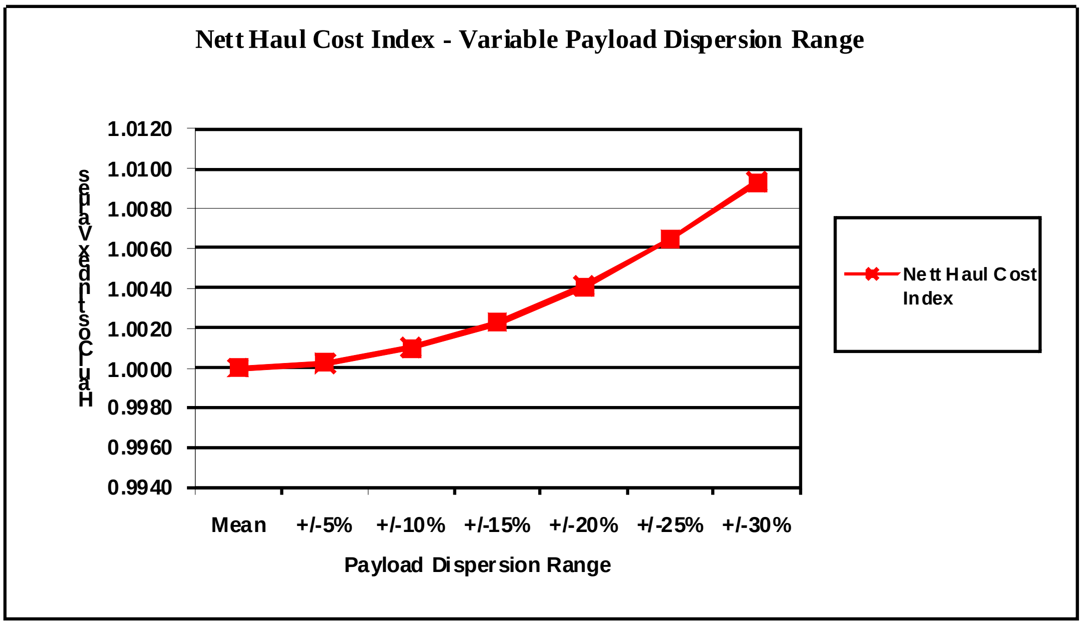


Figure 5.12 Dispersion (Nett) Haul Cost Index vs. Variable Payload Dispersion Range - From Table 5.6

5.3.7 Interpretation and Comments

On the basis of previous analysis of empirical evidence (truck payload data), deterministic simulation (FPC) and assumptions identified through the process, analysis in this section indicates that the effect of payload dispersion variability on driveline component life is small but relevant. Certainly, the effect on driveline life/cost is much smaller than the effect on productivity/cost of equivalent variation from target payload. This implies that the dispersion of payload distribution is not a significant issue if the upper range limit does not exceed the safety limit for truck operating, specifically braking. Considering sensitivity to the assumed scale for component life trend line in Figure 5.8, if assumed component life indices at extreme +/- 20% payload limits, used for solving the quadratic continuous function were doubled, Dispersion Haul Cost Indices would still be small; and, in the context of the analysis, can be considered as not significant.

There is still benefit from reducing upper limit of the payload distribution range below the quasi standard of +20% for compliance with the “20” element of Caterpillar’s 10/10/20 policy. The more centralized the payload sample becomes, i.e., narrower the payload distribution range the less variable will be truck performance, i.e., the more consistent will be truck travel times T_V and consequently truck trip times T_T. Hauling "rhythm" will improve tending to reduce bunching affect and increase truck-hauling efficiency.

These issues are further discussed in Section 5.5 below.

5.4 Costs and Bucket-Pass Sacrifice

5.4.1 Introduction

Productivity implications and circumstances justifying sacrifice of bucket loads to improve load and haul "rhythm" were analyzed and discussed in Section 3.3.9.

Analysis in Section 3.3.9, compiled as Table 3.57 appended in Volume 2, is limited to productivity only. This section analyzes and determines appropriate cost indices applied comparatively to test validity of bucket load sacrifice in terms of costs.

Also analysis in Section 3.3.9 and this section ignores any reduced efficiency of loading equipment due to increased truck exchanges. That is, loading equipment is virtually over trucked at any bucket sacrifice event. For a bucket cycle time of 30 seconds and truck exchange time of 20 seconds the theoretical percentage decrease in productive loading time (from Table 3.4):

- 5 pass to 4 pass is 88.24% to 85.71% = -2.5%.
- 6 pass to 5 pass is 90.0% to 88.24 % = -1.8%
- 7 pass to 6 pass is 91.3% to 90% = -1.3%
- 8 pass to 7 pass is 92.31% to 91.3% = -1.0%

So, generally bucket load sacrifice will be accompanied by 1.0% to 1.8% theoretical reduction in loading efficiency over the practical range of sacrificed passes 8^{th} to 7^{th} through to 6^{th} to 5^{th} passes. This reduction in total productivity (loading equipment controls total productivity when over trucked – Figures 3.68 and 3.71) is small and, for purposes of analysis and discussion, is ignored.

5.4.2 Comparative Cost Indices

An applicable Haul Cost Index (HCI) for additional payload proportions realized by all trucks, i.e., PL_X in Table 3.57 appended in Volume 2, can be derived from Table 5.2.

In the absence of bucket load sacrifice, waiting trucks have an estimated cost index per unit of time of 0.183 (Figure 5.2). Obviously if, by bucket load sacrifice, waiting time is converted to productive time, then additional production incurs an HCI of 0.817. This factor can be applied to PL_X to arrive at a comparative incremental haul cost HCX for the additional payload proportion realized.

Haul cost increment, HC_{LOSS}, due to payload lost by sacrificing one or more bucket loads (PL_{LOSS}) is a function of:

- Payload proportion sacrificed.
- HCI of the proportion sacrificed.

Payload proportion sacrificed can be determined from the ratio of the sacrificed bucket load to truck payload. In the compilation of Table 3.57 appended in Volume 2, this proportion was conveniently calculated for sacrifice of the 7th pass as follows:

PL_{LOSS} =0.75/6.75* = 0.111

The origin of denominator 6.75* is:

- 6 normal bucket fill passes rated at 1.00 (not necessarily bucket fill factor 1.00) – plus
- 1 final top-up pass factored at 0.75 of a normal bucket fill as per discussion in Section 3.3.9.

It could be argued that the first-pass bucket load is generally higher than the mean of subsequent intermediate loads (excluding the last bucket load) as discussed in Section 3.2.8 and illustrated by Tables 3.7 and Table 3.12 (appended) where it is indicated that mean of first bucket loads is in the order of 130% of mean intermediate bucket loads.

It is intuitively believed that some one-third, i.e., 10% of the apparent increased bucket load is accumulated debris on the truck frame and carry-back in the truck body that should be registered as NMW by recalibration of the payload measuring facility on the mining trucks studied as discussed in Section 3.2.8. This implies that first bucket loads could be in the order of some 20% in excess of mean intermediate bucket loads.

For the current discussion, this further implies that factoring down of bucket fill of last bucket (top-up) loads will be offset by factoring up first bucket loads. At first sight, review of Tables 3.7, 3.10, 3.15, and 3.17, shows that the mean of "all" bucket loads and "intermediate" bucket loads are the same, implying that first and last bucket loads are compensating.

Due to the reduced number of first bucket-load observations relative to last bucket loads, the author is uncomfortable with this as a general conclusion. Any anomalies in NMW due to oversight of tare adjustment or problems with efficacy of auto-adjustment will affect first bucket load measurement.

For the purposes of the current discussion all bucket loads to the last have been assigned a factor of 1.00 with the last, top-up, bucket load assigned 0.75. Assumed factor assignments may tend to enhance the apparent benefit of bucket sacrifice; but for the practical range of bucket passes per truck payload any error due to the adopted factors applied to bucket fills will be 0.5% or less.

Table 5.9 below develops Haul Cost Indices (HCI) for the payload proportion sacrificed – PC_{LOSS}.

Table 5.9 Haul Cost Index for Payload Proportion Sacrificed

* Haul cost comparisons are truck and haul course specific. For example, Table 5.7 provides tabulated indices by interpolation for a Cat 793C on Course 2 over the range of truck under loads after bucket load sacrifice.

Nominal Passes	Sacrificed	Payload Proportion Sacrificed	Payload Proportion Hauled	Haul Cost Index (HCI) from Table 5.7 *	Value Hauled	Value Sacrificed	Haul Cost Index of Sacrificed Proportion
6	**8**	0.097	0.903	1.012	0.914	0.086	**0.888**
5	***7***	***0.111***	***0.889***	***1.015***	***0.902***	***0.098***	***0.880***
4	**6**	0.130	0.870	1.025	0.891	0.109	**0.833**
3	**5**	0.158	0.842	1.037	0.873	0.127	**0.803**
1	2	3	4	5	6	7	8

Referring to Table 5.9:

- Column 1 – shows the nominal number of passes – i.e., the "modal" value.
- Column 2 – indicates pass sacrifice.
- Column 3 – indicates payload proportion sacrificed calculated by:

  $PL_{LOSS} = 0.75/(\text{Pass sacrificed} - 1 + 0.75)$

  Example: $PL_{LOSS} = 0.75/(7 - 1 + 0.75) = 0.111$ as in Table 5.4.2.1
- Column 4 – indicates balance of payload hauled.

  That is: Payload nett of PL_{LOSS}.

  In Table 5.9 Payload factor = 1.0
- Column 5 – lists HCI derived from analysis over a range of payloads either side of the mean shown in Table 5.7.
- Column 6 – shows:

  Value Hauled = Payload proportion hauled x HCI.
- Column 7 – shows:

  Value Sacrificed = 1 – Value Hauled
- Column 8 – shows:

  HCI of Sacrificed Portion = Value Sacrificed/ Payload Proportion Sacrificed

5.4.3 Cost Criteria for Sacrificing Bucket Loads

Table 5.10, appended in Volume 2, is an extension of Table 3.57, also appended in Volume 2. The HCI for the pass sacrificed is applied to PL_{LOSS} – the proportion of truck payload sacrificed by a bucket load or loads sacrificed. A copy of the file is provided on the CD inside the back cover of this thesis to complement the file for Table 3.57.

As discussed in Section 5.4.2, the HCI to be applied to PL_X – the additional payload proportion realized by bucket load sacrifice is derived from Table 5.2 by the difference between operating HCI 1.00 and waiting time HCI of 0.183, i.e., 0.817.

The comparative data is shown in Table 5.10, appended in Volume 2. Within that table shading and lines delineate significant zones, each with status as follows:

- The shaded zone indicates values for Haul Cost Benefit – HC_X – that do not exceed Haul Cost Sacrifice – HC_{LOSS}; so bucket pass sacrifice is not beneficial for the selected case of nominal 5-pass loading and sacrificing the 7th pass.

- The heavy full line identifies the boundary of the Number of Trucks v. Truck Trip Time array where bucket pass sacrifice changes from “unfavorable” to “favorable”. The heavy dashed line indicates the boundary where bucket pass sacrifice passes from “unfavorable” to “favorable”, if the total haul cost is charged to waiting time when the truck is technically operating, but waiting to be loaded.

This latter instance is conservative in reducing the zone favorable for bucket load sacrifice. It could apply where accounting treatment of amortization is expensed as an operating cost or cost provision. Traditional cost estimating for mining contract scheduled rates necessarily includes capital recovery for all equipment supplied in the rates for contract work (unless it is priced transparently as a periodic, such as monthly, fee).

It is the author’s opinion that including total amortization cost in waiting time costs is invalid for analyzing bucket-pass sacrifice benefits. But the error by inclusion is intuitively considered to be small in the context of the comparative cost test.

As discussed in Section 5.3.2, driveline componentry life and costs are related to fuel burn. There is a logical case to cost the entire capital write-off over fuel burn for the life of the mining truck.

As an example: If a Cat 793C mining truck uses 2.7 million litres of fuel over an engine life before overhaul – enduring some 14,000 operating SMU (engine hours) in the process – then we could allow, say, 5 x engine lives (70,000 SMU) as the life of the mining-truck frame and all componentry not replaced or rebuilt earlier in the truck life. The life of a Cat 793C mining truck would then be 13.5 million litres of fuel. This begs the question: Is this a reasonable life assessment? Assuming it is reasonable for purposes of discussion, it can be concluded that including full write off of capital cost per SMU whilst a truck is waiting to be loaded would be unreasonable as fuel used by an idling engine is some 5% of the mean operating fuel consumption. This supports the above discussion and the position taken in compiling Table 5.10, appended in Volume 2.

5.4.4 Interpretation and Comments

Generally there is justification to sacrifice any bucket pass exceeding the sum of nominal passes (modal value) plus one additional pass provided that:

- There is a truck waiting to be loaded or in position to exchange immediately on departure of the loaded truck.
- The difference in comparative outcomes between the productivity test of Table 3.57 and the cost test of Table 5.10, both appended in Volume 2, is generally small. Compared with the haul-cost test the productivity test excludes only a small number of additional cases because the applicable HCI for PL_{LOSS} is only slightly higher than the HCI applicable to PL_X.
- The higher the HCI for PL_X (and vice versa – lower the HCI for PL_{LOSS}) the more robust will be the justification for bucket-pass sacrifice.
- If the proportion of operating labour in the HCI applicable to PL_X (see Table 5.2.) is reduced by increasing truck payload scale, HC_X will tend to increase - favoring bucket-pass sacrifice.
- As fuel and tyre cost proportions increase (by general cost escalation and sharp rises due to international supply limitations) HC_X will tend to increase – again favoring bucket-pass sacrifice.

Variability of travel time T_V for mining trucks means that, (for best practice truck haulage with $CV_{TV} \leq 0.10$), unless truck match is 50% over or under exact match some degree of bunching will occur – refer to Figure 3.70. Regardless of the theoretical status of loading equipment/truck match within the +/- 50% limits, loading equipment experiences over trucking, under trucking and, logically, occasionally an exact match.

Bucket-pass sacrifice is likely favored where there are trucks waiting to be loaded or where a returning truck can seamlessly complete the truck exchange with a loaded truck. In these circumstances, application of a protocol limiting bucket passes will be productivity positive and cost beneficial. In the absence of trucks waiting it is commonsense for loading equipment operators to continue loading trucks to the target (mean) payload using volumetric judgment or onboard payload sensing output as a guide.

5.5 Costs and Over trucking, Under trucking; also Bunching

5.5.1 Introduction – Defining the Problems

Relative merits of over trucking and under trucking were discussed in terms of productivity in Section 3.5.3.

The three states, over trucked, exactly matched and under trucked were considered as follows:

- Over trucked, where productivity is at maximum loading equipment performance.
- Under trucked, where productivity is at a proportion of the maximum loading equipment performance consistent with the actual number of trucks relative to the theoretical number of trucks at the match point.
- The hypothetical exactly matched case where actual trucks correspond to theoretical trucks – obviously only possible where the theoretical number of trucks is an integer.

The matched case can be considered trivial because it is rarely, if ever, going to occur. It is that point where discrete functions expressing the relationship between selection parameters for either over trucking or under trucking intersect. At that point the theoretical number of trucks, calculated by methods described in Section 3.5.2 and Section 3.5.3, coincides with an integer number.

Relevance to Equipment Selection

Since the “matched” case can be considered trivial, there are two practical options:

1. To operate overtrucked; or
2. To operate undertrucked – or “over-shoveled” as indicated in Section 3.5.3.

When over trucked, loading equipment controls productivity that coincides with the practical performance of that loading equipment – but with a cost penalty for under-utilized trucks.

When loading equipment is under trucked effective trucks are fully utilized; but loading equipment is under-utilized to the extent that the actual number is less than the theoretical number of trucks – at a cost penalty due to inefficiency of loading equipment.

Practical operational choices are to:

- Operate over trucked at full loading equipment productivity with a cost penalty for under-utilized trucks; or
- Operate under trucked at reduced loading equipment productivity with an attendant cost penalty due to loading inefficiency.

The above choices beg the question – Which option is the most economic? Intuitively the cost penalty for loading equipment inefficiency will be less than the cost penalty for overtrucking. Relativity of hauling cost some three times loading cost is indicated by Figure 2.2, supporting under trucking as a best-economics policy. But reduced loading-equipment productivity when under trucked could have cost implications for downstream treatment and subsequent activities. Opting to under truck is not a simple matter for intuitive decision.

Estimation of number of trucks, initially based on relativity of loading time to truck travel time, amended by probabilistic analysis is discussed in Section 3.5.2. The estimated match is rarely an exact integer number of trucks. When the theoretical number of trucks is not an exact integer we have the option of rounding up or down.

Experience indicates that mining productivity and cost estimators have a conservatively-driven penchant to round up. This preference will tend to over-truck with cost penalty due to truck waiting time. Rounding down under trucks, but inflicts a cost penalty due to loading inefficiency. Guidance for the appropriate rounding decision is needed. This is the focus of Section 5.5.2.

5.5.2 Productivity, Cost Indices and Relationships

5.5.2.1 Hanby's Solution – A Review

In Part 1 of a three-part paper I R Hanby addressed the problem of "shovel and truck matching to obtain whole numbers" of trucks (Hanby, 1991, Part 1). The basic criterion for the "simple solution" developed by Hanby is "minimum cost per unit of volume moved". Hanby's objective was to provide a method of determining the solution of an integer (selection) number of trucks for any theoretical number of trucks determined by traditional estimating methods to provide lowest load and haul costs.

In review, Hanby's deterministic method is, to a degree, simplistic in that the unit cost of owning and operating loading equipment and trucks is assumed constant for all states of operation whether equipment is performing its assigned function (loading or hauling) or waiting for trucks (loading equipment) or waiting to be loaded (trucks). Hanby assumed a single loading equipment item and that all trucks were alike and that truck-travel and loading times (and consequently loading times, if any) are mean values with all trucks of identical specification and performance.

Hanby's analysis was based on the theoretical number of trucks t for a match to be calculated deterministically by his equivalent to equation (7) from 3.5.2, viz.,

$t = (t_L + t_T)/t_L$ - using Hanby's notation and time definitions.

Generally equivalent to (using notation and definitions herein):

$$t = T_T/(T_L + T_S) \quad \equiv \quad t = (T_L + T_V + T_S + T_D)/(T_L + T_S)$$

The above equation produces continuous values of t that are generally non-integer. Benefits of probabilistic analysis in determining the theoretical truck match number before rounding to an integer number were described in Section 3.5.2. Theoretical matched number of trucks is discussed in Section 3.5.3.

Hanby's analysis is applicable for the theoretical number of trucks regardless of the method of determination. Relevant simplifications fundamental to Hanby's analysis are reviewed in detail in Section 5.5.3.

5.5.2.2 Costs and Outputs

Notation used below in this section; also in Section 5.5.3 below, is based on Hanby with modification necessary to suit notation, particularly subscripts, throughout the research and analysis described in this thesis.

Hanby showed that the ratio of total load and haul cost to truck cost is a function of the ratio of loading equipment cost to truck cost, the actual integer number of trucks selected and the theoretical number of trucks required for haulage service.

In summary:

$PS/C_T = R + n$ where $t < n$

$PS/C_T = (R + n) \cdot t/n$ where $t \geq n$

P, S, C_T, R, n and t are defined in Section 5.5.3. PS/C_T is a dimensionless cost index that represents relative load and haul cost per unit volume as shown in Section 5.5.3. The right hand side of the above equations is also dimensionless and relates load-and-haul cost to the ratio between loading equipment and truck cost, R, actual integer number of trucks n and theoretical number of trucks t.

5.5.3 Previous Research - Extended Analysis

Hanby summarizes results to provide a method of solving for truck numbers in whole, integer, number of trucks. Outcomes of Hanby's analysis could be rearranged to provided a method of determining whether non-integer theoretical truck numbers t should be rounded up or down to provide actual trucks n for best load and haul costs.

Part 1 of Hanby's paper is a summary, omitting most of the algebraic manipulation to arrive at his results (Hanby, 1991, Part 1). Analysis herein is complete in more detail and extended to provide a basis for interpretation of the most cost effective rounding procedure; also to consider the effect of truck waiting time on cost analysis.

Following is a review of Hanby's analysis as summarized in Section 5.5.2.

5.5.3.1 Terms and Definitions

t = Theoretical number of trucks (non-integer).

n = Actual number of trucks selected (integer).

C_L = Owning and operating costs of loading equipment per unit time.

C_T = Owning and operating costs of trucks per unit time.

C = Owning and operating costs of load and haul group per unit time.

R = C_L/C_T

S = Loading equipment productivity per unit of time.

V = Productivity of load and haul group per unit of time – using n trucks.

P = Cost per unit volume of production.

5.5.3.2 Analysis

Owning and operating costs of operating "team" of a loading equipment item and a truck group of n trucks:

$$C = C_L + n \cdot C_T \quad (1)$$

Productivity of a production "team":

$$V = S \quad \text{if } t < n \quad (2a)$$

$$V = S \cdot n/t \quad \text{if } t \geq n \quad (2b)$$

$$P = C/V \quad \text{cost per unit volume} \quad (3)$$

Substituting equations (2a) and (2b) in equation (3) for the two cases;

$$P = (C_L + n \cdot C_T) / S \quad t < n \quad (4a)$$

$$P = (C_L + n \cdot C_T) \cdot t / S \cdot n \quad t \geq n \quad (4b)$$

Substituting R for C_L/C_T – i.e., the ratio of owning and operating cost of loading equipment / truck owning and operating cost:

$$PS/C_T = R + n \quad t < n \quad (5a)$$

$$PS/C_T = (R + n) \cdot t/n \quad t \geq n \quad (5b)$$

Linear equations (5a) and (5b) above for the two states of $t < n$ (overtrucked) and $t \geq n$ (matched or undertrucked) provide the following interpretations:

- For $t \geq n$ cost index PS/C_T is a linear function of t of gradient $(R + n)/n$.
- For $t < n$ cost index PS/C_T is a constant function $(R + n)$ (pedantically a linear function of t with gradient zero).
- Where $t = n$, the match point, the two functions have the same value $(R + n)$ and so intersect.

For any selected value of n trucks the relationship between cost index PS/C_T and theoretical trucks t can be expressed as a pair of straight lines intersecting at the match point where $t = n$.

Defining theoretical number of trucks (for a loader – truck match) for minimum cost per unit volume loaded and hauled by:

t_1 = the minimum value of t theoretical number of trucks

t_2 = the maximum value of t theoretical number of trucks

for, in each case, any selection of n trucks – an integer number.

Hanby considered PS/C_T as the dependent variable and t as the independent variable in constructing his illustration of equations (5a) and (5b). Hanby's focus was on a technique to derive n integer number of trucks. In reviewing and extending Hanby's calculations, opportunity was recognized to derive a protocol for rounding theoretical truck number derivations up or down to actual integer number of trucks n retaining best load and haul economics. The following analysis views theoretical number of trucks t as the dependent variable. For each individual calculation one of the remaining variables, n, R and cost index PS/C_T, is chosen as the independent variable and the other variables remain constant. This simplifies the analysis to sets of linear equations. Equation (5b) was transposed as follows:

$$t = PS/C_T \bullet n/(R + n)$$

Considering the match point where t = n :– moving on to t_2,

$$t_2 = n + n/(R + n)$$

$$= n(R + n + 1)/(R + n) \qquad (6b)$$

t is a continuous variable that, for each pair of equations corresponding to a given n value, repeats so that t_2 for any pair is t_1 for the next ascending pair of linear equations. By substituting n - 1 for n in (6b), t_1 is derived.

$$t_1 = (n-1)(R+n-1+1)/(R+n-1)$$

$$t_1 = (n - 1)(R + n)/(R + n - 1) \qquad (6a)$$

Hanby interpreted equations (6a) and (6b) as follows: "for a cost ratio R, n trucks should be used if the theoretical number of trucks lies between t_1 and t_2".

Table 5.11, Parts, A, B and C – appended in Volume 2, lists values for t_1, matching t and t_2 for:

- R = 1, 2 or 3.
- Ranges of selected truck numbers n (integers).

Hanby implies that the practical range for R-values is 1 < R < 3. Any loading equipment/truck cost combinations outside of this range: “may indicate either a type mismatch or unusual owning and operating costs” (Hanby, 1991, Part 1).

In Table 5.11 a practical range of n from 1 to 7 was supplemented with hypothetically large selected numbers of trucks (20 to 100) to examine extreme effects. Results corresponding to the practical range n = 1 to 7 were summarized at the bottom left of each table to facilitate illustrations.

Figure 5.13 (from Table 5.11 Part A) illustrates the relationship between theoretical number of trucks t as the dependent variable and actual selected trucks n as the independent variable. t maxima, t match and t minima are plotted for loader/truck cost ratio R = 1.

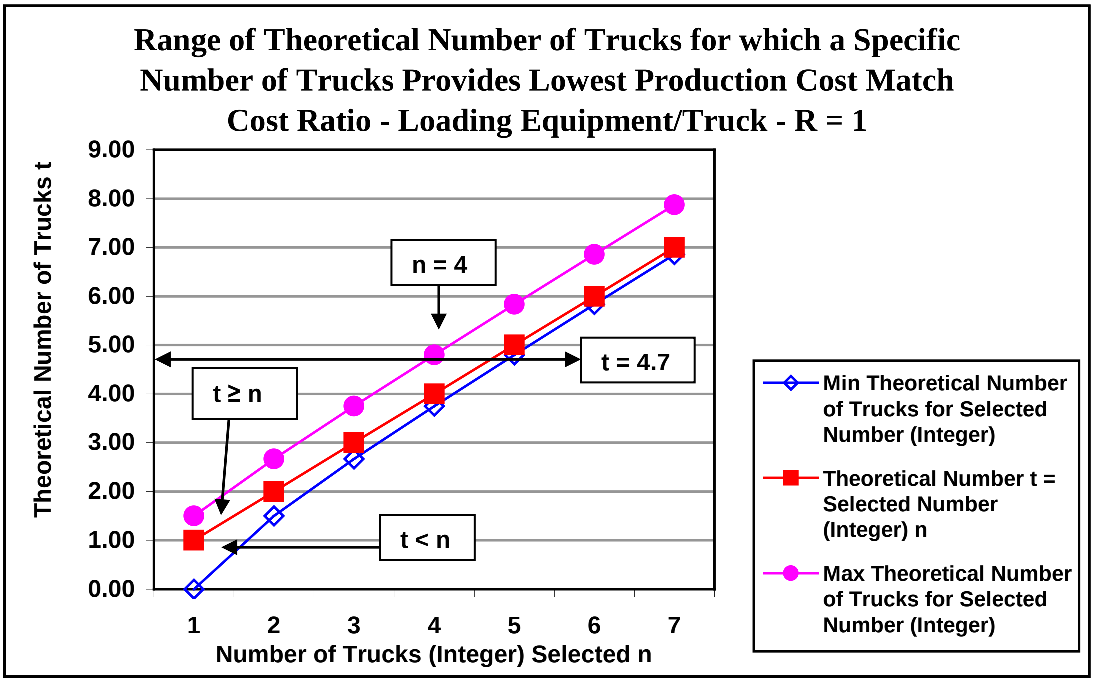


Figure 5.13 Range of Theoretical Number of Trucks From Table 5.11 Part A

Features of Figure 13:

- The t range where the two regimes of t < n an t ≥ n apply is indicated.
- A hypothetical value for t = 4.7 is illustrated by entering the diagram from the ordinate value of 4.7 and projecting horizontally to intersect the n = 4 vertical projection.
- t = 4.7 passes below the minimum value of t for n = 5.

- Rounding up to 5 in this instance would over truck at increased unit load and haul cost.

The example illustrated by Figure 5.13 is for R = 1, i.e., towards the lower end of the practical R range. The effect of higher R values is further considered in this section.

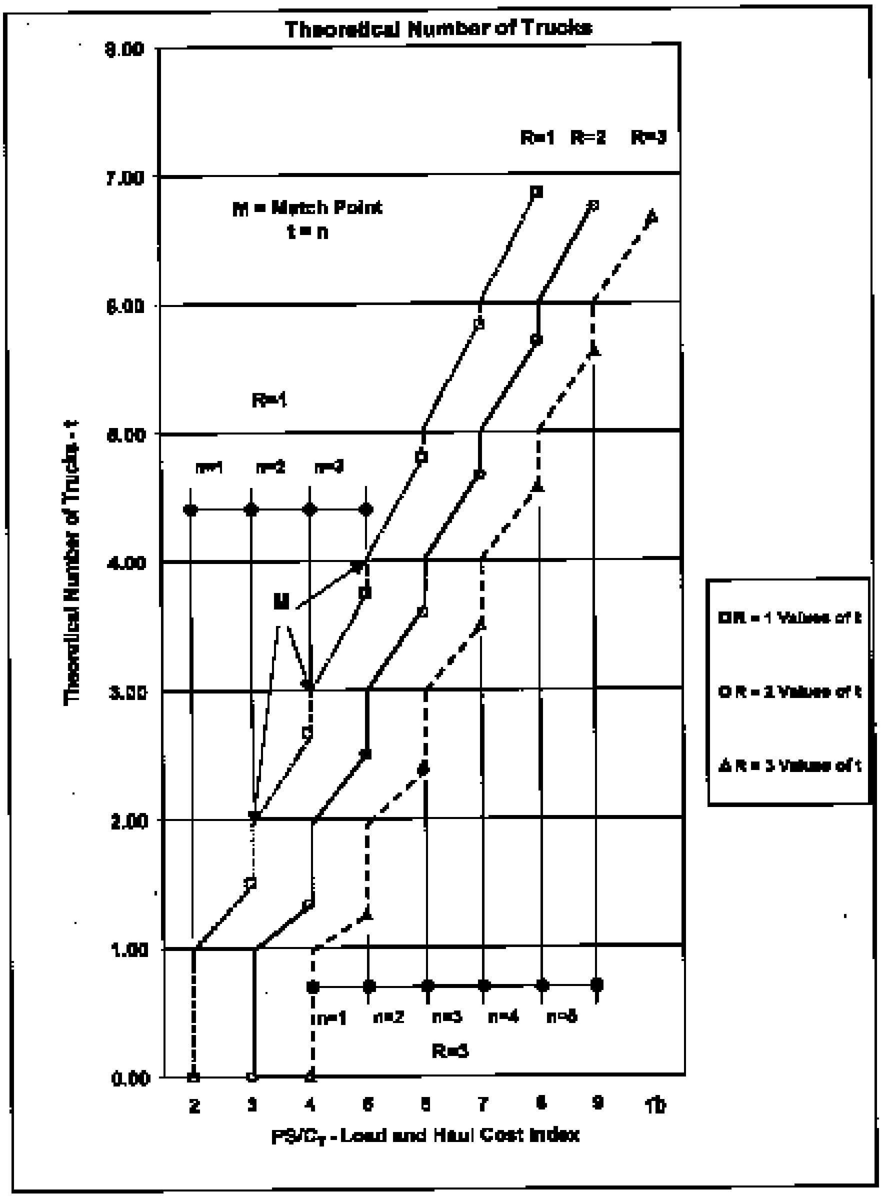


Figure 5.14 Theoretical Number of Trucks
From Table 5.12

Table 5.12 Summary of Theoretical Truck Numbers for Selected Actual Trucks n and Related Values of Cost Index PS/C_T

R = 1									
n	1	2	3	4	5	6	7		
PS/C_T	2	3	4	5	6	7	8	9	10
t	0.00	1.50	2.67	3.75	4.80	5.83	6.86		
R = 2									
n	1	2	3	4	5	6	7		
PS/C_T	2	3	4	5	6	7	8	9	
t		0.00	1.33	2.50	3.60	4.67	5.71	6.75	
R = 3									
n	1	2	3	4	5	6	7		
PS/C_T	2	3	4	5	6	7	8	9	10
t			0.00	1.25	2.40	3.50	4.57	5.63	6.67

Figure 5.14 below is similar to Hanby's illustration of the same concept (Hanby, 1991, Part 1), but with dependent and independent variables reversed.

Table 5.14 is an extension from Table 5.11, appended in Volume 2, with PS/C_T values interpolated for t_1 values that were calculated from equation (6a). For each pair of linear equations, for a given value of n, and for each of the three cases R = 1, 2 and 3, minimum values of t_1 were plotted. The three sets of minimum values – t_1 - were connected the match points, t = n, in each case. Connecting match points to the maximum value t_2 in each case (i.e., the minimum value t_1 for the next ascending pair of linear equations) completed the illustration. Figure 5.14 was constructed (with theoretical trucks t as the dependent variable and cost index PS/C_T as the independent variable) to trace in detail Hanby's calculations for best understanding of the relationships illustrated.

Table 5.13 is an extract rearranged and extrapolated from Table 5.11.

Figure 5.15 is from data in Table 5.13, and replicates Hanby's treatment of the relationship between loading/truck cost ratios R, theoretical number of trucks and best-cost truck number selection.

Table 5.13 Maximum Theoretical Number of Trucks for Specific Truck Number (Integer) Selection

Cost Ratio Index R	0	1	2	3	4	5	6
Number of Trucks - n	Values of Maximum Number of Trucks - t_2						
1	2.00	1.50	1.33	1.25	1.20	1.17	1.14
2	3.00	2.67	2.50	2.40	2.33	2.29	2.25
3	4.00	3.75	3.60	3.50	3.43	3.38	3.33
4	5.00	4.80	4.67	4.57	4.50	4.44	4.40
5	6.00	5.83	5.71	5.63	5.56	5.50	5.45
6	7.00	6.86	6.75	6.67	6.60	6.55	6.50
7	8.00	7.88	7.78	7.70	7.64	7.58	7.54

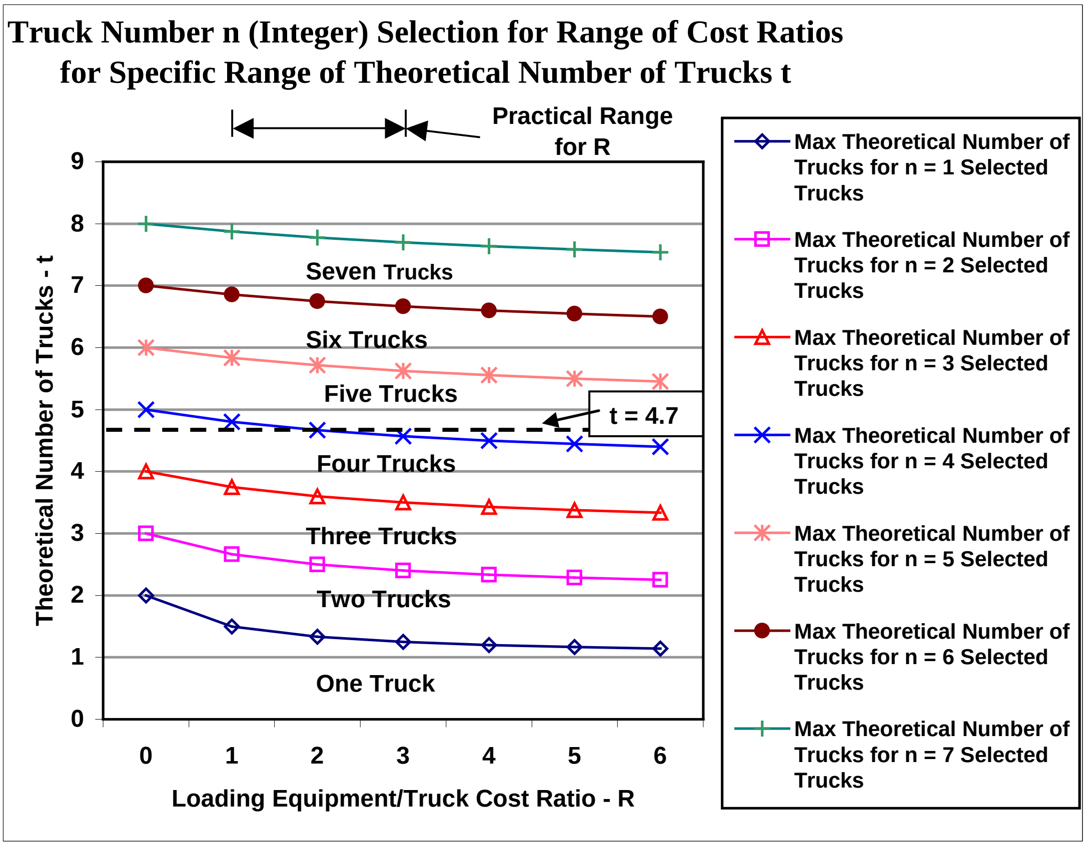


Figure 5.15 Truck Number n (Integer) Selection for Range of Cost Ratios For Specific Range of Theoretical Number of Trucks t From Table 5.13

Table 5.13, retains theoretical trucks t as the dependent variable – specifically t_2 – the maximum theoretical number of trucks for best load-and-haul costs consistent with each field of n selected trucks. Loading equipment/truck cost ratio R is assigned as the independent variable. Figure 5.15 illustrates the successive fields consistent with each number of selected trucks n. Hanby described this "nomogram" as a "graphical solution" for determining an integer number of trucks from equations (5a) and (5b) for theoretical number of trucks t (Hanby, 1991, Part 1). This was the principal outcome of Hanby's research.

Hanby's "nomogram" is consistent with the facility of Figure 5.15 where, having determined theoretical trucks t (by any acceptable method) and the load/haul cost ratio R for the particular equipment fleet, the best-cost number of trucks n can be selected. The "nomogram" is entered from the ordinate at the theoretical number of trucks to intersect with a vertical through the appropriate R-value for loading equipment/truck cost ratio. The field where the intersection falls determines the integer number of trucks n for best load and haul costs. As an example a theoretical result of t = 4.7 is drawn on Figure 5.15. For R = 1 n = 4 trucks (by rounding down). But as R increases n = 5 trucks (by rounding up) is indicated for optimum costs. For R = 2 the diagram does not clearly discriminate between n = 4 or n = 5. Hypothetically either solution of "4" or "5" for n would produce practically the same load and haul unit costs. Scale of the diagram could be adjusted to provide a clearer discrimination. But in practical terms there will be R-values for theoretical t solutions where rounding up or rounding down will result in similar load and haul unit costs.

Table 5.14 below is a further extract of data from Table 5.11, appended in Volume 2, to facilitate Figure 5.16.

Table 5.14 Summary of Theoretical Truck Numbers for Range of Cost Ratio - R

Summary - R = 1							
Selected Number of Trucks n	1	2	**3**	**4**	**5**	**6**	7
Minimum Theoretical Trucks t_1	0.00	1.50	**2.67**	**3.75**	**4.80**	**5.83**	6.86
Theoretical Truck Match t = n	1	2	**3**	**4**	**5**	**6**	7
Maximum Theoretical Trucks t_1	1.50	2.67	**3.75**	**4.80**	**5.83**	**6.86**	7.88
Summary - R = 3							
Selected Number of Trucks n	1	2	**3**	**4**	**5**	**6**	7
Minimum Theoretical Trucks t_1	0.00	1.25	**2.40**	**3.50**	**4.57**	**5.63**	6.67
Theoretical Truck Match t = n	1	2	**3**	**4**	**5**	**6**	7
Maximum Theoretical Trucks t_1	1.25	2.40	**3.50**	**4.57**	**5.63**	**6.67**	7.70

Figure 5.16 is an alternative (simpler?) illustration of the relationship between and theoretical number of trucks t (assigned as the dependent variable) and best-cost truck number selection n (the independent variable) for three??? cases - loading/truck cost ratios R. Using theoretical 4.7 trucks as an example, as drawn on Figure 5.5.3.4, it is shown that, for R = 1, rounding down to n = 4 trucks is favoured. As R increases towards 3 rounding up to n = 5 is favoured.

Practitioners can develop the most convenient form for their own truck determination diagrams to suit the loading equipment and truck parameters applicable to their operations.

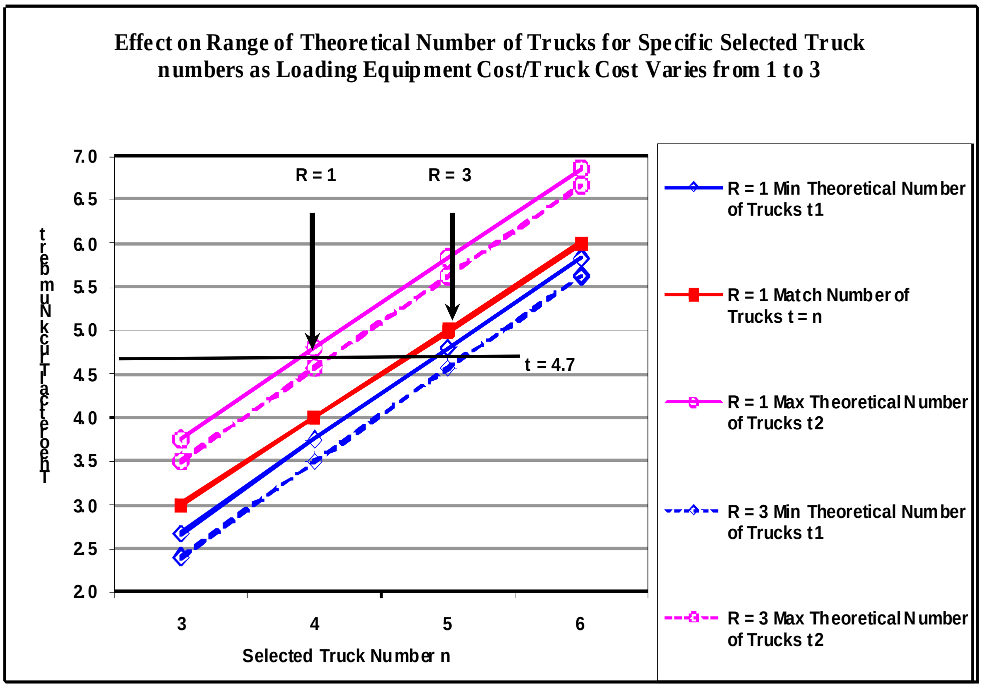


Figure 5.16 Effect on Range of Theoretical Number of Trucks for Specific Selected Truck Numbers as Loading Equipment Cost/Truck Cost ratio Varies from 1 to 3
From Table 5.14

5.5.4 Outcomes, Interpretation and Comment

5,5,4,1 Summary of Outcomes

Relationships between theoretical trucks t, loading/truck cost ratio R and selected number of trucks n indicated:

- For practical ranges of selection parameters, rounding down of determined t values to selected n trucks is generally more favoured than rounding up – consistent with the support for under trucking to provide best costs for loading and hauling as discussed in Section 3.5.3.
- As truck numbers increase, minimum theoretical number of trucks t_1 tends to be asymptotic to the matching number of trucks, i.e., the corresponding selected actual number of trucks n – therefore, for increasing number of trucks, rounding down tends to be more favourable.
- As loading-equipment/truck cost ratio increases, rounding up from the theoretical number t to selected integer number n tends to be more favourable.
- Where loading equipment cost is high ($R \to 3$ or more), the addition of an extra truck has less impact on load and haul unit costs and rounding up tends to be more favourable.

The relationship between nominal numbers of passes per truck payload, one-way haul distance and interrelated loading equipment and truck operating parameters has been discussed in a number of contexts. In general it can be predicted that:

- For short hauls, where three or four-pass loading is favoured, then cost ratio R will tend towards the upper limits of the practical range ($R \to 3$ say). Selected trucks n will tend to be theoretical trucks t +/- 0.5 - difference does not clearly indicate favoured rounding direction.
- For long hauls, where five or six-pass loading is acceptable, R values will tend to be in the lower end of the range, $R < 2$ say; number of trucks will be in the upper range – say 5 plus – then selected number of trucks n will tend to be theoretical trucks t + 0.25/-0.75 – rounding direction will be more obvious.

Pursuing three and four-pass loading, in certain circumstances, has been shown to have some disadvantages, including payload distribution dispersion and lower intrinsic loading equipment efficiency. Analysis in this Section 5.5.4.1 produces loading and hauling performance indices with higher R-values that result in rounding errors tending to over truck or under truck to a degree reflected by cost penalties measurably higher than alternative fleet arrangements. That is – more passes, more trucks and lower loading equipment/truck cost ratio R can be measurably cost beneficial.

5.5.4.2 Limitations of Hanby's Analysis

Hanby's analysis assumes that owning and operating cost of loading equipment and trucks is constant for each hour of operating time, adding an element of conservative bias. As developed by traditional methods, owning and operating cost per hour or SMU (service meter unit) are, to some degree, empirically based; so tend to be average costs per unit of elapsed time. Generally owning and operating costs spread annualized cost over estimated productive operating time. Equipment costs are either estimated averages for any state of operating (working or waiting) or zero when non-operating. Traditional practices tend to estimate conservatively high unit production costs that, to a substantial degree, tend to offset errors and omissions.

During operations there are periods when loading equipment and trucks are operating but not productive. These differing operational phases were discussed in terms of time definitions in some detail in Section 3.6.3 and illustrated by Table 3.75 appended in Volume 2.

In load and haul operations both loading equipment and trucks experience times when equipment is operational but not productive, i.e., waiting. This raises the issue of accuracy implications of the simple assumption of only two cost phases, productive and non-operational, the basis of Hanby's analysis. Justification for adding another phase of waiting time when equipment is ready for work but not productive is warranted.

Intuitively the two-phase cost simplification is considered not to reflect adversely on Hanby's results and conclusions. Analysis in Section 5.5.5, in addition to considering waiting-time cost-effects in some detail, also provides confirmation that the outcomes of Hanby's analysis in terms of integer truck numbers are generally valid. But when considering comparisons in terms of absolute cost indices analysis based on more realistic time phases (three-phase cost analysis) will provide more accurate assessment of costs

5.5.5 Overtrucking, Undertrucking - Productivity and Costs

The general problems of overtrucking, undertrucking, fleet matching and bunching were discussed in terms of productivity in Section 3.5.3 and Section 3.5.4. It should be noted that productivity and related cost analysis are generally in terms of mean or average values for variables. The following analysis is similarly limited.

This section will address resultant cost effects due to productivity influences of overtrucking, undertrucking and bunching.

5.5.5.1 Effect of Waiting Time on Load and Haul Costs

The analysis in Section 5.5.3 is used as a framework to develop equations equivalent to (4a), (4b), (5a) and (5b); but for three phases of costs, operating waiting and non-operating compared with the simple two phases adopted by Hanby and for the analysis in Section 5.5.3 and discussions in Section 5.5.4.

Starting from the same premise - owning and operating costs of operating "team" of a loading equipment item and a truck group of n trucks:

$$C = C_L + n \cdot C_T \qquad (1')$$

Let: c_L = owning and operating cost of waiting loading equipment.

c_T = owning and operating cost of waiting trucks.

And: $r = c_L/c_T$ c.f. $R = C_L/C_T$

Let: k = proportion of productive operating time of trucks, so

1 – k = proportion of waiting time of trucks; and

p = proportion of productive operating time of loading equipment, so

1 – p = proportion of waiting time of loading equipment.

Repeating the analysis in Section 5.5.3 - we arrived at:

$$PS/C_T = R + n \qquad t < n \qquad (5a)$$

$$PS/C_T = (R + n) \cdot t/n \qquad t \geq n \qquad (5b)$$

If equations (5a) and (5b) are developed allowing for waiting time by modifying (4a) and (4b):

$$P = (C_L + k \cdot n \cdot C_T)/S + (1 - k) \cdot n \cdot c_T/S \qquad t < n \qquad (4a')$$

$$P = (p \,.\, C_L + n \bullet C_T) \bullet t/(S \bullet n) + (1 - p)\, c_L \bullet t/S \bullet n \qquad t \geq n \quad (4b')$$

Substituting R for C_L/C_T – i.e., the ratio of owning and operating cost of loading equipment / truck owning and operating cost:

$$PS/C_T = R + k \bullet n + (1 - k) \bullet n \bullet c_T/C_T \qquad t < n \quad (5a')$$

$$n/t \bullet PS/C_T = p \bullet R + n + (1 - p) \bullet c_L/C_T \qquad t \geq n \quad (5b')$$

$$n/t \bullet PS/C_T = p \bullet R + n + (1 - p) \bullet R \bullet c_L/C_L \qquad t \geq n \quad (5b')$$

Substituting in (5a') and (5b'):

$C_T = C_L/R$

$k = t/n$ and $p = n/t$

$c_L/C_L \approx 0.2$ and $c_T/C_T \approx 0.2$ for practical purposes.

Index values from Tables 5.2., and 5.3, for operating time and waiting time justify, for practical purposes, assigned values for cost ratios c_L/C_L and c_T/C_T of 0.2.

For $t < n$, i.e. overtrucked, productivity is controlled practically constant at loading equipment capacity; and truck productivity declines in direct proportion to the degree of overtrucking. For $t \geq n$, i.e., undertrucked, productivity is limited to the capacity of trucks; and loading equipment productivity declines in proportion to the degree of undertrucking.

Equations after substitution and simplification:

$$PS/C_T = R + t + 0.2 \bullet (n - t) \bullet n \qquad t < n \quad \text{Overtrucked} \quad (5a'')$$

$$PS/C_T = R + t + 0.2 \bullet (t - n) \bullet R \qquad t \geq n \quad \text{Undertrucked} \quad (5b'')$$

Tables 5.15 and Table 5.16 provide index values for two examples of t = 5 trucks and t = 7 trucks, both for R values of 1 or 3, over a range of n trucks from under trucking through the match point to over trucking for:

- Productivity indices
- Cost indices for two-phase cost analysis – i.e., ignoring waiting time.
- Cost indices for three-phase cost analysis – considering waiting time.

Analysis summarized in Table 5.15, and Table 5.16, is illustrated by Figure 5.17, for the t = 5 case and Figure 5.18, for the t = 7 case.

There is obvious similarity between the production index trend line in Figures 5.17 and 5.18 and the production profile labeled “Loader Efficiency” in Figure 3.72.

Reflecting on the similarity of equations (5a”) and (5b”), it is manifest that unit load and haul cost is a compilation of:

- Cost of loading;
- Cost of fully utilized trucking capacity (can included part of a truck where t < n, but only an integer number of trucks t ≥ n); and
- Cost of waiting time of excess truck capacity when overtrucked, t < n, and cost of waiting time of excess loading capacity when undertrucked, t > n.

Table 5.15 -Production and Cost Indices – R = 1 & 3, t = 5

	t > n - Undertrucked				n = t	t < n - Overtrucked			
	R = 1				t = 5				
n	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**
Production Index	0.200	0.400	0.600	0.800	1.000	1.000	1.000	1.000	1.000
PS/C_T Index No Allowance for Waiting Time	10.00	7.50	6.67	6.25	6.00	7.00	8.00	9.00	10.00
Production Unit Cost Index - Ignoring Cost Effect of Waiting Time	1.667	1.250	1.111	1.042	1.000	1.167	1.333	1.500	1.667
PS/C_T Index Allowing for Waiting Time	6.800	6.300	6.133	6.050	6.000	6.200	6.400	6.600	6.800
Production Unit Cost Index - Allowing for Waiting Time Cost Effect	1.133	1.050	1.022	1.008	1.000	1.033	1.067	1.100	1.133
	R = 3				t = 5				
	1	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**
Production Index	0.200	0.400	0.600	0.800	1.000	1.000	1.000	1.000	1.000
PS/C_T Index No Allowance for Waiting Time	20.00	12.50	10.00	8.75	8.00	9.00	10.00	11.00	12.00
Production Unit Cost Index - Ignoring Cost Effect of Waiting Time	2.500	1.563	1.250	1.094	1.000	1.125	1.250	1.375	1.500
PS/C_T Index Allowing for Waiting Time	10.400	8.900	8.400	8.150	8.000	8.200	8.400	8.600	8.800
Production Unit Cost Index - Allowing for Waiting Time Cost Effect	1.300	1.113	1.050	1.019	1.000	1.025	1.050	1.075	1.100

Hanby’s analysis focused on values for selected n trucks close to theoretical t. So the bias of inflated cost indices due to ignoring, compared to allowing for, waiting time is not so significant – as illustrated by Figures 5.17, and 5.18. It is concluded that Hanby's comparative analysis is valid within the adopted, limited, range of truck numbers either side of the match point.

Review of Figures 5.17 and 5.18 reveals:

- Production index indicates that productivity controlled by loading equipment utilization is constant when over trucked and diminishes linearly by the ratio of n/t for under trucking.
- Consequently unit cost index is increasing as a linear function of n when overtrucked and increasing as a hyperbolic function of n when under trucked.

The result of these differing functional relationships is that, for n values close to t unit cost indices are lower for under trucking than for over trucking. This is consistent with analysis and conclusions when developing the truck selection rounding protocol.

Table 5.16 -Production and Cost Indices – R = 1 & 3, t = 7

	t > n - Undertrucked						n = t	t < n - Overtrucked					
	R =	1				t =	7						
n	1	2	3	4	5	6	7	8	9	10	11	12	13
Production Index	0.143	0.286	0.429	0.571	0.714	0.857	1.000	1.000	1.000	1.000	1.000	1.000	1.000
PS/C_T Index No Allowance for Waiting Time	14.00	10.50	9.33	8.75	8.40	8.17	8.00	9.00	10.00	11.00	12.00	13.00	14.00
Production Unit Cost Index - Ignoring Cost Effect of Waiting Time	1.750	1.313	1.167	1.094	1.050	1.021	1.000	1.125	1.250	1.375	1.500	1.625	1.750
PS/C_T Index Allowing for Waiting Time	9.200	8.500	8.267	8.150	8.080	8.033	8.000	8.200	8.400	8.600	8.800	9.000	9.200
Production Unit Cost Index - Allowing for Waiting Time Cost Effect	1.150	1.063	1.033	1.019	1.010	1.004	1.000	1.025	1.050	1.075	1.100	1.125	1.150
	R =	3				t =	7						
n	1	2	3	4	5	6	7	8	9	10	11	12	13
Production Index	0.143	0.286	0.429	0.571	0.714	0.857	1.000	1.000	1.000	1.000	1.000	1.000	1.000
PS/C_T Index No Allowance for Waiting Time	28.00	17.50	14.00	12.25	11.20	10.50	10.00	11.00	12.00	13.00	14.00	15.00	16.00
Production Unit Cost Index - Ignoring Cost Effect of Waiting Time	2.800	1.750	1.400	1.225	1.120	1.050	1.000	1.100	1.200	1.300	1.400	1.500	1.600
PS/C_T Index Allowing for Waiting Time	13.60	11.50	10.80	10.45	10.24	10.10	10.00	10.20	10.40	10.60	10.80	11.00	11.20
Production Unit Cost Index - Allowing for Waiting Time Cost Effect	1.360	1.150	1.080	1.045	1.024	1.010	1.000	1.020	1.040	1.060	1.080	1.100	1.120

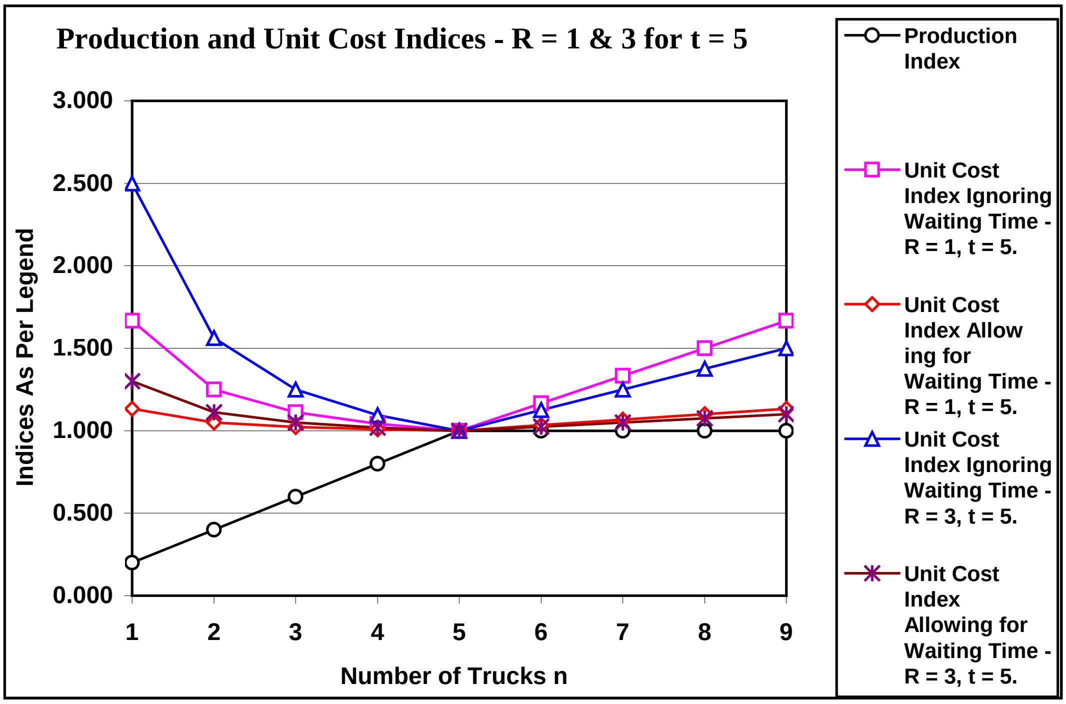


Figure 5.17 Production and Unit cost indices – R = 1 & 3 for t = 5
From Table 5.16

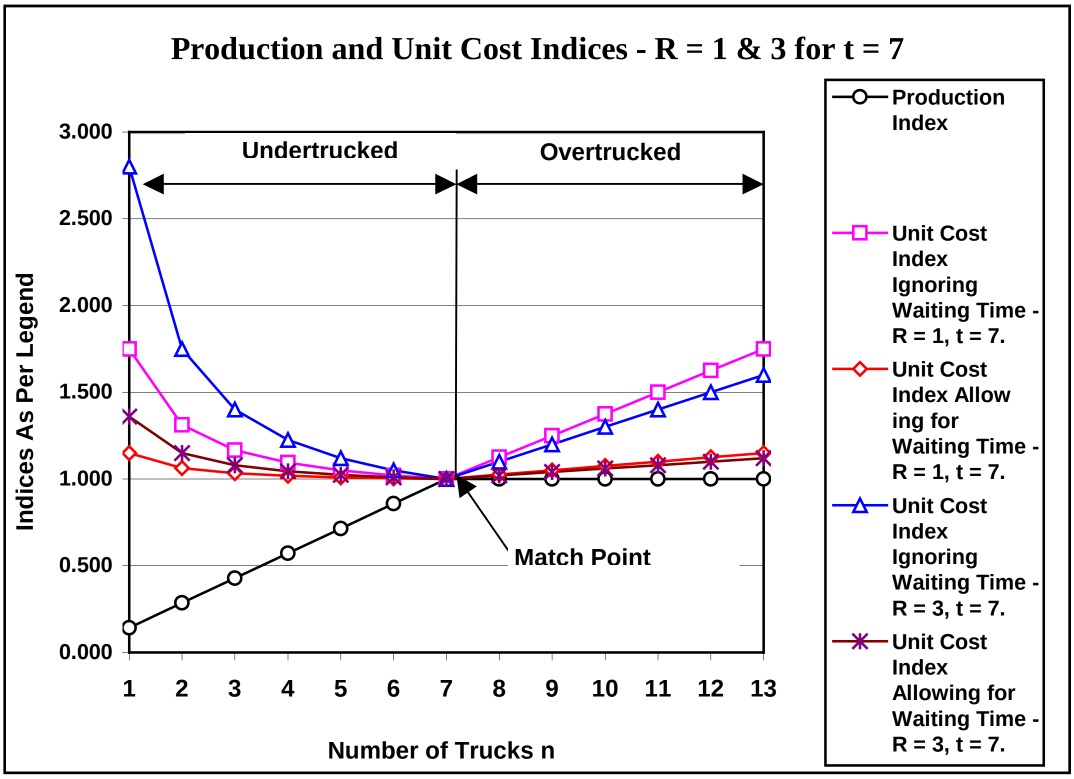


Figure 5.18 Production and Unit cost indices – R = 1 & 3 for t = 7
From Table 5.16

When under trucked, the unit cost of loading in the three-phase-cost case, where allowance is made for cost of waiting time for loading equipment, is the same as for fully utilized loading capacity at the match point or when over trucked. Reduced utilization of loading equipment is offset by reduced production in proportion to number of trucks selected, which is less than theoretical trucks required for a match.

So the loading component of load and haul unit cost is practically constant.

A similar situation applies to hauling unit cost of fully-utilized selected trucks n when hauling unit cost for the consequent reduced production is the same as the hauling unit cost of theoretical trucks t at the match point.

When over trucked, excess truck capacity is waiting and adds a commensurate hauling unit cost. When under trucked, excess loading capacity is waiting and similarly adds a commensurate loading unit cost.

Cost amelioration effect of considering waiting time is clearly demonstrated by Tables 5.15, and 5.16; also Figures 5.17, and 5.18. The inflated cost bias of analysis that ignores reduced cost for waiting time is more obvious as n values increase or decrease from the match point.

Observations and conclusions from review of Tables 5.15 and 5.16; and Figures 5.17 and 5.18 include:

- Under trucking to any degree consistently increases load and haul unit costs less than does the same degree of over trucking – so favouring under trucking where the production programme can tolerate the small reduction.
- As expected, for increasing numbers of theoretical trucks, each additional over match; or omitted truck under match, has less load-and-haul-cost impact – compare Tables 5.15 and 5.16.
- As over trucking or under trucking increases, the “B” case (t = 7) indicates similar bias gradients as does the “A” case (t = 5) – refer Tables 5.15 and 5.16; but index gradients for the “B” case (t = 7) are significantly flatter than for the “A” case (t = 5) – consistent with the point immediately above.
- Subject to the discussion on Owning and Operating Costs below, actual over trucking or under trucking is less cost penalizing than simplified analysis

based on simple two-phase cost analysis, ignoring waiting time effects, would lead us to believe.

In the end result – over trucking or under trucking imposes cost penalties much less than those indicated by two-phase cost analysis as adopted by Hanby; also by the analysis herein in Section 5.5.3.

Simply, adding trucks that are not utilized only adds the cost of labour plus some 5% of productive-operating fuel consumption and lubricants and a similar proportion of driveline maintenance, investment amortization, and no tyre costs.

Similarly, under trucking underutilizes loading equipment. Productive time of loading equipment, during which unit loading costs are generally constant, is reduced. The balance, loading-equipment waiting time, imposes a cost similar in compilation to waiting trucks. That is, cost of labour plus some small proportion; say in the order of 5%, of energy/fuel consumption and lubricants, a similar proportion of maintenance and investment amortization; and no ground engaging tools, running gear, bucket repairs or digging implement structural/mechanical maintenance cost.

5.5.5.2 Owning and Operating Cost – Discussion

The above analysis adopts basic assumptions for the derivation of owning and operating (O & O) costs for mining equipment items per unit of time (solar hour or SMU). It is assumed that O & O costs are based on productive operating hours. That is after all deductions for downtime and for non-productive time whether or not deemed necessary for productive operations; and further, after allowance for job management efficiency factors or deductions. This procedure allows for cost analysis to separately and transparently account for waiting time, however caused, which clearly indicates reduced O & O costs.

Table 3.75, appended in Volume 2, details time definitions that provide an understanding of the implications of differing O & O costs for available equipment that is not directly operating, i.e., on the job but not directly productive.

5.5.6 Bunching - Productivity and Costs

The efficiency-reducing effect of bunching was discussed generally in Section 3.5.4.

5.5.6.1 Derivation of $\eta_B = (1 – CV_T)$

As discussed in Section 3.5.4, as reported by Caterpillar, empirical evidence indicates that if:

$CV_T = 0.15$ $\eta_B = (1 – 0.15) = 0.85$ or generally:

$$\eta_B = (1 – CV_T) \quad \text{(15 - see Section 3.5.4)}$$

Where:

CV_T = Coefficient of variation of truck cycle time (total trip time T_T herein).

η_B = Bunching efficiency.

As discussed in Section 3.5.4, and based on empirical evidence, Caterpillar have recommended a practical range of bunching efficiencies from a high of 0.90, through medium of 0.85 to a low of 0.80.

Bunching is a significant cause of T_W, i.e., a small proportion of truck total trip time T_T. Variability, i.e., CV_T – for total trip time T_T, will always be less than CV_W for truck (or loading equipment) waiting time T_W. Parallel logic is applied here as for reasoning discussed in Section 3.5.4 for truck payload and NMW as proportions of GMW, in Section 3.2.8 for bucket loads and truck payloads; and in Section 3.2.9 for bucket cycle times as a proportion of truck loading time.

Bunching is a physical manifestation of variability of performance of the entity under review. Queuing effects, measured by waiting time T_W and number of trucks in the queue are caused by bunching. Causes of bunching were discussed in some detail in Section 3.5.4. Waiting time T_W is mainly due to variability of total truck trip time T_T. That T_T is a combination of several time identities discussed above – specifically $T_T = T_L + T_V + T_D + T_S$ – is irrelevant for purposes of the current discussion. For convenience the following rationale adopts total trip time T_T as a basis. Outcomes are consistent with empirical-based equation (15) above, bunching-effect discussion in Section 3.5.4 and in this section. Adopted data following is also consistent with analysis and references identified in Section 3.3.10.

It is an important distinction that truck waiting time, i.e., hauling time lost directly related to bunching, can be generally classified as intrinsic. That is time losses from inherent variability of the several components that combine to determine equipment performance as discussed in Section 3.5.4.

Non-intrinsic losses, generally an outcome of operational misadventure, unintended interruption to operations by influences external to direct capability or operation of equipment. As discussed in Section 3.3.10 non-intrinsic time losses must be treated separately to ameliorate or eliminate their influence on productivity.

Rationale:

The following rationale is offered as confirmation of equation (15) in terms of statistical analysis:

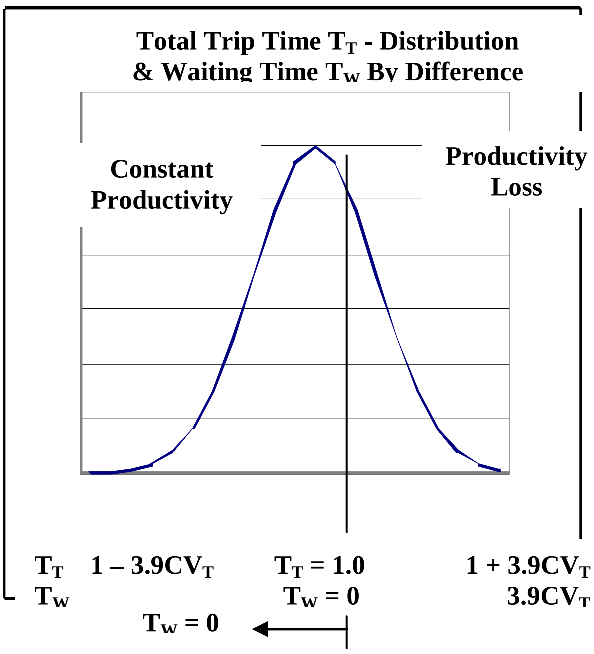
General Case: $\sigma_T = CV_T \bullet \text{mean } T_T$

Assuming a normal distribution for truck trip time and waiting time (this discussion is based on intrinsic elapsed time for variables – nett of all non-intrinsic time events that interfere with load and haul operations - analysis in 3.2.9 and 3.3.10 indicates this assumption to be reasonable):

$$\text{Range } T_T \approx +/- \ 3.9 \bullet \sigma_T \ (+/-3.9\sigma \text{ for } P_{0.9999} \text{ – i.e., 1:10,000})$$

$$\text{Range } T_T \approx +/- \ 3.9 \bullet CV_T \bullet \text{Mean } T_T \qquad (1)$$

Confining consideration only to the lower-than-mean proportion of the waiting time distribution – higher-than-mean occurrences do not materially affect productivity until the match point is passed -:



Notes: Figure 5.5.6.1A:

Range $T_T = 1 - 3.9CV_T.T_T$ to $1 + 3.9CV_T.T_T$

By difference:

For Productivity Loss only -

Range $T_W = 0$ to $+ 3.9CV_T.T_T$ (2)

Mean $T_W = +1.95CV_T \bullet T_T$ (3)

Equations (2) and (3) provide estimates of normal distribution parameters for T_W

Figure 5.19 Total Trip Time T_T – Distribution & Waiting Time T_W By Difference

Waiting time T_W is a random variable dependent mainly on travel time randomly variable travel time T_V. As loading time and other components are small compared with T_V their variability may be ignored, i.e., these elemental times can be considered constant (expected) values.

For fleet arrangements where loading equipment is theoretically under trucked or over trucked by less than a unit truck. Some of the time bunching may reduce conditions to loading equipment being undertrucked so productivity reduces. For the balance of the time over trucked conditions will prevail. But this only results in additional waiting time for trucks only and no material change to productivity.

Because total trip times exhibit central tendency, it is reasonable to assume that waiting time data will be symmetrical. It is reasonable to assume that for practically half of the time, there is a tendency for loss of productivity; and for the other half of the time, constant productivity.

The rationale is illustrated by Figure 5.19 and related notes.

From equation (3) – for some 50% of operating time:

$$\text{Mean } T_W = 1.95CV_T \bullet T_T$$

$$\text{Apparent } \eta_B = (T_T - T_W)/T_T$$

$$= (T_T - 1.95CV_T \bullet T_T)/T_T$$

$$= (1 - 1.95CV_T)$$

This is valid if all waiting events can affect production – consistent with the high end of the range evidenced empirically. But a-productive effect of waiting time only affects some 50% of waiting time events.

$$\text{Mean } \eta B = [1 + (1 - 1.95CV_T)]/2$$

$$= (2 - 1.95CV_T)/2$$

$\eta B \approx 1 - CV_T$ practically corresponds to empirical equation (15).

The above rationale is considered to analytically support empirically based bunching efficiency variability recommended by Caterpillar and generally adopted by industry.

It should be noted that the above relationship between bunching efficiency and distribution parameters of truck trip time T_T is independent of the origin of data, i.e., whether empirical (inclusive of non-intrinsic time losses), simulated as intrinsic or estimated.

Caterpillar observed empirical values of CV for T_T ranging from 0.1 to 0.2 and apparently made the connection to bunching efficiency from empirical data. Empirical observations during the research for CV of T_T of 0.19, reducing down to 0.12 when filtered to emulate intrinsic residuals; and used for analysis in Tables 3.60 and 3.61, are reasonably consistent with Caterpillar's observations.

Three cases are considered below:

(1) $CV_T = 0.1$: (2) $CV_T = 0.15$: (3) $CV_T = 0.2$.

5.5.6.2 Productivity and Cost Implications

Bunching effect increasing waiting time of trucks to the extent of apparent over trucking does not significantly affect loading productivity. Only truck utilization reduces, increasing the hauling cost component of unit load and haul costs similar to the circumstances for selecting additional trucks to over truck as described in Section 5.5.3.

Bunching efficiency η_B applies similar to an equivalent degree of undertrucking. Referring to Tables 5.15 and 5.16, interpolating as necessary, Table 5.17 provides analysis of Cost Index increases allowing for waiting time resulting from bunching.

Table 5.17 Bunching Effect Parameters for Mining Trucks

η_B	**0.90**	**0.85**	**0.80**
Trucks:			
5 Truck Case – t	4.5	4.3	4.0
Productivity Factor	0.9	0.85	0,8
Cost Index – Allowing for waiting time.	1.010	1.015	1.019
Cost Index – Ignoring waiting time.	1.047	1.070	1.094
7 Truck Case - t	6.3	6.0	5.5
Productivity Factor	0.9	0.85	0,8
Cost Index – Allowing for waiting time.	1.007	1.013	1.018
Cost Index – Ignoring waiting time.	1.035	1.057	1.078

Treatment in the process of equipment selection is described in Section 5.5.4 and generally requires:

- Completion of virtual truck selection to satisfy theoretical truck requirements in accordance with criteria discussed in Section 5.5.3 and Section 5.5.4.

- Consideration of operating conditions, load and haul equipment, operating configurations as planned and haulage conditions.
- Other considerations as discussed in Section 3.5.4 or Section 5.5.4.

In addition bunching productivity and cost effects need to be analyzed in accordance with the above discussion in this Section 5.5.6 to fine-tune the fleet for acceptable productivity at optimum cost. The nett result of bunching considerations is generally an increase in the basic number of trucks as delivered by initial estimates. Caterpillar's FPC and similar load and haul simulation software apply bunching treatment integral with the simulation applications.

CHAPTER 6

SUPPLEMENTARY ACTIVITIES

6.1 ADDITIONAL INTER-ACTIVES

This section briefly reviews additional inter-actives that overlap, or interface with, the activities reviewed and analyzed in Chapters 3, 4 and 5. These supplementary topics include recently emerging and developing activities that promise to significantly influence future development of load and haul practices in open pit mining.

The general approach of the research has been for improvement of understanding of the basic selection criteria for load-and-haul equipment. These additional topics have not been thoroughly investigated for this thesis. They have been identified in Chapter 4; related productivity and cost effects have been discussed in Chapter 5; are further discussed in the context of application in this section; and finally considered in Chapters 7 and 8 in the context of conclusions and recommendations for future research.

During research of criteria for selection of mining equipment each topic was analyzed to a degree of understanding, where that topic could be considered to have met research objectives. In common with research in general, as research proceeds, other inter-active, and supplementary topics emerge of sufficient relevance to encourage consideration of inclusion in the research. Because of scope limitations, determined by time and sheer bulk of research undertaken, some topics, investigation of which appeared to be warranted, were excluded to provide priority for those topics researched as described in this thesis.

Some of the more cogent complementary topics have been identified and briefly discussed in Chapters 3 to 5. Additional topics, previously bypassed, are included in the following discussion, at least to a preliminary degree sufficient to recognize their importance as inter-active to the topics covered in more detail by the research.

This Chapter 6 reviews and briefly discusses those topics that, although interrelated to primary activities researched, have not been investigated in detail.

6.1.1 Planning and Equipment Selection

In the broad activities of promoting a mineral prospect to an economic development project, there is a succession of iterative processes that can generally be described as project planning. The many disciplines applied to bring a development project to reality each need to make important planning contributions. Integral with general planning activities, including resource interpretation and delineation and ore reserve definition, a number of activities, generally termed Mine Planning, must proceed. Mine Planning not only involves consideration of exploitation methods but must also consider operating scale. This leads to mining equipment selection.

Equipment selection, at least of principal items of loading and hauling equipment, is considered early in mine planning. The interdependency of mine planning and equipment selection is such that neither can be completely finalized without complete resolution of the other – so requiring iterative process. Equipment selection is carried forward integral with an ever-refining process through the many planning iterations that yield finalized cost-optimized production schedules and mine plans, estimation of costs, economic evaluation and feasibility testing.

Along the way changing, but continuously refining criteria for equipment selection must be gathered, both internally from project planning and externally by due diligence, to identify options available and characteristics of those options to form a basis for decision.

The following four sections discuss, albeit briefly, some of the more cogent, supplementary topics that interface with or overlap the research described in this thesis.

6.1.2 Drilling and Blasting

Section 3.3.6 describes the analytical trail from truck payload dispersion to the drill-and-blast quality control. The implication that control of truck payload dispersion may most effectively be realized by best-practice drilling and blasting is an obvious conclusion needing verification empirically. The promise for payload dispersion control justifies the general review and comments in this Section 6.1.2.

Drilling and blasting parameters can be considered in two groups:

1. Blasting agent, explosives – specific quantities measured as "powder factor", generally kilogrammes/bank cubic metre; blasting accessory specifications and characteristics, hole charging details including stemming – and cost per unit volume or tonne.
2. Blast design, bench height, hole diameter and drilling pattern dimensions and parameters that determine the distribution of blasting agent or explosive through the material volume that is to be fragmented by blasting; also the detonation sequence that determines material displacement, seismic vibration and noise level of blasts.

Within these two groups activities and attributes related to quality control of blasted material are of greatest interest for these discussions. Except for unconsolidated highly-weathered materials – generally materials that can be extracted by so-called "free digging" - such as detritals and tertiary sediments including wind or water deposited sands and gravels, bank material, i.e. insitu material, generally needs fragmentation in preparation for loading. Materials to be loaded from the sub-weathered zones, i.e., particularly "fresh" or "transition" horizons, need to be drilled and blasted. Variabilities of truck payloads, loading equipment-digging performance, especially bucket fill factor; and consequently the number of passes are all influenced by quality of fragmentation of material measured by maximum particle size, size grading and voids ratio.

For cyclic loading there is a cost tradeoff where the total cost of fragmentation, plus cost of loading and hauling is a minimum.

The relationship between variabilities of truck payload and voids ratio of material, fragmented (generally by drilling and blasting) in preparation for loading, was investigated and discussed in Section 3.3.6. Experience indicates that, where fragmentation produces smaller mean particle size and more consistent size grading of fragmented material, expected or enhanced digability is achieved. The relationship between particle size and fragment grading and drilling and blasting design parameters has been established. Therefore, it is hypothesized that a relationship between fragmentation quality, specifically controlled voids ratio variability, and truck payload variability, loading and hauling productivity and costs can be defined in mathematical terms and substantiated empirically.

Considering two operating scenarios:

1. Small drilling patterns, small diameter holes with adequate powder factor (kg/BCM of insitu material) will produce reduced particle sizes, i.e., high fragmentation. But this could be at prohibitively high cost that, when added to cost of loading and hauling (as productive and load-and-haul cost effective as this may be) will be uneconomically high.
2. Conversely, given that bench height is adequate, large dimension patterns with large holes at the same powder factor may cost less to provide the blast holes, reduce cost of blasting accessories so reduce drill-and-blast costs; but at the sacrifice of digability – decreased loading productivity, low bucket factors, increased passes, increased payload dispersion and overall reduced load and haul productivity and higher costs. Time losses, inconsistent loading times and truck cycle times will exacerbate bunching/queuing and erode operational "rhythm". Since load-and-haul costs are three or more times drill-and-blast costs, evidenced by Figure 2.2, the total of drill-and-blast costs and load-and-haul costs could tend to be potentially uneconomic.

Between these two extremes there will be an optimum operating position where the total cost of drill-and-blast and load-and-haul is a minimum. Comparing the continuum of positions in terms of cost, using mean voids ratio and/or coefficient of variation of voids ratio as characteristic parameters will produce a typical "bathtub" curve. At the "nadir" of costs vs. mean voids ratio all underlying parameters throughout the drill-and-blast and load-and-haul system will combine to produce optimum (not necessarily maximum) productivity and minimum cost.

6.1.2.1 Bench Height – General Comments

For mining of some materials bench height can be predetermined by natural features. Examples are coal seam width, flat dipping deposits such as sedimentary horizons including mineral and tar/oil sands. Ore dilution considerations may determine bench height in metalliferous open pit mines.

For successful, quality drill and blast results drilling patterns, drillhole diameter and bench height and other drill and blast parameters must satisfy the principles of blast design. A desire to increase drillhole diameter and pattern size drives the need to increase bench height. Drill holes must be deep enough to provide adequate stemming height and efficient fragmentation. But, as drillhole diameter reduces, bench height does not necessarily need to reduce. An interesting example is basic materials quarrying where high benches with small diameter holes and small-dimension patterns are general practice for producing blasted hard rock to be crushed and sized for aggregates. For these special design criteria reduced crushing and screening costs offset increased cost of drilling and blasting.

Where continuous improvement programmes (CIP) investigate all cost components, the author's experience indicates that drill and blast costs tend to be an easy target. Although drilling and blasting contributes only some 15% of total mining costs (Figure 2.2), significant savings can be realized by increasing bench height, drillhole diameter and pattern dimensions, with little if any change in powder factor.

There has been a trend for increased bench heights and drillhole diameters, with appropriate adjustment of other blast design parameters, for drilling and blasting practices in metalliferous mining. It is not clear that any resulting load-and-haul productivity reduction or increased dilution effects have been fully accounted for when assessing the apparent drill and blast savings. This is a classic case of considering an activity in isolation with apparent attractive outcomes that are not realized when all interactives are effective – as discussed in Section 2.2.1.

6.1.3 Dumps and Stockpiles

Selection of trucks both for capacity and supplier is determined by the service required in terms of productivity and costs. One-way haul routes throughout the life of the mine, at least the tenure of any selected mining method, need to be determined; and productivity/cost implications identified and quantified. Production schedules that identify destinations and target quantities per unit time are required.

As described, during the research deterministic simulation of hauling operations provided data to progress loading and hauling equipment selection. An obvious prerequisite is the location, dimensions and limitations on placement of both product and waste material produced. This requires design of dumps and stockpiles complementary to open pit design.

Continuously improving computer techniques have been developed for cost-optimized pit designs. In contrast dumps and stockpiles have traditionally been designed to fill available spaces within tenements, sometimes with height limitations, using simple manually applied geometrical procedures. More recently fundamental cost-element techniques applied to pit optimization have emerged in computer applications for dump design.

The research did not undertake any major treatment of stockpile and waste dump investigation, analysis or design. Some practical issues are identified with comments on this reasonably important subject that, in the author's opinion, has historically tended to become a secondary planning consideration, often neglected, which can ultimately impose an embarrassing production and cost penalty.

The following checklist with comments provides preliminary considerations for planning stockpiles and dumps:

- Tenement boundary location by independent land survey should be available supplying sufficient boundary markers to set out site works including clearings, topsoil stripping and stockpiling, tailings dams, dumps and stockpiles, explosives storage all buildings and constructions, and processing facilities, sources of basic civil construction materials, mine water evaporation ponds; and garbage/sewerage disposal.

- Half-metre (or closer) surface contour plans of mining tenements, at least the areas with potential for occupation, to identify drainage courses and catchments, to plan diversion of drainage channels, flood protection bunds and dams as appropriate; and to provide a base digital terrain model for determination of volumes of dumps and tailings dams.

- Sterilization drilling and geological interpretation of all areas that could possibly be inundated by stockpiles, dumps and tailings, or occupied by mine infrastructure.

- Determine general details of dump design, including height limitations, finished batter profiles for long term stability, surface drainage control facilities, rehabilitation - including re-vegetation details consistent with permit conditions on tenements, requirements or recommendations of statutory authorities and general best practice for dump/stockpile construction.

- Determination of setback from the open pit rim of waste dump or stockpile of materials for future processing, civil works and rehabilitation; allowances to include for safety bund wall, potential long term subsidence – particularly in weathered/transition zone – in compliance with statutory requirements and as more conservatively amended by competent geotechnical assessment; and to allow for potential pit deepening when resources, excluded from current pit designs as uneconomic, can be upgraded to proven ore reserves; also sufficient additional setback to allow for re-profiling as-dumped waste batters to conform to a long-term stable angle of repose as required or recommended by statutory authorities.
- Design dump access ramp(s) to provide optimum haulage costs after mining trucks reach the surface crest of in-pit access ramps including, generous ramp width, allowance for drains and spillage storage, straight ramps where possible and, where necessary for high speed truck return, bends to have large radius and adequate camber(?); and, where possible, a single constant gradient from toe of the dump access ramp to crest at the top of the dump.
- Design dump-access ramps to crest at the centroid of dump surfaces. If dumps are elongated, to match a long-narrow open pit form, then multiple dump access ramps should be provided to match multiple open pit access ramps in the early stages of open pit development to divide waste dumps into sections with dimensions to minimize surface haul distances. Where, in the mature period of pit development (say below mid-design depth so that some 60% or more of total material has been mined), where access may be limited to one ramp then dump management should provided space to accommodate the residual pit volume delivered from the final access ramp at minimum haulage cost.
- Dump construction needs to be scheduled so that, as dumps extend laterally, additional lifts are implemented as soon as practicable to provide haul-distance options for tuning of trip time T_T to available trucks.

Further comments on practices on top of waste dumps:

- Crests of access ramps should preferably be located near the geometrical centroid of the dump surface for minimum on-dump haulage. Average on-dump distances hauled are then to centroidal positions - of sectors for circular-form dumps with average on-dump haul 0.71 times dump radius and of triangles for square or rectangular form dumps with average on-dump haul 0.67 times the rectangular distance to the dump crest. So, practically, on-dump haul distances can be averaged as 70% of average distance to the dump edge from the access ramp crest.

- On-dump roads should fan out from the access-ramp crest built on a formation above dump surfaces to provide for cross fall and road drainage; also roads should be to best construction standards commencing from compaction of the dump surface (loaded trucks are convenient compactors), a selected-compacted formation, preferably with crushed rock sheeting to provide an all-weather hauling surface of profile quality suitable for high speed return hauls.
- Temporary extensions to tipping points of roads from termini of long-term on-dump roads should be kept as short as practicable.
- As soon as possible after commencement of mining it is good practice to mark out dumping limits with rows of truckloads dumped at ground level to serve as more permanent markers than survey pegs and similar temporary markers. These marker loads also act as “toe checkers” tending to stabilize slopes where material tends to flow to a flatter-than-expected angle of repose; also “toe checkers” catch most large particles that become free on dump batters tending to build up rolling inertia.
- Before each lift, preferably progressively as dump surface develops, it is good practice to loose dump loads on the dump surface. This provides convenient and safe dumping, especially at night. The next lift is then dumped over the loose dumped “moonscape”. There is a secondary benefit of reducing the wind fetch on dump surfaces, encouraging turbulent rather than laminar flow and reducing dust generation.
- It is beneficial to finish final dump surfaces with loose dumped loads to reduce dust generation and contain precipitation until dumps can be rehabilitated and finished with topsoil and seeding to promote vegetation.

- A comprehensive rehabilitation/vegetation protocol needs to be developed complying with any statutory requirements. Such a protocol will generally be a prerequisite to acquiring tenure of the property and/or permission to develop. Implementation of rehabilitation provisions should commence as soon as dumps are part completed as per plan with due conservative consideration of risk of an extension to mining and waste disposal.

The described practices are important for acceptable hauling conditions and best-practice mine management. Optimum truck cycle time is interrelated with ore stockpile and waste dump configuration. Inadequate attention to the detailed design of load-and-haul payload destinations has negative effect on mining productivity and costs. Engineering and management of dumps and stockpiles is a stand-alone activity for investigation/research, project planning and mine development.

6.1.4 Mixed Fleets

Many load and haul operations in open pit mines have mixed loading equipment and mining truck fleets. This may be by design where ex-mine production consists differing materials such as coal, partings and superincumbent overburden; or by discrete increases or decreases to operating production scale. Destinations and one-way hauls can be different for mining product for processing and superincumbent waste material. So there can be valid operating reasons for mixed fleets. It should be noted that OEM produce mining trucks with generic performance characteristics to reduce the affects of mixed fleets with differing performance attributes. But there is always small, but significant differences in the performance of equipment of differing scale and between similar capacity equipment produced by different OEM.

Voluntarily acquiring mixed fleets to load and haul material of similar physical nature over similar round-trip hauls will generally be disadvantageous by reducing productivity and increasing costs. Where there is choice, avoiding mixed fleets is the best commercial decision.

As discussed in Chapter 3 loading and hauling with mining trucks is a cyclic rather than continuous production system. The cyclic nature with tendency to variability of equipment performance and productivity introduces a-productive effects.

Differing performance of loading equipment and trucks is increased by:

- Mismatch between loading-equipment bucket capacity and truck payload.

- Discrete time step variability in truck loading tending to increase queuing effect.
- Potential differing speeds of trucks so that normal T_T variability and bunching inefficiency are increased by additional time loss.
- Increased inventory of maintenance parts and exchange components (insurance spares).
- Maintenance facilities may have to be expanded to suit largest equipment – with some facilities limited in utility; also single or limited occupancy facilities such as wash-down pads, routine service bays and tyre bays need to be for the largest equipment.
- Adverse operator and maintenance attitudes to individual equipment items perceived as less operator/mechanic friendly.

Higher capacity equipment tends to be inhibited by lower performance loading equipment and trucks with a tendency towards the lowest common denominator, at least lower than expectancy based on average performance over the mixed fleet.

6.1.5 Dispatch Systems and Remote Control

In the early 1990's dynamic programming (DP) of complex production systems enabled development of computer applications for operational control of open pit mining. Currently larger open pit mining operations with multiple load and haul fleets generally utilize a "dispatch system" (DS) applying DP algorithms in sophisticated information technology and communications systems.

DS applying DP can only be as effective as the comprehensiveness of the algorithm in representing specific DP and degree to which logic of the underlying DP emulates the actual system. From some observations of DS in action with dispatch attendants overriding the DS when they consider it necessary, it seems to the author there is a strong possibility that DP and DS are not always superior to the human mind – especially the consolidated power of several human minds all seeking to sustain "rhythm" of loading and hauling operations by minimizing waiting time and variability of T_T and its parts, T_L, T_V, T_D and T_S (as defined in 3.3.10).

It seems possible that the benefits of dispatch systems can be enhanced by some old-fashioned rhythmical practice, especially a bucket-load-sacrifice protocol as discussed in Sections 3.3.9 and 5.4.3 and hauler operators in each fleet keeping aware of passing points and whether they are ahead or behind time in the circuit, identifying the slowest truck, advising supervision and replacing a slow truck for maintenance attention if a spare truck is available. There must be a economic limit where the delay of a slow truck becomes unacceptable. A simple answer could be where the proportional delay of total trip time is equal to the inverse of the number of trucks attending an item of loading equipment. In this circumstance effective truck time lost is equivalent to an effective truck. These were all practices utilized on sites managed by the author in the 1970's before PC, DP and DS were available. Two-way radios were often a luxury in those times. Load-and-haul "rhythm" was often in evidence, especially on night shifts where strategic planning and pit "housekeeping" during the day set up the pit for minimum queuing delay of loading and hauling equipment.

Unless DP is heuristic, with some facility for recall of experience, application of a fixed logical process reflecting DP may be applied by a DS to a fault - requiring constant human monitoring and, occasionally, intervention as necessary.

The above comment is not intended as a criticism of DP and DS applications. A DS can only be as effective as the underlying DP emulates the operating system. But current state-of-the-art DS is a most welcome acquisition for open pit management. Dispatch systems are amongst the most valuable recent developments in open pit mining management, with the bonus of facility to collect and analyze data and provide directions - all in real time. They also provide performance data in real time and can record almost any selected performance indicators to aid management oversight; and to measure performance against plans and projections. A DS is a necessary foundation for development of autonomous mining systems. Control of autonomous trucks can adapt current DS technology, retaining all of the initial benefits described in Section 4.2.7, and facilitating autonomous hauling operations. Future development of DS will parallel development of autonomous systems.

Currently development of remote control of DS off mine sites offers flexibility and convenience for specialist labour to manage logistics of mining operations from city locations. All mines within convenient geographic boundaries could be controlled from a specific-purpose facility. The savings of accommodation capital and/or fly-in-fly out transport costs are manifest.

6.1.6 Autonomous Haulage Systems

"Autonomy" implies complete freedom to function of the system's own freewill, i.e., without human intervention. But, in these early stages, development of fully-autonomous systems with complete freedom of solely self-determined operation seems ambitious. At least, in the initial stages, constant human monitoring and intervention seems to be a practical expectancy. As refinement of autonomous systems with time encourages increasing confidence by operators and clients, degree of autonomy and remoteness of controlling facilities can be expected to increase. This is the promise of the future of DS and autonomous load-and-haul, even complete mining systems. But it would be pure speculation to predict a time scale for application of completely autonomous mining systems with only small management intervention.

Some practical parallels provide interesting insight. Plans to automate the railway systems of the Pilbara iron-ore province in the north-west of Western Australia have been only modestly publicized. Predictable negative industrial reaction, and virtually silent political response, is an indication that, even though the technology has been demonstrated to be reliable and safe; and is readily available, implementation will be steady rather than startling.

Another parallel is the potential for air transport to graduate to autonomous systems. Developed for military pilotless aircraft, technology for safe autonomous control of the world's airliners is readily available; but there are few passengers who would fly in them.

Recent development and trials of autonomous load and haul equipment, especially trucks, to the best of the author's knowledge, have not been publicly reported. With performance and cost information generally limited to hearsay, this topic has not been researched for this thesis. It is understood that economics of early trials were obscured by the early-developmental cost of the truck add-ons and that early control applications required remote operator attendance in a purpose built facility on site. As the cost of truck control units reduces, when control centres are set up in lower-cost environments remote from mine sites, with more reliability and increased confidence in the systems autonomous haulage systems appear to be a natural progression for development of open mining technology. Certainly this is a cogent opportunity for future research.

It is understood that economics of prototype trials were obscured by the early-developmental cost of the truck add-ons; and that early control applications required remote operator attendance in a purpose built facility on site. As the cost of truck control units reduces, when control centres are set up in lower-cost environments remote from mine sites, with more reliability and increased confidence in the systems, autonomous haulage systems appear to be a natural evolution for development of open mining technology.

It is interesting to reflect on the interrelationship between equipment selection criteria and characteristics of autonomous haulage systems. In Sections 4.2.2, 4.2.3 and 4.2.4 the diminishing unit cost-benefits of trucks of increasing scale was reviewed. The cause of diminished unit cost-benefit was identified as off-setting costs mainly for increased access-ramp volume and increased scale of maintenance accommodation.

Operating labour, considered as a “fixed” cost, unitized over truck payload contributes to reducing unit-haul costs as truck scale increases. Section 4.2.5 indicates the practically uniform unit cost of fuel and uniformity of other unit-cost components that are directly proportional to fuel burn. This implies an underlying practically constant unit cost over the payload range for many of the haul-cost components. In contrast operating labour, in the order of 12 % of unit haulage cost (Section 5.1.3), provides a diminishing-unit-cost effect as truck payload increases. Diminishing unit cost of operating labour is a significant contribution to the perceived unit-cost benefit of up-scaling to larger mining trucks.

It is interesting to consider the impact on unit costs by substantially removing operating labour cost for an autonomous haulage system. Removing the majority of labour cost will tend to have a retrogressive effect on justification of larger payload and ultra class (UC) mining trucks. Additional flexibility may be available to consider selection of trucks with a wide range payload without significant cost penalty. This leads to a further possibility that truck selection will be driven by comparison of other factors related to most convenient fleet number logistics.

The potential commercial impact on UC truck options is manifest.

A further benefit from autonomous haulage could be the potential for rhythmical smoothing of truck cycle times i.e., virtual elimination of bunching and resulting queuing effects. Autonomous haul systems need to control separation between trucks for safe operations. It seems a small advance to control separation between trucks to minimize (practically eliminate?) bunching with flow-on benefit of reduced fuel and other consumable costs.

It seems only another small step for an autonomous control system to identify a slow truck, compare system productivity with or without the offending truck, register whether or not a replacement is on the “Go” line and make a decision to either leave the offender in the fleet or deliver it to the workshop complete with complaint diagnostics in real time on the remedial work required – all without human intervention.

Certainly autonomous mining systems are a cogent and most interesting field for future research.

CHAPTER 7

SUMMARY OF RESEARCH

7.1 CRITERIA INVESTIGATED

Productivity and cost criteria investigated as described in Chapters 3, 4 and 5 include:

- Truck payload distributions and implications for component life, safe operations and compliance with limitations on operation of mining trucks.
- Bucket cycle time, number of passes, consequent truck loading time including intrinsic efficiency and operating "rhythm" benefits in total truck trip times by limiting non-intrinsic time losses.
- Accuracy and reliability of methods of measuring truck payloads.
- Perceived focus on productivity with optimum unit-cost realization an apparent secondary objective.
- Truck payload variability that is seen as largely dependent on condition of material to be loaded, with loading techniques and number of loading passes having lesser influence.
- The *de facto* standards of three or four-pass loading is questioned, analyzed and discussed in terms of real productivity/cost benefit.
- Importance of reliability, specifically equipment support and maintenance to achieve expected performance.
- Operating environment, particularly roads and general site severity issues that reduce haulage performance.
- Probabilistic determination of truck fleet numbers and related issues.
- Consideration of indirect cost of infrastructure and additional mining cost to accommodate larger mining equipment.
- Loading equipment and truck mismatch; also truck-bunching effects interpreted in terms of production and unit production costs.

7.2 SUMMARY OF OUTCOMES

The following summary covers the outcomes of analysis and discussion of Chapters 1 through 5 and related tables, illustrations and supplementary information.

7.2.1 In the Beginning ----

This summary of outcomes is generalized and not comprehensive in detail of analysis results and outcome lists. More detailed outcomes are provided within each topic throughout the thesis where readers can view summaries of topics researched in context with the analysis, discussion and interpretation that precedes them.

Research undertaken was encouraged by historical mining background and experiences as discussed in Chapter 1. The significant influence of mining commodity prices, reducing in real terms over the past 150 years (Figure 1.1), diminishing operating cost margins and consequent demand for increased operating scale, in turn, requiring increased equipment scale have been fundamental cost-reduction drivers. With the potential for future, further cost pressures, the question needs to be answered - how to improve measuring, consequent planning and selection of operating systems and appropriate equipment to meet the ever-increasing precision required?

7.2.2 Research Framework and Preliminaries

Chapter 2 covers necessary matters of compliance, sets limits for the research and generally describes techniques, procedures, standards and methodology of the research.

Some indication of the philosophical approach to research by the author is provided for readers to gain understanding of personal views, investigation, analysis and interpretation.

7.2.3 Performance

Traditionally production and cost estimates have been deterministic, applying linear equations to mean/average values of variable performance parameters. Ever improving refined estimating techniques are needed to accommodate increasing cost pressures (Figure 1.1). Techniques delivering more reliable, realistic productivity and cost estimates require an understanding of effect of variability of analysis outcomes measured by distribution parameters including range and coefficient of variation (standard deviation/mean). Much of the research recorded in Chapter 3 involves statistical analysis and stochastic techniques. Suitable models for sample distributions of empirical data for bucket loads, truck payloads, bucket cycle times, truck loading times and truck trip times have been identified. Development of operating criteria provides a reliable basis for performance estimating and equipment selection. Probabilistic methods are a necessity for reliable estimation of truck fleet numbers.

The described techniques have been used for analysis during research with outcomes generally discussed, interpreted in some detail and clarified by tabulations and illustrations.

That the research has depended on statistical analytical methods is not meant to imply that deterministic methods are flawed or outdated. Initial productivity and estimates and equipment selection will generally be deterministic with adoption of statistical mathematical techniques when the degree of precision demands fine tuning.

Research of the many equipment selection criteria generally started from a deterministic basis; but naturally led to the more refined techniques of statistical analysis for greater understanding of the nature of the performance factors investigated. The outcomes of the research have hopefully provided starting points for readers to take the equipment selection process, mining productivity and cost estimation to higher levels of refinement and reliability.

7.2.3.1 Loading Equipment

Early topics in Chapter 3 cover loading equipment selection criteria and performance characteristics of loading equipment options for loading mining trucks. Critical operating practices and characteristics have been covered in some detail. Especially the intrinsic variability of loading equipment efficiency relative to number of passes and truck changeover time has been analyzed. Bucket load/truck payload interrelated variability and significance of number of passes, bucket cycle time and truck loading time relative variability have been investigated in detail. The important understanding of central tendency of intrinsic bucket cycle time, skewing effect of non-intrinsic high outliers and consequent positive skewing of bucket cycle and truck loading time distributions is also described in some detail.

The relationship between variability of truck payloads measured by coefficient of variation (CV) and number of bucket passes was explored; also implications of truck payload CV for compliance with Caterpillar's 10/10/20 Policy for control of truck payloads.

Implications for truck performance of payload and loading time variability have been briefly discussed.

7.2.3.2 Mining Trucks

The research moved on to mining trucks where initial considerations emphasized the relatively greater importance of unit cost of trucks compared with loading equipment. Truck types and applications were discussed.

Factors affecting intrinsic performance of mining trucks were reviewed with body capacity selection discussed in some detail.

An important outcome of the research is the recognition of a relationship between variability of voids ratio of material to be loaded and hauled and variability of truck payloads; and the logical extension that quality control of drilling and blasting has significant effect on truck payload dispersion.

The research moved on to influence on wheel loads of trucks of movement of truck payload centre of gravity within the truck body; also wheel chocking and gradient wheel-load effects. Implications for on-board payload sensing from suspension pressures, as well as tyre life, are discussed.

Tyres were investigated in terms of life and costs over a range of mining trucks from nominal 53 tonne through to ultra class 350 tonne payload trucks. Analysis identified tyre costs to be of increasing importance as truck scale increases, due to the need for in-service improvements to tyres over significant time periods. Tyre design and manufacture for large and ultra-class trucks is tuned to specific service for a particular mine site.

Bucket passes and truck payloads were further investigated – especially the perceived benefit of sacrificing passes under appropriate circumstances to improve operating “rhythm”. This desirable state of cyclic transport most closely approaches and realizes some of the benefits, at least in part, of continuous operations.

Truck performance and productivity were researched in terms of truck trip times separated into intrinsic, reasonably predictable variability of inherent truck performance; and less predictable, non-intrinsic time elements introduced by the nature of the operations. Non-intrinsic time elements are generally casual delays and interruptions to hauling operations that are estimated by empirical evidence of experience in-house or at similar operations.

Managing intrinsic time loss requires control and continuation of consistent equipment performance. Managing non-intrinsic time loss requires action to remedy undesirable operational activities, operating misadventure, unexpected delays from latent conditions; also from poor operating practices.

The necessity for and importance of sustaining correctly designed and maintained haul roads to achieve best possible truck-operating “rhythm” was emphasized.

Maintenance issues related to mining equipment selection were examined. The importance of equipment reliability that is substantially a benefit from selecting equipment with sound, adequate product support from dealers and OEM was emphasized. Reliability is seen as most important, likely more so than difference in purchase price and direct owning and operating costs. Differences in direct purchase costs, owning and operating costs and overall performance between different models of equipment of similar performance offered by individual OEM/suppliers tend to be subtle rather than significant.

Deciding necessary truck fleet numbers to satisfy a total production plan was initially analyzed deterministically. The limitations of deterministic analysis were demonstrated. Probabilistic techniques for deciding truck fleet numbers; also the effect of changing numbers of loading units were explored, summarized and interpreted and the more reliable results illustrated.

Fleet matching of trucks (with loading equipment), over trucking and under trucking were considered in terms of performance effect. Causes of bunching, and remedial measures were also reviewed and discussed.

Time categorization for load and haul operations was reviewed with time elements defined and discussed.

7.2.4 Equipment Selection

In Chapter 4 key considerations for loading equipment and mining truck selection were identified and discussed. Process and strategy was reviewed. Loading and hauling equipment options have been discussed in some detail, illustrating the relatively small performance differences of similar equipment from competitive manufacturers.

Diminishing cost benefit with increasing scale has been reviewed. The question: “Is

Bigger Better?” has been discussed in context with diminishing unit cost benefits as truck scale increases; also in terms of differing operational conditions.

Direction and focus of future improvements in loading and hauling equipment were also reviewed. Robust generic design criteria for mining trucks consistent over the full range of nominal truck capacities have been demonstrated. Consistency of truck design criteria has been used to extrapolate selected criteria for a hypothetical nominal 500 tonne capacity mining truck with relevant theoretical specification.

Historically application of Trolley Assist systems in Australia has not been justifiable due to:

- Comparatively low price of diesel fuel in Australia.
- Relatively lower power efficiency of DC wheel motors compared with diesel powered mining trucks.
- Questionable reliability of overhead conductor systems due to overload rejection (mainly caused by over loading of trucks) with need for extra access ramp width; and
- Applications limited to mines where power supply infrastructure on access ramps has many years of permanence to facilitate cost amortization.

Recent changes in the relevant cost factors, including development of AC wheel motors with improved efficiency; also substantially-increased diesel fuel prices have closed the economic gap. Trolley Assist can no longer be summarily dismissed for open pit mine haulage in Australia. In the current economic context it is certainly an option that needs to be included in consideration of feasibility of projects with deep-open-pit hauls.

7.2.5 Costs – Relative Indices and Comparisons

Absolute costs, estimates or actual expenses, are a "snapshot in time". Equipment operating costs, in money terms, are only valid for limited time due to volatile cost elements of fuel, lubricants and tyres. Recently shortages of skilled labour have caused wage escalation making labour costs less predictable. All analysis for the research has been in terms of cost indices to make cost analysis generic and applicable for longer time periods. Cost apportioning is generally based on empirical evidence. Obviously the relativity of cost indices changes with time. But it is convenient to update indices by factoring changes as effected for the research. Cost indices so derived are considered sufficiently accurate for the comparative purposes applied in the research.

Derivation of cost indices has been explained, including adjustments for recent substantially-increased fuel and tyre costs.

Capital costs and recovery (amortization) have been discussed in the context of owner mining; also in terms mining contract pricing where service contractors need to recover depreciation (conventionally accounted for by owners as a non-cash cost) through contract remuneration.

The concept of fuel consumption as a basis for estimating equipment life has been reviewed.

Cost effects of overloading trucks and payload dispersion are analyzed, interpreted and implications discussed. Also, cost effect of truck payload dispersion on truck driveline components including tyres is analyzed, interpreted and summarized in terms of cost significance.

Bucket-pass sacrifice, discussed in terms of productivity earlier in the research, is analyzed in cost-effect terms to supplement productivity analysis.

Finally, over trucking, under trucking and bunching are analyzed in some detail to provided a basis for definitive cost estimation that affords due recognition of these factors in terms of productivity and costs.

CHAPTER 8

CONCLUSIONS

AND

RECOMMENDATIONS FOR FURTHER RESEARCH

8.1 CONCLUSIONS

8.1.1 Compliance with Objectives

In final review:

- Outcomes of the research generally satisfy Research Objectives stated in Section 1.2.
- Research methodology, including techniques, procedures and limitations generally complies with commitments outlined in Chapter 2.
- In common with most human endeavour, open pit mining, specifically load and haul operations, is a complex of interrelationships that influence interfacing and overlapping activities.
- Therefore boundaries adopted during the research are artificial and determined by the author; not necessarily a logical boundary or limit where the conditions change to a degree justifying topic isolation.
- Load and haul operations are a continuum of interrelated activities. No sooner has consideration, analysis or interpretation of a topic reached a logical conclusion than another topic emerges with as cogent and demanding justification for research as topics initially investigated.

It is left to readers to determine the value of the research for their own purposes. For the author it has consolidated observation and study of mining operations over a working lifetime with more recent focus on open pit mining operations.

8.1.2 Conclusions

Section 7.2.1 indicates that detailed outcomes of the research follow individual sections. The following conclusions are a summary of selected research results.

Outcomes of the research include the understanding that, for all selection criteria that are variables, variability needs to be contained within acceptable intrinsic and controlled non-intrinsic limits. That includes payload, loader cycle time, loading time, truck cycle time. As variability truck payload increases truck performance reduces and costs for fuel, mechanical maintenance, tyres and all other consumables increase. Generally productivity decreases and operating cost increases with increase in variability of selection criteria.

For best operational production and cost performance of mining trucks, based on analysis of a sample >1,000 loads, truck payloads need to be kept in a distribution range with a mean value close to target payload , say in the range +/- 3%; and a coefficient of variation (CV) $\leq$ 6.5% (complies with Caterpillars “20” criterion of their 10/10/20 Policy). For best loading efficiency and truck loading time, non-intrinsic time delays (not directly related; but often operationally unavoidable time attributes) must be managed to a minimum where a tendency for symmetry of the loading time distribution begins to emerge.

Within a truck payload range of +/- 10% lower unit costs are realized from under-loading compared with over-loading. For the same truck payload range productivity gain or loss is similar, +/- some 5%.

A well defined equipment selection process, inclusive of all stakeholders, is an absolute necessity for satisfactory downstream productivity and cost performance.

Selection of “ultra-class” mining trucks ($\geq$ 290 -tonne payload) and suitable loading equipment is for special mining applications only. Where favourable local operating environment and cost factors supplement diminishing cost-benefits of truck scale, ultra-class trucks may be justified. Replacing a fleet of large mining trucks (150 to 220 tonnes payload) with ultra-class trucks is unlikely to deliver any reliable productivity or cost improvements except for exceptionally large-scale bulk-product mining operations in environments where establishment and infrastructure costs are most significant.

Bigger is not always better – only where bigger can be shown to be better by reasons in addition to the only-modest unit-cost benefits of ultra-class equipment.

Truck over-loading may, to a moderate degree, increase productivity, but only at increased unit cost. From a unit-cost perspective it is better to under-load than over-load mining trucks.

Where unit production cost is more important than absolute productivity, under trucking is favoured compared with over trucking loading equipment.

The nett effect of bunching of mining trucks is a queuing effect of loss of effective truck hours. To offset the queuing effect, basic productivity required needs to be adjusted to anticipate "bunching inefficiency" i.e., queuing effect. The "basic number of trucks" delivered by deterministic estimating will, therefore, provide for bunching inefficiency before application of simulation applications or stochastic analysis is used to determine the necessary number of trucks in the fleet.

In difficult digging conditions it is more important to retain truck operating rhythm than to focus on achieving target payload by indiscriminately adding loader passes. Where trucks are waiting to load, operational tempo should be restored by sacrificing one or more passes. Obviously it should not need commenting that, with trucks constantly queuing, there is always the trivial option to stand a truck down. Trucks generally should be loaded by not more than the nominal (modal) number plus one pass. When there are no trucks waiting or about to form a queue, then filling the trucks to target payload provides best-production and lowest-cost benefits. Experienced operators adopt these operating practices intuitively.

It has been generally concluded that where criteria for the selection of load and haul equipment for open pit mining are variables then it is necessary to:

1. Understand the inter-relatives and drivers that determine intrinsic performance of an item of equipment; and to manage the influence of those inter-relatives for best-practice variability control. That is, to change the drivers of the inter-relatives, and accept the fundamental effect of drivers of intrinsic attributes.

2. Identify and quantify non-intrinsic attributes that reduce performance and to manage to minimize or eliminate where possible to reduce to a minimum extraneous losses and inefficiencies.

3. Understand the difference between what can be changed and what needs to be accepted as a natural consequence.

8.2 RECOMMENDATIONS FOR FURTHER RESEARCH

The outcomes of the work raise the prospect of further research. There are likely many avenues worthy of investigation including:

1. Correlation of drilling and blasting design parameters through condition of blasted material, particularly voids ratio, through to truck payload dispersion – that would provide a basis by measuring voids ratio to predict load and haul productivity and so indicate mining cost for each blasted block. This could be expected to yield a process for optimizing drill and blast designs and provide indication, both qualitatively and quantitatively, prior to load and haul of blasts where reduced quality of fragmentation will result in reduced load and haul productivity.
2. To further extend the research of the recognized link between truck payload variability and voids ratio variability in order to develop the relationship between error magnitudes of drill hole patterns and hole alignment and Coefficient of Variation for Voids Ratio - consequently to predict the necessary error limits for drill hole parameters to acceptably limit truck payload dispersion..
3. Correlation of drilling and blasting design parameters through particle size distribution to particle stacking in loading equipment buckets, relationship between particle size and form and bucket dimensions and shape, to bucket fill factors, bucket designs and capacity, productivity and related cost implications. This could be expected to enable practical setting of drill and blast design parameters for available buckets; or alternatively indicate necessary bucket dimensions to accommodate expected fragmentation.
4. Effect of autonomous equipment including remote control and operating management of open pit mines on productivity, mining costs and total product unit cost; and most suitable payload capacity of trucks for autonomous operation.
5. To determine relationships between cost parameters and so indicate where Trolley Assist is favoured or unfavourable. Recent global escalation of oil prices to a higher plateau will strengthen the case for trolley assist of electric-drive mining trucks. Improved efficiency of AC power systems has also

contributed. Thorough research of the production and cost drivers of trolley-assisted haulage systems for comprehensive understanding is required. It should be possible to create a generically-applicable analytical programme that tests economic feasibility of Trolley Assist given values of fuel price, local electric power charges and other relevant inputs.

6. Certainly there are many opportunities throughout the open pit mining industry for gathering empirical evidence to confirm, or disprove, the analysis and hypotheses developed from the research described in this thesis.

7. Comments on some potential benefits and influences on criteria for equipment selection from adoption of autonomous load and haul systems are provided in Section 6.1.6. Certainly evolution of DS applications and recent elevation to development of autonomous load and haul systems is a most fertile field for future research. It could be expected that, as individual scopes are determined for research of autonomous systems, other research avenues will manifest. The overall prospects for valuable research and potential for rewards in the form of increased productivity and reduced unit mining costs are cogent incentives that are unlikely to be ignored.

To those future researchers who wish to continue:

- To investigate any of the above prospective research opportunities or any other avenues that arise from them.
- Investigation for confirmation of the research outcomes herein, for critical review, or to generally advance understanding of open pit mining economics.
- To increase the understanding of load and hauls operations in open pit mining, or mining in general:

my sincere best wishes.

Raymond J Hardy

September 2007.

REFERENCES

Agricola, Georgius **(1556)**. *De Re Metallica*, 1912 English Translation by Hoover Herbert C and Hoover Lou, printed Dover 1950: reprinted by The Mining Magazine, Edgar Rickard, London.

Atkinson#1, Thomas, **(1992)**, *Selection and Sizing of Excavating Equipment*, SME Mining Engineering Handbook, Two Vols. Ed., H. L. Hartman, Littleton, Colorado USA, Society for Mining Metallurgy, and Exploration, Inc. Vol. 2 pp1311 to 1333.

Atkinson#2, Thomas, **(1992)**, *Future Concepts in Surface Mining*, SME Mining Engineering Handbook, Two Vols. Ed., H. L. Hartman, Littleton, Colorado USA, Society for Mining Metallurgy, and Exploration, Inc. Vol. 2 pp1355 to 1357.

Atkinson#3, Thomas, **(1992)**, *Design and Layout of Haul Roads*, SME Mining Engineering Handbook, Two Vols. Ed., H. L. Hartman, Littleton, Colorado USA, Society for Mining Metallurgy, and Exploration, Inc. Vol. 2 pp1334 to 1342.

Banks J, Carson J S II and Nelson B L **(1996)**, *Discrete Event System Simulation 2nd Edit.*, Upper Saddle River, New Jersey, Prentice Hall.

Berkman D A **(1989)**, *Monograph 9 Field Geologists Manual 3rd Edit*,, The Australasian Institute of Mining and Metallurgy.

Blackwell G H **(1999)**, *Estimation of large open pit haulage truck requirements*, CIM Bulletin, Vol. 92 Issue 1028, pp143-148.

Bohnet Ernest, **(1992)**, *Method Advantages and Disadvantages*, SME Mining Engineering Handbook, Two Vols. Ed. H. L. Hartman, Littleton, Colorado USA, Society of Mining, Metallurgy and Exploration, Inc. Vol. 2 pp1533 - 1537.

Bozorgebrahimi A E, Hall. R A and Scoble M. **(2003)**, *Economies and Scale in Surface Mining Equipment - Dimensions and Impact Analysis*, Twelfth International Symposium on Mine Planning and Equipment Selection, Kalgoorlie, Western Australia, Australasian Institute of Mining and Metallurgy.

Caterpillar#1 (2004), *Caterpillar Performance Handbook Edition 35*, Caterpillar Inc, Peoria, Illinois, USA.

Caterpillar#2 (2004), *Fleet Performance and Cost Ver. 3.05B – Manual*, Caterpillar Inc, Peoria, Illinois, USA.

Chou Ya-lun, **(1969)**. *Statistical Analysis*, New York, Holt, Rinehart and Winston.

Burt Christina, Caccetta L, Hill S and Welgama P **(2005)**. *Models for Mining Equipment Selection*, Mathematics Department, Bentley, Curtin University of Technology, Western Australia (draft of paper in preparation).

Croser, Amanda, **(2004)**, *Continuous Improvement at a Large Open Pit Operation,*. The AusIMM New Leaders Conference, Ballarat, Victoria, Australia, The Australasian Institute of Mining and Metallurgy.

Cutler Tony, **(2002)**, *Ultra-class Truck Tire Operation and Management,* World Mining Equipment Haulage Conference, 2002, Tucson, Arizona, USA.

Cutler Tony, **(2004)**, *Tyre Life Indices*, Personal Communication, Otraco, Perth, Western Australia.

Cutts Terry, **(2005)**. *Wenco System Definitions*, Personal Communication, Kalgoorlie Central Gold Mine, Kalgoorlie, Western Australia.

Devore Jay L, **(1999)**, *Probability and Statistics for Engineering and the Sciences, 5th Edit.* California Polytechnic State University, Duxbury.

Elbrond J. **(1990)**, *Haulage System Analysis: Queuing Theory,* Surface Mining, Ed. B A Kennedy, Littleton CO USA, Society for Mining, Metallurgy and Exploration Inc., pp743 to 748.

Feddock J E **(2002)**, *Haul Roads, Mining Reference Handbook*, Ed. R L Lowrie, Littleton CO USA, Society for Mining, Metallurgy and Exploration Inc. pp197 to 204.

Filas F J **(2002)**, *Excavation, Loading and Material Transport,* Mining Reference Handbook, Ed. R L Lowrie, Littleton CO USA, Society for Mining, Metallurgy and Exploration Inc. pp215-241.

Files T I **(1990)**, *Hydraulic Excavators,* Surface Mining, Ed. B A Kennedy, Littleton CO USA, Society for Mining, Metallurgy, and Exploration, Inc., pp634 to 638.

Fivaz N, Cutland J R and Balchin C J **(1973)**, *Allocation and control of the truck fleet at Nchanga open-pit, Zambia,* Institute of Mining and Metallurgy.

Fourie G A F and Dohm Jnr G C **(1992)**, *Open Pit Planning and Design*, SME Mining Engineering Handbook, Two Vols., Ed. H L Hartman, Littleton CO USA, Society for Mining, Metallurgy, and Exploration, Inc. Vol. 2 pp1283 - 1292.

Gove Dan G and Morgan Bill. **(1994)**, *Optimizing Truck - Loader Matching,* Miners Forum 94 - Infinite Possibilities, (by invitation), Perth Western Australia, Caterpillar Inc. Peoria IL USA.

Gregory#1 Bruce S **(2002)**, *Truck Selection*, Driving Down Operating Costs Conference, Kalgoorlie, Western Australia, Australian Journal of Mining.

Gregory#2 Bruce S **(2003)**, *Excavator Selection*, Twelfth International Symposium on Mine Planning and Equipment Selection, Kalgoorlie, Western Australia, The Australasian Institute of Mining and Metallurgy.

Gupta Ritu **(2004)**. *Recommendation of Kolmogorov - Smirnov test for distribution modeling*, Personal Communication, Mathematics Department, Curtin University of Technology, Bentley, Western Australia.

Hanby I R **(1991, Part 1)**, *Enhancement of dump truck operating efficiency: Part 1 – method of shovel and truck matching to obtain whole numbers,* Transactions of the Institute of Mining and Metallurgy (Section A: Mining industry) Vol. 100 (January - April 1991).

Hanby I R **(1991, Part 3)**, *Enhancement of dump truck operating efficiency: Part 3 - effect of reduced payload*, Transactions of the Institute of Mining and Metallurgy (Section A: Mining industry) Vol. 100 (January - April 1991).

Hardy#1 Raymond J **(2003)**, *Four-Pass Loading, Must Have Or Myth?* Fifth Large Open Pit Mining Conference, 2003, Kalgoorlie, Western Australia, The Australasian Institute of Mining and Metallurgy.

Hardy#2 Raymond J **(2003)**, *Outsource or Owner Operate,* Twelfth International Symposium on Mine Planning and Equipment Selection, 2003, Kalgoorlie, Western Australia, Australasian Institute of Mining and Metallurgy.

Hardy#3 Raymond J **(2005)**, *Outsource versus Owner Operate*, IIR Contract Mining Conference 2005, Perth, Western Australia. An update of the previous paper, Hardy#2, 2003.

Harnischfeger (2003), *Peak Performance Practices*, Milwaukee, Wisconsin USA, P & H Mining Equipment, Brochure.

Harr Milton E **(1977)**, *Mechanics of Particulate Media - A Probabilistic Approach*, West Lafayette, Indiana, USA, McGraw Hill International Book Company.

Hays Ronald M. **(1990)**, *Trucks*, Surface Mining, Ed. B A Kennedy, Littleton CO USA, Society for Mining, Metallurgy and Exploration, Inc., pp672-691.

Humphrey James D **(1990)**, *Walking Draglines*, Surface Mining, Ed. B A Kennedy, Littleton, CO. USA, Society for Mining, Metallurgy, and Exploration, Inc. pp638 to 652.

James John F. (**1995)**, *Contract Management in Australia*, Harper Educational (Australia) Pty. Ltd. Ch. 6 pp91 to 100.

Kaufman Walter K. and Ault James C. **(1977)**, *Design of Surface Mine Haul Roads - A Manual*, United States Department of the Interior, Bureau of Mines, Information Circular 8758.

Kirk L J **(2000)**, *Owner versus contract mining*, Paper delivered at a conference in Athens, November 2000.

Kloverkorn Henning **(2004)**, *Comments on Liebherr TI 272*, Personal Communication, Perth, Western Australia.

Maloney Chad F **(2002)**, *A Comparative Analysis and Optimization of the Haulage Fleet at the KCGM Fimiston Pit Operations*, Thesis for Bachelor of Mining Engineering, WA School of Mines, Kalgoorlie, Western Australia, Curtin University of Technology pp55 & 56.

Mason A A C (Bert) **(1994)**, *No Two The Same*, The AusIMM Publication Series 3/94, The Australasian Institute of Mining and Metallurgy.

McClelland James F **(1941)**, *Prospecting, Development and Exploitation of Mineral Deposits*, Mining Engineers' Handbook, Two Vols. Robert Peele and Church J A, New York, John Wiley & Sons, Vol. 1 pp10-430 to 10-472.

Perich Peter N **(2003)**, *Used Equipment Prices*, Personal Communication, WesTrac, Caterpillar dealer, South Guildford Western Australia.

Power Anthony **(2005)**, *VIMS Payload Response with New Ride Struts*, Personal Communication, WesTrac, Caterpillar dealer, South Guildford, Western Australia.

Ramani R V **(1990)**, *Haulage Systems Simulation Analysis,* Surface Mining, Ed. B A Kennedy, Littleton CO USA, Society for Mining Metallurgy and Exploration pp724-742.

Rea David **(2005)**, *VIMS Accuracy Summary,* Personal Communication, Caterpillar Inc. Peoria ILL USA.

Samuelson Paul A **(1958)**, *Economics, 4th Edit.* New York, McGraw-Hill.

Sargent F R **(1990)**, *Mining and Quarry Shovels*, Surface Mining, Ed. B A Kennedy, Littleton CO USA, Society for Mining Metallurgy and Exploration Inc. pp626-638.

Smith L D **(1997)**, *A critical examination of the methods and factors affecting the selection of an optimum production rate,* CIM Bulletin, Vol. 90 Issue 1007, pp48-54.

Spiegel Murray R **(1961)**, *Theory and Problems of Statistics,* Schaum's Outline Series, New York, McGraw-Hill.

Suzuki David **(1990)**, *Inventing the Future,* Sydney, New South Wales, Allen & Unwin.

Tasman (1997), *Bench Marking the Productivity of Australia's Black Coal Industry,* Tasman Asia Pacific Pty Ltd, Lyneham Australian Capital Territory. pp i to xv and 1 to 85.

Tasman (1997), *The Scope for Productivity Improvement in Australia's Open Cut Black Coal Industry,* Tasman Asia Pacific Pty Ltd, Lyneham Australian Capital Territory, pp i to xv and 1 to 85.

Voss B **(2001)**, *Is Bigger Better?* World Mining Equipment, Issue November 2001.

Wiebmer John **(1994)**, *Different Horses for Different Courses -Choosing the right loading tool for your operation,* Miners Forum 94 Conference, Las Vegas USA, Society for Mining, Metallurgy, and Exploration Inc.

VOLUME 2

APPENDED TABLES

A comprehensive List of Tables and location of first reference in the text is provided in pp xii to xv following the Table of Contents

Tabulations are in chronological order as they are referred to in the text.

TABLE 2.1. TYPICAL MINING OPERATIONS - KPI MINE OPERATIONS PLAN										Page 1 of 2 Month:
CONTINUOUS IMPROVEMENT REQUIREMENTS										Year: 1998
	OBJECTIVE PERFORMANCE MEASUREMENT		BENCHMARK	TABLE OF ACTUAL PERFORMANCE						
KEY ELEMENTS	Key Performance Indicator (KPI)	Unit of Measure	Performance Standard	October		November		December		
				Month	To Date	Month	To Date	Month	To Date	
Load and Haul - Productivity	**Total Volume Mined to Dec '98**	**Average BCM/Day**	**81,500**							
	Total Volume Mined Jan '99 on	**Average BCM/Day**	**56,000**							
	Truck Utilisation	**%**	**93%**							
	BCM/Truck Hr - % of Monthly Target (17 x Cat 793C only)	**%**	**100%**							
	Shovel Production – Hard	**BCM/Hour**	**1,500**							
	(Below 504RL) Medium		**1,650**							
	Soft		**1,900**							
	Shovel Digability – Hard	**BCM/Hour**	**1,730**							
	Medium		**1,900**							
	Soft		**2,185**							
	Supplementary Performance Indicators (SPI)									
	Average Load Cat 793C	Tonnes	240							
	Average Load Cat 789	Tonnes	180							
	Average Load Cat 777D	Tonnes	97							
	Liebherr R996 Shovel Utilisation	%	95%							
Prepared: R J Hardy										

:	TABLE 2.1 TYPICAL MINING OPERATIONS - KPI MINE OPERATIONS PLAN									Page 2 of 2 Month:
CONTINUOUS IMPROVEMENT REQUIREMENTS										Year: 1998
KEY ELEMENTS	**OBJECTIVE PERFORMANCE MEASUREMENT** Key Performance Indicator (KPI)	Unit of Measure	**BENCHMARK** **Performance Standard**	**TABLE OF ACTUAL PERFORMANCE**						
				October		**November**		**December**		
				Month	**To Date**	**Month**	**To Date**	**Month**	**To Date**	
Load and Haul - Efficiency	**17 x Cat 793C only: Truck Efficiency**	**%**	**87%**							
(Subject to review on basis of Modular data and definition)	**3 x Liebherr R996 only: Shovel Efficiency**	**%**	**95%**							
Roads/Day Works	**Avg. Tyre Life (Cat 793C only)**	**Hours**	**3,500**							
	Supplementary Performance Indicators									
	Road Condition - Strut Pressure Analysis	Yes/No	Yes							
	Floor Condition - Strut Pressure Analysis	Yes/No	Yes							
Ancillary Equipment	**Ancillary Plant Costs Per BCM**	**$/BCM**	**0.343**							
Drill and Blast	**Drill Penetration Rate:**									
	311mm diameter holes	**Metres/Hour**	**32m/hour**							
	229mm diameter holes	**Metres/Hour**	**35m/hour**							
In-Pit Stocks At End of Month	**Broken Material Available**	**BCM**	**700K**							
Crusher Feed	**Crusher Feed Rate**	**Tonnes/ Op. Hour**	**2,000**							
Prepared: R J Hardy										

Table 3.5 - Bucket Load Data Observed 16 February 2004
700 tonne Hydraulic Shovel Loading 220 tonne Trucks

		Pass 1			Pass 2			Pass 3			Pass 4			Pass 5				
Item	**Truck**	**Cum.**	**Nett**	**Corrected**	**Cum**	**Nett**	**Corrected**	**Cum**	**Nett**	**Corrected**	**Cum**	**Nett**	**Corrected**	**Cum**	**Nett**	**Corrected**	**Final Payload**	**Number Bucket Loads**
		Tonnes	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	**Tonnes**	
1	**222**	69.1	69.1	69.1	120.9	51.8	51.8	181.2	60.3	60.3	221.4	40.2	40.2				221.4	4.0
2	**215**	51.8	51.8	52.8	107.0	55.2	56.2	157.9	50.9	51.8	199.5	41.6	42.4				203.2	4.0
3	**215**	44.7	44.7	43.8	87.3	42.6	41.8	142.7	55.4	54.3	204.5	61.8	60.6				200.5	4.0
4	**208**	72.0	72.0	70.6	127.6	55.6	54.5	178.3	50.7	49.7	226.5	48.2	47.2				222.0	4.0
5	**208**	73.0	73.0	71.7	116.5	43.5	42.7	170.5	54.0	53.0	226.4	55.9	54.9	267.7	41.3	40.6	262.9	5.0
6	**202**	84.4	84.4	83.7	140.1	55.7	55.2	193.0	52.9	52.4	240.4	47.4	47.0				238.3	4.0
7	**208**	57.5	57.5	58.0	114.5	57.0	57.5	185.3	70.8	71.4	231.4	46.1	46.5				233.5	4.0
8	**202**	87.5	87.5	88.7	144.0	56.5	57.3	201.7	57.7	58.5	236.3	34.6	35.1				239.5	4.0
9	**215**	61.5	61.5	62.3	110.6	49.1	49.7	149.0	38.4	38.9	199.6	50.6	51.2				202.1	5.0
10	**202**				98.4			153.9	55.5	56.5	196.7	42.8	43.5	228.8	32.1	32.7	232.8	4.0
11	**202**	60.8	60.8	60.7	96.7	35.9	35.8	141.8	45.1	45.0	180.7	38.9	38.8	208.7	28.0	27.9	208.2	5.0

Table 3.8 - Bucket Load Data Observed, 16 - February 2004
190 tonne Wheel Loader loading 220 tonne Trucks

			Pass 1			Pass 2			Pass 3			Pass 4			
Item	Truck		Cumulative	Nett	Corrected	Cumulative	Nett	Corrected	Cumulative	Nett	Corrected	Cumulative	Nett	Corrected	
			Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	
1	204		53.3	53.3	55.6	70.8	17.5	18.3	91.2	20.4	21.3	118.2	27.0	28.2	
2	210		52.8	52.8	54.8	81.0	28.2	29.3	106.0	25.0	26.0	131.7	25.7	26.7	
3	219			0.0	0.0				103.5		0.0	124.6	21.1	21.9	
4	204		58.6	58.6	60.2	78.1	19.5	20.0	103.3	25.2	25.9	124.4	21.1	21.7	
5	219			0.0	0.0		0.0	0.0	85.3		0.0	105.6	20.3	20.9	
6	204			0.0	0.0	79.4		0.0	103.5	24.1	23.5	125.0	21.5	20.9	
7	219		47.8	47.8	51.7	67.8	20.0	21.6	89.7	21.9	23.7	113.7	24.0	26.0	

Pass 5			Pass 6			Pass 7			Pass 8			Pass 9			
Cumulative	Nett	Corrected	Cumulative	Nett	Corrected	Cumulative	Nett	Corrected	Cumulative	Nett	Corrected	Cumulative	Nett	Corrected	Final Payload
Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes
141.3	23.1	24.1	164.3	23.0	24.0	184.8	20.5	21.4					0.0		192.9
156.4	24.7	25.7	181.7	25.3	26.3	203.5	21.8	22.6		0.0			0.0		211.4
146.5	21.9	22.7	167.5	21.0	21.8	183.6	16.1	16.7	199.5	15.9	16.5				206.9
147.1	22.7	23.3	171.0	23.9	24.6	193.8	22.8	23.4		0.0			0.0		199.2
125.1	19.5	20.1	145.4	20.3	20.9	169.8	24.4	25.2	183.8	14.0	14.4		0.0		189.5
143.2	18.2	17.7	163.5	20.3	19.8	188.1	24.6	24.0	208.1	20.0	19.5	225.9	17.8	17.3	220.0
135.9	22.2	24.0	158.4	22.5	24.3	174.3	15.9	17.2	194.1	19.8	21.4		0.0		209.9

Table 3.13 Bucket Load Summary - April 2004
190 tonne Wheel Loader loading 220 tonne Trucks

First Loads	Intermediate Loads										Last Loads		Number Bucket Loads
Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	
		23.0	22.1	28.8	25.3	22.8	20.1				15.7		9
38.2	27.7	29.2	23.4	26.6	25.7						28.4		7
		21.7	20.4	25.4	29.0	22.1	22.1				13.5		9
49.9	24.3	21.5	22.5	15.2	22.2	19.1	18.8	20.8	12.6	22.0		15.0	12
		21.2	24.7	24.2	26.1	21.5	23.2				20.0		9
33.8	18.8	34.7	22.3	21.6	20.7	21.7	23.3				19.2		9
		37.1	26.1	28.6	30.9						22.8		7
41.4	21.2	25.7	24.4	21.7	14.8	22.4	22.7					18.9	9

Table 3.28 - Summary of Measures of Central Tendency and Dispersion

Mode/Median/Mean & Variation Coefficients

		FILTER LEVEL 1				FILTER LEVEL 2				FILTER LEVEL 3			
Data Sub-samples	Description	Pass 1 + Changeover Cycle Time	All Passes	Passes Less First	Loading Time	Pass 1 + Changeover Cycle Time	All Passes	Passes Less First	Loading Time	Pass 1 + Changeover Cycle Time	All Passes	Passes Less First	Loading Time
		Seconds	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds
All Passes	Mode	44.00	30.00	30.00	193.00	44.00	28.00	28.00	193.00	44.00	28.00	28.00	199.00
	Median	59.00	32.00	30.00	213.00	56.00	32.00	30.00	205.00	46.00	30.00	29.00	189.50
	Mean	80.52	40.80	32.39	233.44	62.40	37.38	32.04	212.59	46.53	32.89	29.98	187.18
	σ	75.07	37.43	10.07	84.81	24.68	17.03	7.76	42.70	6.69	8.18	4.85	25.70
	CV	0.932	0.917	0.311	0.363	0.395	0.456	0.242	0.201	0.144	0.249	0.162	0.137
	Modal CV	1.706	1.248	0.336	0.439	0.561	0.608	0.277	0.221	0.152	0.292	0.173	0.129
Five Passes	Mode					56.00	28.00	28.00	168.00	56.00	28.00	28.00	168.00
	Median					56.00	31.00	31.00	189.50	45.00	33.00	30.00	168.00
	Mean					62.75	39.12	33.21	195.58	46.23	34.16	31.14	170.81
	σ					24.40	17.82	8.61	35.64	7.20	8.48	5.62	19.67
	CV					0.389	0.455	0.259	0.182	0.156	0.248	0.180	0.115
	Modal CV					0.436	0.636	0.308	0.212	0.129	0.303	0.201	0.117
Six Passes	Mode					43.00	30.00	30.00	198.00	43.00	29.00	29.00	193.00
	Median					55.00	30.00	30.00	211.50	46.50	30.00	29.00	192.50
	Mean					62.36	36.99	31.92	221.95	46.72	32.55	29.72	195.30
	σ					24.32	16.56	7.53	40.04	6.63	8.08	4.63	16.75
	CV					0.390	0.448	0.236	0.180	0.142	0.248	0.156	0.086
	Modal CV					0.566	0.552	0.251	0.202	0.154	0.279	0.160	0.087
Seven Passes	Mode					39.00	27.00	27.00	203.00	39.00	26.00	26.00	207.00
	Median					54.00	30.00	29.00	233.00	46.00	29.00	28.00	214.00
	Mean					64.44	34.67	29.71	242.70	46.57	31.17	28.61	218.22
	σ					28.84	17.05	5.50	37.03	5.89	7.49	3.69	14.57
	CV					0.448	0.492	0.185	0.153	0.127	0.240	0.129	0.067
	Modal CV					0.740	0.631	0.204	0.182	0.151	0.288	0.142	0.069

Table 3.29 - Summary of Measures of Central Tendency and Dispersion

Mode/Median/Mean & Variation Coefficients

		FILTER LEVEL 1				FILTER LEVEL 2				FILTER LEVEL 3			
Data Sub-samples	Description	Pass 1 + Changeover Cycle Time	All Passes	Passes Less First	Loading Time	Pass 1 + Changeover Cycle Time	All Passes	Passes Less First	Loading Time	Pass 1 + Changeover Cycle Time	All Passes	Passes Less First	Loading Time
		Seconds	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds
All Passes	Mode	44.00	30.00	30.00	193.00	44.00	28.00	28.00	193.00	44.00	28.00	28.00	***193.00***
	Median	59.00	32.00	30.00	213.00	56.00	32.00	30.00	205.00	46.00	30.00	29.00	189.50
	Mean	80.52	40.80	32.39	233.44	62.40	37.38	32.04	212.59	46.53	32.89	29.98	187.18
	σ	75.07	37.43	10.07	84.81	24.68	17.03	7.76	42.70	6.69	8.18	4.85	25.70
	CV	0.932	0.917	0.311	0.363	0.395	0.456	0.242	0.201	0.144	0.249	0.162	0.137
	Modal CV	1.706	1.248	0.336	0.439	0.561	0.608	0.277	0.221	0.152	0.292	0.173	***0.133***
Five Passes	Mode					***40.00***	28.00	28.00	168.00	***40.00***	28.00	28.00	168.00
	Median					56.00	31.00	31.00	189.50	45.00	33.00	30.00	168.00
	Mean					62.75	39.12	33.21	195.58	46.23	34.16	31.14	170.81
	σ					24.40	17.82	8.61	35.64	7.20	8.48	5.62	19.67
	CV					0.389	0.455	0.259	0.182	0.156	0.248	0.180	0.115
	Modal CV					***0.610***	0.636	0.308	0.212	***0.180***	0.303	0.201	0.117
Six Passes	Mode					43.00	30.00	30.00	***193.00***	43.00	29.00	29.00	193.00
	Median					55.00	30.00	30.00	211.50	46.50	30.00	29.00	192.50
	Mean					62.36	36.99	31.92	221.95	46.72	32.55	29.72	195.30
	σ					24.32	16.56	7.53	40.04	6.63	8.08	4.63	16.75
	CV					0.390	0.448	0.236	0.180	0.142	0.248	0.156	0.086
	Modal CV					0.566	0.552	0.251	***0.207***	0.154	0.279	0.160	0.087
Seven Passes	Mode					***44.00***	27.00	27.00	***207.00***	***44.00***	26.00	26.00	***207.00***
	Median					54.00	30.00	29.00	233.00	46.00	29.00	28.00	214.00
	Mean					64.44	34.67	29.71	242.70	46.57	31.17	28.61	218.22
	σ					28.84	17.05	5.50	37.03	5.89	7.49	3.69	14.57
	CV					0.448	0.492	0.185	0.153	0.127	0.240	0.129	0.067
	Modal CV					***0.655***	0.631	0.204	***0.178***	***0.134***	0.288	0.142	***0.070***

Table 3.30 - All Truck Loading Times - Filter Level 3
One-Second & Five Second Frequency Distribution

Basic Frequency Interval	**131**	**132**	**133**	**134**	**135**	**136**	**137**	**138**	**139**	**140**	**141**	**142**	**143**	**144**	**145**	**146**	**147**	**148**	**149**	**150**
One-Second Frequency	1	2			1		3		1		2			1	3	1	1	2	1	1
Five-Second Frequency					4					4					6					6
Basic Frequency Interval	**151**	**152**	**153**	**154**	**155**	**156**	**157**	**158**	**159**	**160**	**161**	**162**	**163**	**164**	**165**	**166**	**167**	**168**	**169**	**170**
One-Second Frequency	3		1		3	1	1			3		1		1	1	1	3	**6**		1
Five-Second Frequency					7					5					3					11
Basic Frequency Interval	**171**	**172**	**173**	**174**	**175**	**176**	**177**	**178**	**179**	**180**	**181**	**182**	**183**	**184**	**185**	**186**	**187**	**188**	**189**	**190**
One-Second Frequency	3			1	1	2	5	**6**	2	2	1	5	2	4	3	4	1	3	3	5
Five-Second Frequency					5					17					15					16
Basic Frequency Interval	**191**	**192**	**193**	**194**	**195**	**196**	**197**	**198**	**199**	**200**	**201**	**202**	**203**	**204**	**205**	**206**	**207**	**208**	**209**	**210**
One-Second Frequency	4	5	**7**	2	3	3	3	5	**7**	1	2	2	2	1	1		3	1	1	3
Five-Second Frequency					**21**					19					8					8
Basic Frequency Interval	**211**	**212**	**213**	**214**	**215**	**216**	**217**	**218**	**219**	**220**	**221**	**222**	**223**	**224**	**225**	**226**	**227**	**228**	**229**	**230**
One-Second Frequency	3	2	3	3	1		2	1		1			2				1	1		
Five-Second Frequency					12					4					2					2
Basic Frequency Interval	**231**	**232**	**233**	**234**	**235**	**236**	**237**	**238**	**239**	**240**	**241**	**242**	**243**	**244**	**245**	**246**	**247**	**248**	**249**	**250**
One-Second Frequency	3	1		1	1	1			3					1	1					
Five-Second Frequency					6					4					2					
Basic Frequency Interval	**251**	**252**	**253**	**254**	**255**	**256**	**257**		**Totals**											
One-Second Frequency					1				188											
Five-Second Frequency					1				188											

Table 3.32 - Gamma Distribution Parameters

Figure	Description	Number of Records	Mean Seconds	Standard Deviation Seconds	Coefficient of Variation	Gamma Distribution Parameters				Comments
						Calculated		Used		
			μ	σ	CV	α	β	α	β	
3.2.9.7	**Filter Level 2, Bucket Cycle Times - All Passes Less First**	1725	32.04	7.76	0.242	*17.075*	*1.876*	**17.075**	**1.876**	Scale parameter **α** could be adjusted to an integer value in all four cases without materially affecting qualitative comparison - in which case the model distributions would be Erlang - as featured in references - see text.
3.2.9.8	**Filter Level 2 Truck Loading Times - All Passes**	368	212.59	42.7	0.201	*24.752*	*8.589*	**25.500**	**7.800**	
3.2.9.13	**Filter Level 3, Bucket Cycle Times - All Passes Less First**	882	29.98	4.85	0.162	*38.104*	*0.787*	**38.104**	**0.787**	Increasing scale parameter **α** indicates centralizing tendency
3.2.9.14	**Filter Level 3 Truck Loading Times - All Passes**	188	187.181	25.7	0.137	*53.279*	*3.513*	**53.279**	**3.513**	Tending to symmetrical - normal distribution

Table 3.35 - Comparison of Operating Weights of Caterpillar Mining Trucks

Caterpillar Performance Handbook 35, October 2004

Truck Model	GMW	Target Payload	Ratio Target Payload : GMW	Chassis Weight **	Ratio Chassis : GMW	Body Weight	Liner Kit	Total Body + Liners	Ratio Body : GMW	Calculated Payload	Ratio Calc : Target Payload	Ratio Body Weight : Payload	Full Fuel		Nominal Body Volume SAE Struck	SAE Heaped 2:1	Ratio Heaped : Struck
	Tonnes	Tonnes		Tonnes		Tonnes	Tonnes	Tonnes		Tonnes		Tonnes	Litres	Tonnes			
Dual Slope Body																	
773E	99.3	54.26	0.55	31.93	0.32	9.21	3.90	13.11	0.13	54.26	1.00	0.24	700	0.609	26.6	35.2	1.32
777D	163.3	90.73	0.56	51.33	0.31	15.78	5.46	21.24	0.13	90.73	1.00	0.23	1137	0.989	42.1	60.1	1.43
785C	249.5	140	0.56	80.195	0.32	21.47	7.64	29.11	0.12	140.20	1.00	0.21	1893	1.647	NA	78.0	NA
789C	317.5	180	0.57	101.006	0.32	26.28	9.43	35.71	0.11	180.78	1.00	0.20	3218	2.800	NA	105.0	NA
793C	383.7	218	0.57	122.415	0.32	32.11	11.08	43.19	0.11	218.10	1.00	0.20	4467	3.886	96.0	129.0	1.34
Flat Body																	
773E	99.3	53.82	0.54	31.93	0.32	9.55	4.00	13.55	0.14	53.82	1.00	0.25	680	0.592	26.6	35.5	1.33
777D	163.3	90.54	0.55	50.61	0.31	16.69	5.46	22.15	0.14	90.54	1.00	0.24	1137	0.989	42.0	60.2	1.43
793C	383.7	223	0.58	122.415	0.32	35.91	2.51	38.42	0.10	222.87	1.00	0.17	4467	3.886	110.0	147.7	1.34
797B	623.7	345	0.55	223.413	0.36	47.26	3.99	51.25	0.08	349.04	1.01	0.15	6814	5.928	173.0	220.0	1.27

Note: ** Chassis weight includes: coolants, lubricants, 100% fuel and 4% of chassis weight allowed as debris

Table 3.36 Voids Ratio, Loose Density and Body Volume Issues

Rock Type	*Granite*	
Insitu Density	*2.73*	Tonnes/BCM
Target Payload	*220*	Tonnes

Load Factors	Best Practice Expected	Poor Fragmentation & Grading	Very Poor Fragmentation & Grading
	0.61	*0.67*	*0.75*
Fill Efficiency Factor			*0.87*

Conversion Factor Estimated Body Capacity To SAE 2:1		
	1.10	

All Above Data (In Italics) Optional

Information and Comments	Mean Load Factor	Mean Loose Density	Mean Voids Ratio	Voids Ratio - CV	Voids Ratio Maximum $P_{0.999}$	Voids Ratio Minimum $P_{0.999}$	Expected Maximum Loose Density	Expect-ed Mean Loose Density	Expect-ed Minimum Loose Density	Theoretical Body Volume at Selected Fill Efficiency	Estimated SAE 2:1 Capacity Rating At Selected Body Factor	Theoretical Payload at Voids Ratio Minimum - Selected Fill Factor	Theoretical Payload at Voids Ratio Mean - Selected Fill Factor	Theoretical Payload at Voids Ratio Maximum - Selected Fill Factor	Maximum Loose Density Proportion of Mean	Theoretical Potential Overload at Voids Ratio Minimum - Selected Fill Factor	Minimum Loose Density Proportion of Mean
		Tonnes /LCM					Tonnes /LCM	Tonnes /LCM	Tonnes /LCM	Cubic Metres	Cubic Metres	Tonnes	Tonnes	Tonnes	%	%	%
Load Factor For Broken Granite From Cat Performance Handbook	0.61	1.67	0.64	0.00	0.639	0.639	1.665	1.665	1.665	151.85	167.034	220.00	220	220.00	100.00%	0.00%	100.00%
		1.67	0.64	0.05	0.738	0.541	1.772	1.665	1.571	151.85	167.034	234.06	220	207.54	106.39%	6.39%	94.33%
		1.67	0.64	0.10	0.836	0.442	1.893	1.665	1.487	151.85	167.034	*250.03*	*220*	*196.41*	*113.65%*	*13.65%*	*89.28%*
		1.67	0.64	0.15	0.935	0.344	2.031	1.665	1.411	151.85	167.034	*268.35*	*220*	*186.41*	*121.98%*	*21.98%*	*84.73%*
		1.67	0.64	0.20	1.033	0.246	2.192	1.665	1.343	151.85	167.034	*289.57*	*220*	*177.39*	*131.62%*	*31.62%*	*80.63%*

Table 3.36 Voids Ratio, Loose Density and Body Volume Issues - Continued

Information and Comments	Mean Load Factor	Mean Loose Density	Mean Voids Ratio	Voids Ratio - CV	Voids Ratio Maximum $P_{0.999}$	Voids Ratio Minimum $P_{0.999}$	Expected Maximum Loose Density	Expected Mean Loose Density	Expected Minimum Loose Density	Theoretical Body Volume at Selected Fill Efficiency	Estimated SAE 2:1 Capacity Rating At Selected Body Factor	Theoretical Payload at Voids Ratio Minimum - Selected Fill Factor	Theoretical Payload at Voids Ratio Mean - Selected Fill Factor	Theoretical Payload at Voids Ratio Maximum - Selected Fill Factor	Maximum Loose Density Proportion of Mean	Theoretical Potential Overload at Voids Ratio Minimum - Selected Fill Factor	Minimum Loose Density Proportion of Mean
Load Factor Amended to Reflect Hypothetical Poor Fragmentation and Grading	**0.67**	**1.83**	**0.49**	**0.00**	**0.493**	**0.493**	**1.829**	**1.829**	**1.829**	**138.25**	**152.075**	**220.00**	**220**	**220.00**	**100.00%**	**0.00%**	**100.00%**
		1.83	**0.49**	**0.05**	**0.568**	**0.417**	**1.927**	**1.829**	**1.741**	**138.25**	**152.075**	**231.78**	**220**	**209.36**	**105.35%**	**5.35%**	**95.16%**
		1.83	**0.49**	**0.10**	**0.644**	**0.341**	**2.036**	**1.829**	**1.660**	**138.25**	**152.075**	***244.89***	***220***	***199.70***	***111.31%***	***11.31%***	***90.77%***
		1.83	**0.49**	**0.15**	**0.720**	**0.265**	**2.158**	**1.829**	**1.587**	**138.25**	**152.075**	***259.57***	***220***	***190.90***	***117.99%***	***17.99%***	***86.77%***
		1.83	**0.49**	**0.20**	**0.796**	**0.189**	**2.296**	**1.829**	**1.520**	**138.25**	**152.075**	*276.13*	*220*	*182.83*	*125.51%*	*25.51%*	*83.11%*
Load Factor Further Amended to Reflect Very Poor Fragmentation and Grading	**0.75**	**2.05**	**0.33**	**0.00**	**0.333**	**0.333**	**2.048**	**2.048**	**2.048**	**123.50**	**135.854**	**220.00**	**220**	**220.00**	**100.00%**	**0.00%**	**100.00%**
		2.05	**0.33**	**0.05**	**0.385**	**0.282**	**2.129**	**2.048**	**1.972**	**123.50**	**135.854**	**228.81**	**220**	**211.84**	**104.00%**	**4.00%**	**96.29%**
		2.05	**0.33**	**0.10**	**0.436**	**0.231**	**2.218**	**2.048**	**1.901**	**123.50**	**135.854**	**238.35**	**220**	**204.27**	**108.34%**	**8.34%**	**92.85%**
		2.05	**0.33**	**0.15**	**0.487**	**0.179**	**2.315**	**2.048**	**1.835**	**123.50**	**135.854**	***248.73***	***220***	***197.22***	***113.06%***	***13.06%***	***89.65%***
		2.05	**0.33**	**0.20**	**0.539**	**0.128**	**2.420**	**2.048**	**1.774**	**123.50**	**135.854**	***260.05***	***220***	***190.64***	***118.20%***	***18.20%***	***86.66%***
Column Number	*1*	*2*	*3*	*4*	*5*	*6*	*7*	*8*	*9*	*10*	*11*	*12*	*13*	*14*	*15*	*16*	*17*

Legend Columns 12 to 17:

Bold	**Complies with 10:10:20 Rule**
Bold Italics	***Complies with 10:10 Rule; but not with 20 Rule***
Italics Only	*Does not comply with 10:10:20 Rule*

Table 3.37 Truck Payload CV v. Voids Ratio CV

Rock Type	*Granite*		
Insitu Density	*273*	Tonnes /BCM	
	Target Payload	*220*	Tonnes

Load Factors	Best Practice Expected	Poor Fragmentation & Grading	Very Poor Fragmentation & Grading
	0.61	*0.67*	*0.75*
	Fill Efficiency Factor	*0.87*	

Conversion Factor Estimated Body Capacity To SAE 2:1
1.10

<u>Caterpillar 10:10:20 Rule:</u>
CV = 0.065 complies with 120% absolute limit
CV = 0.0780 complies with not more than 10% greater than 110% limit

All Above Data (In Italics) Optional

Information and Comments	Load Factor	Expected Mean Loose Density	Mean Voids Ratio	Voids Ratio - CV	Expected Maximum Loose Density	Theoretical Body Volume at Selected Fill Efficiency	Theoretical Payload at Voids Ratio Minimum - Selected Fill Factor	Theoretical Payload at Voids Ratio Mean - Selected Fill Factor	Derived Payload CV	10:10:20 Rule Compliance & Probability
		Tonnes /LCM			Cubic Metres	Cubic Metres	Tonnes	Tonnes		
Load Factor For Broken Granite From Cat Performance Handbook	**0.61**	1.665	0.64	0.00	1.665	151.85	220.00	220.00	0.000	
		1.665	0.64	0.05	1.772	151.85	234.06	220.00	0.021	
		1.665	0.64	0.10	1.893	151.85	250.03	220.00	0.046	
		1.665	**0.64**	**0.14**	**1.999**	**151.85**	**264.04**	**220.00**	**0.065**	**"20"/0.999**
		1.665	0.64	0.15	2.031	151.85	268.35	220.00	0.073	
		1.665	**0.64**	**0.16**	**2.065**	**151.85**	**272.85**	**220.00**	**0.078**	**"10:10"/0.90**
		1.665	0.64	0.20	2.192	151.85	289.57	220.00	0.105	

Table 3.37 Truck Payload CV v. Voids Ratio CV - Continued

Information and Comments	Load Factor	Expected Mean Loose Density	Mean Voids Ratio	Voids Ratio - CV	Expected Maximum Loose Density	Theoretical Body Volume at Selected Fill Efficiency	Theoretical Payload at Voids Ratio Minimum - Selected Fill Factor	Theoretical Payload at Voids Ratio Mean - Selected Fill Factor	Derived Payload CV	10:10:20 Rule Compliance & Probability
		Tonnes /LCM			Cubic Metres	Cubic Metres	Tonnes	Tonnes		
Load Factor Amended to Reflect Hypothetical Poor Fragmentation and Grading	**0.67**	1.829	0.49	0.00	1.829	138.25	220.00	220.00	0.000	
		1.829	0.49	0.05	1.927	138.25	231.78	220.00	0.018	
		1.829	0.49	0.10	2.036	138.25	244.89	220.00	0.038	
		1.829	0.49	0.15	2.158	138.25	259.57	220.00	0.060	
		1.829	**0.49**	**0.16**	**2.195**	**138.25**	**264.04**	**220.00**	**0.065**	**"20"/0.999**
		1.829	**0.49**	**0.19**	**2.269**	**138.25**	**272.85**	**220.00**	**0.078**	**"10:10"/0.90**
		1.829	0.49	0.20	2.296	138.25	276.13	220.00	0.085	
Load Factor Further Amended to Reflect Very Poor Fragmentation and Grading	**0.75**	2.048	0.33	0.00	2.048	123.50	220.00	220.00	0.000	
		2.048	0.33	0.05	2.129	123.50	228.81	220.00	0.013	
		2.048	0.33	0.10	2.218	123.50	238.35	220.00	0.028	
		2.048	0.33	0.15	2.315	123.50	248.73	220.00	0.044	
		2.048	0.33	0.20	2.420	123.50	260.05	220.00	0.061	
		2.048	**0.33**	**0.22**	**2.457**	**119.39**	**264.04**	**220.00**	**0.065**	**"20"/0.999**
		2.048	**0.33**	**0.25**	**2.539**	**119.39**	**272.85**	**220.00**	**0.078**	**"10:10"/0.90**
Column Number	**1**	**2**	**3**	**4**	**7**	**10**	**12**	**13**	**18**	**19**

Table 3.38 Voids Ratio and Potential Truck Overloading

Rock Type	Granite		
Insitu Density	*2.73*	Tonnes/BCM	
Target Payload	*220*	Tonnes	

Load Factors	Best Practice Expected	Poor Fragmentation & Grading
	0.61	*0.67*
Fill Efficiency Factor		*0.87*

Conversion Factor Estimated Body Capacity To SAE 2:1
1.1

All Above Data (In Italics) Optional

Information & Comments	Mean Load Factor	Mean Loose Density	Mean Voids Ratio	Voids Ratio - CV	Voids Ratio Maximum P0.999	Voids Ratio Minimum P0.999	Expected Maximum Loose Density	Expected Mean Loose Density	Theoretical Body Volume at Voids Ratio 0.64 & Selected Fill Efficiency	Theoretical Payload at Voids Ratio Minimum - Selected Fill Factor	Theoretical Payload at Voids Ratio Mean - Selected Fill Factor	Theoretical Potential Overload at Voids Ratio Minimum - Selected Fill Factor
		Tonnes /LCM					Tonnes /LCM	Tonnes /LCM	Cubic Metres	Tonnes	Tonnes	%
Load Factor For Broken Granite From Cat Performance Handbook	0.61	1.665	0.64	0.00	0.639	0.639	1.665	1.665	151.85	220.00	220.00	0.00%
		1.665	0.64	0.05	0.738	0.541	1.772	1.665	151.85	234.06	220.00	6.39%
		1.665	0.64	0.10	0.836	0.442	1.893	1.665	151.85	***250.03***	***220.00***	***13.65%***
		1.665	0.64	0.15	0.935	0.344	2.031	1.665	151.85	*268.35*	*220.00*	*21.98%*
		1.665	0.64	0.20	1.033	0.246	2.192	1.665	151.85	*289.57*	*220.00*	*31.62%*

Table 3.38 Voids Ratio and Potential Truck Overloading - Continued

Information & Comments	Mean Load Factor	Mean Loose Density Tonnes /LCM	Mean Voids Ratio	Voids Ratio - CV	Voids Ratio Maximum P0.999	Voids Ratio Minimum P0.999	Expected Maximum Loose Density Tonnes /LCM	Expected Mean Loose Density Tonnes /LCM	Theoretical Body Volume at Voids Ratio 0.64 & Selected Fill Efficiency Cubic Metres	Theoretical Payload at Voids Ratio Minimum - Selected Fill Factor Tonnes	Theoretical Payload at Voids Ratio Mean - Selected Fill Factor Tonnes	Theoretical Potential Overload at Voids Ratio Minimum - Selected Fill Factor %
Load Factor Amended to Reflect Hypothetical Poor Fragmentation and Grading	0.67	1.829	0.49	0.00	0.493	0.493	1.829	1.829	151.85	**241.64**	**241.64**	**9.84%**
		1.829	0.49	0.05	0.568	0.417	1.927	1.829	151.85	***254.58***	***241.64***	***15.72%***
		1.829	0.49	0.10	0.644	0.341	2.036	1.829	151.85	*268.98*	*241.64*	*22.26%*
		1.829	0.49	0.15	0.720	0.265	2.158	1.829	151.85	*285.11*	*241.64*	*29.59%*
		1.829	0.49	0.20	0.796	0.189	2.296	1.829	151.85	*303.29*	*241.64*	*37.86%*
Load Factor Further Amended to Reflect Very Poor Fragmentation and Grading	0.75	2.048	0.33	0.00	0.333	0.333	2.048	2.048	151.85	*270.49*	*270.49*	*22.95%*
		2.048	0.33	0.05	0.385	0.282	2.129	2.048	151.85	*281.32*	*270.49*	*27.87%*
		2.048	0.33	0.10	0.436	0.231	2.218	2.048	151.85	*293.06*	*270.49*	*33.21%*
		2.048	0.33	0.15	0.487	0.179	2.315	2.048	151.85	*305.81*	*270.49*	*39.01%*
		2.048	0.33	0.20	0.539	0.128	2.420	2.048	151.85	*319.73*	*270.49*	*45.33%*
Column Number	1	2	3	4	5	6	7	8	10	12	13	16

Legend Columns 13 & 16:

Bold — **Complies with 10:10:20 Rule**

Bold Italics — ***Complies with 10:10 Rule; but not with 20 Rule***

Italics Only — *Does not comply with 10:10:20 Rule*

Table 3.57 Productivity Criteria For Sacrificing Bucket Loads

Bucket Cycle Time			Number of Passes Sacrificed		Proportion of Payload Sacrificed / Number of passes per truck load			Last Load(s) Bucket Fill Discount Factor	
Recommended Range 20 to 40 seconds									
Default Value***	30		Default Value***	1	Default Value***	7		Default Value***	0.75
					Proportion of Payload		PLLOSS	0.111	

Any default value can be replaced - user's choice

TEST CRITERIA: Select PLx value from table

PLX > PLLOSS

Shaded Values Indicate Test Fails - Sacrifice Marginal or Not Beneficial

Values of PLx

Number of Trucks	Truck Trip Time - Seconds 300	400	500	600	700	800	900	1000	1100	1200	1300	1400	1500	1600	1700	1800
2	0.200	0.150	0.120	0.100	0.086	0.075	0.067	0.060	0.055	0.050	0.046	0.043	0.040	0.038	0.035	0.033
3	0.300	0.225	0.180	0.150	0.129	0.113	0.100	0.090	0.082	0.075	0.069	0.064	0.060	0.056	0.053	0.050
4	0.400	0.300	0.240	0.200	0.171	0.150	0.133	0.120	0.109	0.100	0.092	0.086	0.080	0.075	0.071	0.067
5	0.500	0.375	0.300	0.250	0.214	0.188	0.167	0.150	0.136	0.125	0.115	0.107	0.100	0.094	0.088	0.083
6	0.600	0.450	0.360	0.300	0.257	0.225	0.200	0.180	0.164	0.150	0.138	0.129	0.120	0.113	0.106	0.100
7	0.700	0.525	0.420	0.350	0.300	0.263	0.233	0.210	0.191	0.175	0.162	0.150	0.140	0.131	0.124	0.117
8	0.800	0.600	0.480	0.400	0.343	0.300	0.267	0.240	0.218	0.200	0.185	0.171	0.160	0.150	0.141	0.133
9	0.900	0.675	0.540	0.450	0.386	0.338	0.300	0.270	0.245	0.225	0.208	0.193	0.180	0.169	0.159	0.150
10	1.000	0.750	0.600	0.500	0.429	0.375	0.333	0.300	0.273	0.250	0.231	0.214	0.200	0.188	0.176	0.167
11	1.100	0.825	0.660	0.550	0.471	0.413	0.367	0.330	0.300	0.275	0.254	0.236	0.220	0.206	0.194	0.183
12	1.200	0.900	0.720	0.600	0.514	0.450	0.400	0.360	0.327	0.300	0.277	0.257	0.240	0.225	0.212	0.200

Table 3.59 - Part 1
Shovel Cycle Time Calculator

DIGABILITY - SOFT/STOCKPILE

Loading Time Assessment - Seconds

	Soft	Medium	Hard	Extra Hard
Standard Bucket Load Calculated at 1.8 Tonnes per Bucket Cubic Metre at 100% Bucket Fill	4	6	8	12

*** Includes allowance to Dump of 3 Seconds and Spot On Return to Face of 3 Seconds**

		Standard Bucket	Standard Bucket Load	Operating Weight	Flywheel Power	Swing Performance	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*
					Swing Angle Degrees		50	60	70	80	90	100	110
	Unit	BCM	Tonnes	Tonnes	kW	RPM	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds
Manufacturer - Model													
P & H	4100XPB	56	100	1428	NA	2.7	19.7	20.9	22.1	23.4	24.6	25.8	27.1
	4100A	47	85	1108	NA	2.7	23.7	20.9	22.1	23.4	24.6	25.8	27.1
	2800XPB	35	63	1033	NA	2.7	23.7	20.9	22.1	23.4	24.6	25.8	27.1
	2300XPB	25	45	651	NA	3	24.6	21.7	22.8	23.9	25.0	26.1	27.2
Liebherr	R996	34	61	656	2240	3.5	23.5	21.4	23.3	25.2	27.1	29.0	31.0
	R995	23	41	429	1600	3.7	23.0	20.8	22.6	24.4	26.2	28.0	29.8
	R994B	18	32	306	1120	3.7	23.0	20.8	22.6	24.4	26.2	28.0	29.8
Komatsu	PC8000	42	76	710	3000	3.5	23.5	21.4	23.3	25.2	27.1	29.0	31.0
	PC5500	26	47	515	1880	3.4	23.8	21.8	23.7	25.7	27.6	29.6	31.6
	PC4000	22	40	385	1400	4	22.3	20.0	21.7	23.3	25.0	26.7	28.3
	PC3000	15	27	255	940	4.6	21.2	18.7	20.1	21.6	23.0	24.5	25.9
	PC1800-6	11	20	184	676	4.5	21.4	18.9	20.4	21.9	23.3	24.8	26.3
O & K	RH400	47	85	1000	3280	4.4	21.6	19.1	20.6	22.1	23.6	25.2	26.7
	RH340	34	61	552	2240	3.9	22.5	20.3	22.0	23.7	25.4	27.1	28.8
	RH200	26	47	522	1880	3.9	22.5	20.3	22.0	23.7	25.4	27.1	28.8
	RH170	21	38	374	1492	4.8	20.9	18.3	19.7	21.1	22.5	23.9	25.3
	RH120E	15	27	283	1008	4.7	21.1	18.5	19.9	21.3	22.8	24.2	25.6
Hitachi	EX8000	40	72	780	2800	3.2	24.4	22.5	24.6	26.7	28.8	30.8	32.9
	Ex5500	27	49	515	1870	3.3	24.1	22.1	24.1	26.2	28.2	30.2	32.2
	EX3600	21	38	350	1400	3.2	24.4	22.5	24.6	26.7	28.8	30.8	32.9
	EX2500	15	27	242	971	3.8	22.8	20.5	22.3	24.0	25.8	27.5	29.3
	Ex1900	11	20	186	720	4.7	21.1	18.5	19.9	21.3	22.8	24.2	25.6

Table 3.59 - Part 2
Shovel Cycle Time Calculator

DIGABILITY - MEDIUM

Standard Bucket Load Calculated at 1.8 Tonnes per Bucket Cubic Metre at 100% Bucket Fill

Loading Time Assessment - Seconds

Soft	Medium	Hard	Extra Hard
4	6	8	12

*** Includes allowance to Dump of 3 Seconds and Spot On Return to Face of 3 Seconds**

		Standard Bucket	Standard Bucket Load	Operating Weight	Flywheel Power	Swing Performance	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*
		Swing Angle Degrees					50	60	70	80	90	100	110
	Unit	BCM	Tonnes	Tonnes	kW	RPM	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds
Manufacturer - Model													
P & H	4100XPB	56	100	1428	NA	2.7	21.7	22.9	24.1	25.4	26.6	27.8	29.1
	4100A	47	85	1108	NA	2.7	23.7	22.9	24.1	25.4	26.6	27.8	29.1
	2800XPB	35	63	1033	NA	2.7	23.7	22.9	24.1	25.4	26.6	27.8	29.1
	2300XPB	25	45	651	NA	3	24.6	23.7	24.8	25.9	27.0	28.1	29.2
Liebherr	R996	34	61	656	2240	3.5	23.5	23.4	25.3	27.2	29.1	31.0	33.0
	R995	23	41	429	1600	3.7	23.0	22.8	24.6	26.4	28.2	30.0	31.8
	R994B	18	32	306	1120	3.7	23.0	22.8	24.6	26.4	28.2	30.0	31.8
Komatsu	PC8000	42	76	710	3000	3.5	23.5	23.4	25.3	27.2	29.1	31.0	33.0
	PC5500	26	47	515	1880	3.4	23.8	23.8	25.7	27.7	29.6	31.6	33.6
	PC4000	22	40	385	1400	4	22.3	22.0	23.7	25.3	27.0	28.7	30.3
	PC3000	15	27	255	940	4.6	21.2	20.7	22.1	23.6	25.0	26.5	27.9
	PC1800-6	11	20	184	676	4.5	21.4	20.9	22.4	23.9	25.3	26.8	28.3
O & K	RH400	47	85	1000	3280	4.4	21.6	21.1	22.6	24.1	25.6	27.2	28.7
	RH340	34	61	552	2240	3.9	22.5	22.3	24.0	25.7	27.4	29.1	30.8
	RH200	26	47	522	1880	3.9	22.5	22.3	24.0	25.7	27.4	29.1	30.8
	RH170	21	38	374	1492	4.8	20.9	20.3	21.7	23.1	24.5	25.9	27.3
	RH120E	15	27	283	1008	4.7	21.1	20.5	21.9	23.3	24.8	26.2	27.6
Hitachi	EX8000	40	72	780	2800	3.2	24.4	24.5	26.6	28.7	30.8	32.8	34.9
	Ex5500	27	49	515	1870	3.3	24.1	24.1	26.1	28.2	30.2	32.2	34.2
	EX3600	21	38	350	1400	3.2	24.4	24.5	26.6	28.7	30.8	32.8	34.9
	EX2500	15	27	242	971	3.8	22.8	22.5	24.3	26.0	27.8	29.5	31.3
	Ex1900	11	20	186	720	4.7	21.1	20.5	21.9	23.3	24.8	26.2	27.6

Table 3.59 - Part 3

Shovel Cycle Time Calculator

DIGABILITY - HARD

	Loading Time Assessment - Seconds			
	Soft	Medium	Hard	Extra Hard
Standard Bucket Load Calculated at 1.8 Tonnes per Bucket Cubic Metre at 100% Bucket Fill	4	6	8	12

* Includes allowance to Dump of 3 Seconds and Spot On Return to Face of 3 Seconds

		Standard Bucket	Standard Bucket Load	Operating Weight	Flywheel Power	Swing Performance	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*
					Swing Angle Degrees		50	60	70	80	90	100	110
	Unit	BCM	Tonnes	Tonnes	kW	RPM	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds
Manufacturer	Model												
P & H	4100XPB	56	100	1428	NA	2.7	23.7	24.9	26.1	27.4	28.6	29.8	31.1
	4100A	47	85	1108	NA	2.7	23.7	24.9	26.1	27.4	28.6	29.8	31.1
	2800XPB	35	63	1033	NA	2.7	23.7	24.9	26.1	27.4	28.6	29.8	31.1
	2300XPB	25	45	651	NA	3	24.6	25.7	26.8	27.9	29.0	30.1	31.2
Liebherr	R996	34	61	656	2240	3.5	23.5	25.4	27.3	29.2	31.1	33.0	35.0
	R995	23	41	429	1600	3.7	23.0	24.8	26.6	28.4	30.2	32.0	33.8
	R994B	18	32	306	1120	3.7	23.0	24.8	26.6	28.4	30.2	32.0	33.8
Komatsu	PC8000	42	76	710	3000	3.5	23.5	25.4	27.3	29.2	31.1	33.0	35.0
	PC5500	26	47	515	1880	3.4	23.8	25.8	27.7	29.7	31.6	33.6	35.6
	PC4000	22	40	385	1400	4	22.3	24.0	25.7	27.3	29.0	30.7	32.3
	PC3000	15	27	255	940	4.6	21.2	22.7	24.1	25.6	27.0	28.5	29.9
	PC1800-6	11	20	184	676	4.5	21.4	22.9	24.4	25.9	27.3	28.8	30.3
O & K	RH400	47	85	1000	3280	4.4	21.6	23.1	24.6	26.1	27.6	29.2	30.7
	RH340	34	61	552	2240	3.9	22.5	24.3	26.0	27.7	29.4	31.1	32.8
	RH200	26	47	522	1880	3.9	22.5	24.3	26.0	27.7	29.4	31.1	32.8
	RH170	21	38	374	1492	4.8	20.9	22.3	23.7	25.1	26.5	27.9	29.3
	RH120E	15	27	283	1008	4.7	21.1	22.5	23.9	25.3	26.8	28.2	29.6
Hitachi	EX8000	40	72	780	2800	3.2	24.4	26.5	28.6	30.7	32.8	34.8	36.9
	Ex5500	27	49	515	1870	3.3	24.1	26.1	28.1	30.2	32.2	34.2	36.2
	EX3600	21	38	350	1400	3.2	24.4	26.5	28.6	30.7	32.8	34.8	36.9
	EX2500	15	27	242	971	3.8	22.8	24.5	26.3	28.0	29.8	31.5	33.3
	Ex1900	11	20	186	720	4.7	21.1	22.5	23.9	25.3	26.8	28.2	29.6

Table 3.59 - Part 4
Shovel Cycle Time Calculator

DIGABILITY - VERY HARD

Loading Time Assessment - Seconds

	Soft	Medium	Hard	Extra Hard
Standard Bucket Load Calculated at 1.8 Tonnes per Bucket Cubic Metre at 100% Bucket Fill	4	6	8	12

* Includes allowance to Dump of 3 Seconds and Spot On Return to Face of 3 Seconds

		Standard Bucket	Standard Bucket Load	Operating Weight	Flywheel Power	Swing Performance	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*	Total Cycle Time*
					Swing Angle Degrees		50	60	70	80	90	100	110
	Unit	BCM	Tonnes	Tonnes	kW	RPM	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds	Seconds
Manufacturer - Model													
P & H	4100XPB	56	100	1428	NA	2.7	27.7	28.9	30.1	31.4	32.6	33.8	35.1
	4100A	47	85	1108	NA	2.7	27.7	28.9	30.1	31.4	32.6	33.8	35.1
	2800XPB	35	63	1033	NA	2.7	27.7	28.9	30.1	31.4	32.6	33.8	35.1
	2300XPB	25	45	651	NA	3	28.6	29.7	30.8	31.9	33.0	34.1	35.2
Liebherr	R996	34	61	656	2240	3.5	27.5	29.4	31.3	33.2	35.1	37.0	39.0
	R995	23	41	429	1600	3.7	27.0	28.8	30.6	32.4	34.2	36.0	37.8
	R994B	18	32	306	1120	3.7	27.0	28.8	30.6	32.4	34.2	36.0	37.8
Komatsu	PC8000	42	76	710	3000	3.5	27.5	29.4	31.3	33.2	35.1	37.0	39.0
	PC5500	26	47	515	1880	3.4	27.8	29.8	31.7	33.7	35.6	37.6	39.6
	PC4000	22	40	385	1400	4	26.3	28.0	29.7	31.3	33.0	34.7	36.3
	PC3000	15	27	255	940	4.6	25.2	26.7	28.1	29.6	31.0	32.5	33.9
	PC1800-6	11	20	184	676	4.5	25.4	26.9	28.4	29.9	31.3	32.8	34.3
O & K	RH400	47	85	1000	3280	4.4	25.6	27.1	28.6	30.1	31.6	33.2	34.7
	RH340	34	61	552	2240	3.9	26.5	28.3	30.0	31.7	33.4	35.1	36.8
	RH200	26	47	522	1880	3.9	26.5	28.3	30.0	31.7	33.4	35.1	36.8
	RH170	21	38	374	1492	4.8	24.9	26.3	27.7	29.1	30.5	31.9	33.3
	RH120E	15	27	283	1008	4.7	25.1	26.5	27.9	29.3	30.8	32.2	33.6
Hitachi	EX8000	40	72	780	2800	3.2	28.4	30.5	32.6	34.7	36.8	38.8	40.9
	Ex5500	27	49	515	1870	3.3	28.1	30.1	32.1	34.2	36.2	38.2	40.2
	EX3600	21	38	350	1400	3.2	28.4	30.5	32.6	34.7	36.8	38.8	40.9
	EX2500	15	27	242	971	3.8	26.8	28.5	30.3	32.0	33.8	35.5	37.3
	Ex1900	11	20	186	720	4.7	25.1	26.5	27.9	29.3	30.8	32.2	33.6

Table 3.70 Probability That Exactly N Units Will Be Available

Probable Unit Availability 0.80

FLEET SIZE

N	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30
0	0.040	0.008	0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
1	0.320	0.096	0.025	0.006	0.002	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
2	0.640	0.384	0.154	0.051	0.015	0.004	0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
3		0.512	0.410	0.205	0.082	0.029	0.009	0.003	0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
4			0.410	0.410	0.246	0.115	0.046	0.017	0.006	0.002	0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
5				0.328	0.393	0.275	0.147	0.066	0.026	0.010	0.003	0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
6					0.262	0.367	0.294	0.176	0.088	0.039	0.016	0.006	0.002	0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
7						0.210	0.336	0.302	0.201	0.111	0.053	0.023	0.009	0.003	0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
8							0.168	0.302	0.302	0.221	0.133	0.069	0.032	0.014	0.006	0.002	0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
9								0.134	0.268	0.295	0.236	0.154	0.086	0.043	0.020	0.008	0.003	0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
10									0.107	0.236	0.283	0.246	0.172	0.103	0.055	0.027	0.012	0.005	0.002	0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
11										0.086	0.206	0.268	0.250	0.188	0.120	0.068	0.035	0.017	0.007	0.003	0.001	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
12											0.069	0.179	0.250	0.250	0.200	0.136	0.082	0.044	0.022	0.010	0.005	0.002	0.001	0.000	0.000	0.000	0.000	0.000	0.000
13												0.055	0.154	0.231	0.246	0.209	0.151	0.095	0.055	0.029	0.014	0.006	0.003	0.001	0.000	0.000	0.000	0.000	0.000
14													0.044	0.132	0.211	0.239	0.215	0.164	0.109	0.065	0.036	0.018	0.009	0.004	0.002	0.001	0.000	0.000	0.000
15														0.035	0.113	0.191	0.230	0.218	0.175	0.122	0.077	0.044	0.024	0.012	0.006	0.003	0.001	0.000	0.000
16															0.028	0.096	0.172	0.218	0.218	0.183	0.134	0.088	0.053	0.029	0.015	0.008	0.004	0.002	0.001
17																0.023	0.081	0.154	0.205	0.216	0.190	0.145	0.100	0.062	0.036	0.019	0.010	0.005	0.002
18																	0.018	0.068	0.137	0.192	0.211	0.194	0.155	0.111	0.072	0.043	0.024	0.013	0.006
19																		0.014	0.058	0.121	0.178	0.204	0.196	0.163	0.121	0.082	0.051	0.030	0.016
20																			0.012	0.048	0.107	0.163	0.196	0.196	0.170	0.131	0.092	0.059	0.035
21																				0.009	0.041	0.093	0.149	0.187	0.194	0.175	0.140	0.101	0.068
22																					0.007	0.034	0.081	0.136	0.176	0.191	0.178	0.147	0.111
23																						0.006	0.028	0.071	0.123	0.166	0.186	0.179	0.154
24																							0.005	0.024	0.061	0.111	0.155	0.179	0.179
25																								0.004	0.020	0.053	0.099	0.144	0.172
26																									0.003	0.016	0.046	0.088	0.133
27																										0.002	0.014	0.039	0.079
28																											0.002	0.011	0.034
29																												0.002	0.009
30																													0.001

Table 3.71 - Probability That At Least N Units Will Be Available

Probable Unit Availability **0.80**

FLEET SIZE

N	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30
0	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
1	0.960	0.992	0.998	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
2	0.640	0.896	0.973	0.993	0.998	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
3		0.512	0.819	0.942	0.983	0.995	0.999	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
4			0.410	0.737	0.901	0.967	0.990	0.997	0.999	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
5				0.328	0.655	0.852	0.944	0.980	0.994	0.998	0.999	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
6					0.262	0.577	0.797	0.914	0.967	0.988	0.996	0.999	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
7						0.210	0.503	0.738	0.879	0.950	0.981	0.993	0.998	0.999	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
8							0.168	0.436	0.678	0.839	0.927	0.970	0.988	0.996	0.999	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
9								0.134	0.376	0.617	0.795	0.901	0.956	0.982	0.993	0.997	0.999	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
10									0.107	0.322	0.558	0.747	0.870	0.939	0.973	0.989	0.996	0.998	0.999	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
11										0.086	0.275	0.502	0.698	0.836	0.918	0.962	0.984	0.993	0.997	0.999	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
12											0.069	0.234	0.448	0.648	0.798	0.894	0.949	0.977	0.990	0.996	0.998	0.999	1.000	1.000	1.000	1.000	1.000	1.000	1.000
13												0.055	0.198	0.398	0.598	0.758	0.867	0.932	0.968	0.986	0.994	0.997	0.999	1.000	1.000	1.000	1.000	1.000	1.000
14													0.044	0.167	0.352	0.549	0.716	0.837	0.913	0.957	0.980	0.991	0.996	0.998	0.999	1.000	1.000	1.000	1.000
15														0.035	0.141	0.310	0.501	0.673	0.804	0.891	0.944	0.973	0.987	0.994	0.998	0.999	1.000	1.000	1.000
16															0.028	0.118	0.271	0.455	0.630	0.769	0.867	0.928	0.964	0.983	0.992	0.997	0.999	0.999	1.000
17																0.023	0.099	0.237	0.411	0.586	0.733	0.840	0.911	0.953	0.977	0.989	0.995	0.998	0.999
18																	0.018	0.083	0.206	0.370	0.543	0.695	0.811	0.891	0.941	0.970	0.985	0.993	0.997
19																		0.014	0.069	0.179	0.332	0.501	0.656	0.780	0.869	0.926	0.961	0.980	0.991
20																			0.012	0.058	0.154	0.297	0.460	0.617	0.747	0.844	0.910	0.951	0.974
21																				0.009	0.048	0.133	0.264	0.421	0.577	0.713	0.818	0.892	0.939
22																					0.007	0.040	0.115	0.234	0.383	0.539	0.678	0.790	0.871
23																						0.006	0.033	0.098	0.207	0.348	0.501	0.643	0.761
24																							0.005	0.027	0.084	0.182	0.315	0.463	0.607
25																								0.004	0.023	0.072	0.160	0.284	0.428
26																									0.003	0.019	0.061	0.140	0.255
27																										0.002	0.015	0.052	0.123
28																											0.002	0.013	0.044
29																												0.002	0.011
30																													0.001

Table 3.75 Time Definitions Adopted by a Typical Dispatch System

(From the Modular Mining Dispatch System used by Western Premier Coal Limited, Collie Western Australia)

<table>
<tr><td colspan="9">TOTAL TIME (CALENDAR)</td></tr>
<tr><td colspan="8">ROSTERED (SCHEDULED)
(Total Time Equipment is Scheduled in Mine Plans</td><td rowspan="4">NON-ROSTERED
(Time that workforce is not available for work e.g., public holidays)</td></tr>
<tr><td colspan="4">AVAILABLE
(The accumulated time that equipment is operational)</td><td colspan="3">MAINTENANCE DOWNTIME
(Non-available – The time that equipment is not in operational condition)</td><td rowspan="3">LOST TIME
(Potential operating time lost due to Environmental, Fires, Flood, Industrial Action)</td></tr>
<tr><td colspan="2">UTILISED
(Time that equipment is manned and working)</td><td colspan="2">IDLE
(Time that equipment is in operational condition but not working)</td><td colspan="2">UNPLANNED
(Time equipment is non-operational due to mechanical fault)</td><td rowspan="2">PLANNED
(Time equipment is not available due to maintenance requirement)</td></tr>
<tr><td>DIRECT OPERATING
(Time when production is realised)</td><td>INDIRECT OPERATING
(Non-productive time necessary for production operations)</td><td>MANNED
(Time when equipment and operator are idle)</td><td>UNMANNED
(Time when equipment only is idle)</td><td>ACCIDENT DAMAGE/ OPPORTUNE MAINT’NCE</td><td>BREAKDOWN</td></tr>
<tr><td colspan="2">OPERATING</td><td colspan="7">NON-OPERATING</td></tr>
<tr><td colspan="3">MANNED</td><td colspan="6">UNMANNED</td></tr>
</table>

NOTES

Rostered Time	Total Hours accounted for in Planning Schedules.	**MAJOR PRODUCTION AND MAINTENANCE MEASURES**
Available Time	Accumulated hours that equipment is available to work within Rostered time.	**AVAILABILITY** Available Hours = Rostered Hours– (Lost Time + Maintenance Downtime) Availability % = Available Hours / (Available Hours + Maintenance Downtime)
Direct Operating Time	Time necessary to fulfill normal production operations.	**UTILISATION OF AVAILABILITY** Utilized Hours = Available Hours – Idle Utilisation of Available Hours % = Utilised Hours / Available Hours
Indirect operating time	Includes controllable but unavoidable delays to operations.	**UTILISED** Utilised Hours = Available Hours – Idle Utilisation % = Utilised Hours / (Rostered – Lost Time)
Idle and Unplanned Maintenance	Unproductive time – may be due to surplus equipment (unmanned idle time and opportune maintenance) Excess operator delays (shift change, crib).	**Productivity = Production (BCM or Tonnes) / Utilized Hours**
Planned Maintenance	Organized downtime incorporated into Mine Planning schedules (Should included routine servicing and preventative maintenance).	

Table 5.10 Cost Criteria For Sacrificing Bucket Loads

For specific case of a Cat 793C on Course 2 as described in 3.3.10 in the text

Bucket Cycle Time		
Recommended Range 20 to 40 seconds		
Default Value***	30	
***Any default value can be replaced - user's choice		

Number of Passes Sacrificed	
Default Value***	1

Proportion of Payload Sacrificed			Last Load(s) Bucket Fill Factor	
Pass Number:				
Default Value***	7		Default Value***	0.75
Proportion of Payload	PL_{LOSS}	0.111	HCI =0.880	

TEST CRITERIA: Select HC_X value from table below

$HC_X > HC_{LOSS}$

$HC_X > 0.098$ - In the specific case

HCI =0.817

Haul Cost Loss (HC_{LOSS}) = PL_{LOSS} x Haul Cost Index (HCI) from Table 5.4.2.1 - See text for explanation

HC_{LOSS} = 0.098

Shaded Values Indicate Test Fails - Sacrifice Marginal or Not Beneficial

Cat 793C – Course 2 - Haul Cost Benefit $HC_X = PL_X$ x Haul Cost Index from Table 5.2.4.1.A - See text for explanation

Number of Trucks	Truck Trip Time - Seconds 300	400	500	600	700	800	900	1000	1100	1200	1300	1400	1500	1600	1700	1800
2	0.163	0.123	0.098	0.082	0.070	0.061	0.054	0.049	0.045	0.041	0.038	0.035	0.033	0.031	0.029	0.027
3	0.245	0.184	0.147	0.123	0.105	0.092	0.082	0.074	0.067	0.061	0.057	0.053	0.049	0.046	0.043	0.041
4	0.327	0.245	0.196	0.163	0.140	0.123	0.109	0.098	0.089	0.082	0.075	0.070	0.065	0.061	0.058	0.054
5	0.409	0.306	0.245	0.204	0.175	0.153	0.136	0.123	0.111	0.102	0.094	0.088	0.082	0.077	0.072	0.068
6	0.490	0.368	0.294	0.245	0.210	0.184	0.163	0.147	0.134	0.123	0.113	0.105	0.098	0.092	0.087	0.082
7	0.572	0.429	0.343	0.286	0.245	0.214	0.191	0.172	0.156	0.143	0.132	0.123	0.114	0.107	0.101	0.095
8	0.654	0.490	0.392	0.327	0.280	0.245	0.218	0.196	0.178	0.163	0.151	0.140	0.131	0.123	0.115	0.109
9	0.735	0.551	0.441	0.368	0.315	0.276	0.245	0.221	0.201	0.184	0.170	0.158	0.147	0.138	0.130	0.123
10	0.817	0.613	0.490	0.409	0.350	0.306	0.272	0.245	0.223	0.204	0.189	0.175	0.163	0.153	0.144	0.136
11	0.899	0.674	0.539	0.449	0.385	0.337	0.300	0.270	0.245	0.225	0.207	0.193	0.180	0.169	0.159	0.150
12	0.980	0.735	0.588	0.490	0.420	0.368	0.327	0.294	0.267	0.245	0.226	0.210	0.196	0.184	0.173	0.163

Table 5.11A Theoretical Truck Numbers for Range of Selected Truck Numbers (Integer) & for Range of Cost Ratios - R - Loading Equipment,/Truck Costs

PART A

	Cost Ratio Index R = 1																				
Number of Trucks Selected - n	**1**			**2**			**3**			**4**			**5**			**6**			**7**		
	t_1	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**
1	0.00	1.00	1.50																		
2				1.50	2.00	2.67															
3							2.67	3.00	3.75												
4										3.75	4.00	4.80									
5													4.80	5.00	5.83						
6																5.83	6.00	6.86			
7																			6.86	7.00	7.88
Number of Trucks Selected - n	**10**				**20**			**40**			**60**			**80**			**100**				
	t_1	**t=n**	**t_2**		**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**		
10	9.90	10.00	10.91																		
20					19.95	20.00	20.95														
40								39.98	40.00	40.98											
60											59.98	60.00	60.98								
80														79.99	80.00	80.99					
100																	99.99	100.00	100.99		
Summary - R = 1																					
n	**1**	**2**	**3**	**4**	**5**	**6**	**7**														
t_1	0.00	1.50	2.67	3.75	4.80	5.83	6.86														
t=n	1.00	2.00	3.00	4.00	5.00	6.00	7.00														
t_2	1.50	2.67	3.75	4.80	5.83	6.86	7.88														
	20	**40**	**60**	**80**	**100**																
t_1	19.95	39.98	59.98	79.99	99.99																
t=n	20.00	40.00	60.00	80.00	100.00																
t_2	20.95	40.98	60.98	80.99	100.99																

Table 5.11B Theoretical Truck Numbers for Range of Selected Truck Numbers (Integer) & for Range of Cost Ratios - R - Loading Equipment/Truck Costs

PART B

	Cost Ratio Index R = 2																				
Number of Trucks Selected - n	**1**			**2**			**3**			**4**			**5**			**6**			**7**		
	t_1	t=n	t_2	t_1	t=n	t_2	t_1	t=n	t_2	t_1	t=n	t_2	t_1	t=n	t_2	t_1	t=n	t_2	t_1	t=n	t_2
1	0.00	1.00	1.33																		
2				1.33	2.00	2.50															
3							2.50	3.00	3.60												
4										3.60	4.00	4.67									
5													4.67	5.00	5.71						
6																5.71	6.00	6.75			
7																			6.75	7.00	7.78
Number of Trucks Selected - n	**10**				**20**			**40**			**60**			**80**			**100**				
	t_1	**t=n**	**t_2**		**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**		
10	9.82	10.00	10.83																		
20					19.90	20.00	20.91														
40								39.95	40.00	40.95											
60											59.97	60.00	60.97								
80														79.98	80.00	80.98					
100																	99.98	100.00	100.98		
Summary - R = 2																					
	1	**2**	**3**	**4**	**5**	**6**	**7**														
t_1	0.00	1.33	2.50	3.60	4.67	5.71	6.75														
t=n	1.00	2.00	3.00	4.00	5.00	6.00	7.00														
t_2	1.33	2.50	3.60	4.67	5.71	6.75	7.78														
	20	**40**	**60**	**80**	**100**																
t_1	19.90	39.95	59.97	79.98	99.98																
t=n	20.00	40.00	60.00	80.00	100.00																
t_2	20.91	40.95	60.97	80.98	100.98																

Table 5.11C Theoretical Truck Numbers for Range of Selected Truck Numbers (Integer) & for Range of Cost Ratios - R - Loading Equipment/Truck Costs

PART C

	Cost Ratio Index R = 3																				
Number of Trucks Selected - n	**1**			**2**			**3**			**4**			**5**			**6**			**7**		
	t_1	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**
1	0.00	1.00	1.25																		
2				1.25	2.00	2.40															
3							2.40	3.00	3.50												
4										3.50	4.00	4.57									
5													4.57	5.00	5.63						
6																5.63	6.00	6.67			
7																			6.67	7.00	7.70
Number of Trucks Selected - n	**10**				**20**			**40**			**60**			**80**			**100**				
	t_1	**t=n**	**t_2**		**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**	**t_1**	**t=n**	**t_2**		
10	9.75	10.00	10.77																		
20					19.86	20.00	20.87														
40								39.93	40.00	40.93											
60											59.95	60.00	60.95								
80														79.96	80.00	80.96					
100																	99.97	100.00	100.97		
Summary - R = 3																					
	1	**2**	**3**	**4**	**5**	**6**	**7**														
t_1	0.00	1.25	2.40	3.50	4.57	5.63	6.67														
t=n	1.00	2.00	3.00	4.00	5.00	6.00	7.00														
t_2	1.25	2.40	3.50	4.57	5.63	6.67	7.70														
	20	**40**	**60**	**80**	**100**																
t_1	19.86	39.93	59.95	79.96	99.97																
t=n	20.00	40.00	60.00	80.00	100.00																
t_2	20.87	40.93	60.95	80.96	100.97																

MATHEMATICAL PRINCIPLES - NOTES

Includes Table of Contents

TABLE OF CONTENTS

MPN 1 Introduction

Statistical theory and methods have been studied, described and applied to the limited degree necessary to facilitate analysis, interpretation and conclusive outcomes of the research. Proof of the propositions adopted for research purposes has been left for the reader to pursue in the many excellent texts on the subject some of which have been acknowledged and listed in References in the thesis, specifically referred to in the thesis and within these notes.

The mathematical principles visited in the research are not presented as an exhaustive or comprehensive treatment – only as a means to facilitate the research outcomes and assist the reader.

MPN 2 Research and Relevant Statistical Mathematics

Definitive understanding of productivity and performance criteria for loading and hauling equipment is an essential basis for selecting equipment items from available options. Initial comparisons are generally made in terms of expected values, arithmetic means, adopted for deterministic methods to estimate productivity and performance.

Previous experience with investigation of productivity in open pit mining operations, and how to effect improvement, has led to an understanding that deterministic analysis based on expected values is inadequate for the level of understanding necessary to manage and improve operational performance. A stochastic approach to analysis of data and interpretation of the results in terms of operational criteria and protocols is required. Preliminary investigation of load and haul operations by the author in terms of performance and efficiency commenced some years ago. Recent outcomes from feasibility and improvement studies were sufficiently interesting and encouraging to formalize a research project that has ultimately coalesced as the subject of this thesis. Research described in the thesis includes investigation of the bucket load – truck payload relationship, bucket cycle times and truck loading time, influence of number of bucket passes on productivity and efficiency of load and haul operations and related issues. Early investigations recognized the need to adopt descriptive statistics to identify, test and adopt probability distributions used to model observed empirical distributions.

The need for sufficient understanding of the behaviour of mining operational events observed as empirical data distributions ultimately led to frequency analysis using convenient, appropriate frequency intervals and construction of histograms to be compared with continuous model/test distributions. Many of the empirical data distributions rearranged as frequency distributions observed in loading and hauling functions exhibit symmetry and central tendency that indicate observed data is, in many cases, normally distributed.

Sample theory has proved most useful in understanding the relationship between number of bucket passes and dispersion of truck payload distributions; also between bucket cycle times and truck loading times.

Application of Small (Exact) Sample Theory has not been necessary as:

- Generally empirical distributions of bucket loads and truck payloads can be modelled by normal distributions – refer Section 3.2.8.
- Empirical distributions of bucket cycle times and loading times are substantially positively skewed exhibiting significant high values and affected by anomalous outliers (likely a separate random variable distribution) that are non-intrinsic to actual loading operations refer Section 3.2.9.

Analysis of this latter case was facilitated by:

- Testing with gamma distributions that, on realising qualitative acceptance exhibited significantly high shape and scale parameters indicating underlying central tendency of empirical data, i.e., a tendency to skewed normal distributions.
- Objective filtering with regard to origins of anomalous outliers – aiming to separate from the basic intrinsic loading function data affected by activities non-intrinsic to the basic loading operation.

As shown in Section 3.2.9 in the thesis, manifest in the relevant tables and illustrated by associated figures, filtering outcomes can expose incipient central tendency and a drift towards symmetry.

Truck payloads can be viewed as small samples of bucket loads and truck loading time as small samples of bucket cycle times.

Because of normality of bucket loads and, as a consequence normally distributed truck payloads, application of small (exact) sample theory was unnecessary.

Highly skewed bucket cycle times and truck loading times might have been advantageously analysed using Student's (Gosset's) "t" distribution as a model to compare empirical data sets and suitable test models in terms of the "t" statistic.

Proceeding to this level of comparative analysis was considered unnecessary as acceptable results and understanding sufficient for the purposes of the research was achieved by filtering. Operating on the data without filtering would have been tacit acceptance that anomalous outliers in bucket cycle time and loading time distributions are intrinsic to the basic loading function when experienced understanding of anomalous time outliers recognizes such acceptance as unreasonable.

When reviewing Student's "t" distribution for applicability it was noted that, compared with a normal distribution, the family of "t" distributions:

- Have a platykurtic trend as sample numbers decrease.
- Tails extend along the "t" value axis as sample number (N) decreases indicative of increase distribution dispersion.

So, consistent with intuitive understanding and empirical observations, dispersion of truck loading times drawn from a population of bucket cycle times will tend to increase as the number of bucket passes to load trucks decrease. The result will be increased variability in truck loading times as bucket-cycle time increases. This is confirmed by the analysis and interpretation in Section 3.2.9.

Identification of applicable test distributions to compare with empirical data from production operations was, initially, essentially qualitative using histograms and assumed continuous test distribution models – supported by evidence such as relatively small standard deviation as demonstrated comparatively between empirical data and assumed continuous distribution models by coefficients of variation (CV).

A more reliable confirmation of applicability of test distribution forms was considered necessary. From the many test procedures available the non-parametric Kolmogorov – Smirnov (KS) test was chosen for the following reasons:

- Recommendation by Ritu Gupta, a statistical mathematician at Curtin University of Technology, Bentley campus (Gupta, 2004).
- Readily available within SPSS software application – licence extended by Curtin University.
- User friendliness.
- Convenience of application.
- Necessity for only a general understanding of statistical mathematics underlying the distribution test procedure.

Further details of the procedure and outcomes of the KS test are provided in a separate section in the Appendix entitled: "Distribution Testing – Kolmogorov – Smirnov Test".

MPN3 Basic Statistical Theory

References consulted as theoretical basis for analysis and interpretation of distribution data in the course of the research include:

- Devore, Jay L, *Probability and Statistics for Engineering and the Sciences,* Duxbury (Devore, 1999)
- Chou, Ya-lun, *Statistical Analysis for Business and Economic Applications,* Holt, Rinehart and Winston (Chou, 1969)
- Spiegel, Murray R, Theory and Problems of Statistics, Schaum's Outline Series, McGraw-Hill (Spiegel, 1961)

The following theory is used as the basis for analysis in Section 3.2.8 and Section 3.2.9 in the thesis text.

The introductory statistical theory below is an edited extract from a paper by the author entitled *Four Pass Loading – Must have or Myth* (Hardy#1, 2003). A complete copy of the paper is available in Supplementary Information as a file on the CD accompanying this thesis – inside the back cover. The research described in the thesis required a more complete and extensive review of statistical theory as described below.

MPN 3.1 Application of Sample Theory to Bucket and Truck Payloads

Considering the general relationship between truck payloads and bucket loads; and assuming that normal distribution models apply to all actual recorded data, then:

$$\mu_s = \mu_p/N \quad \text{---------------------------------------(1A)}$$

$$\sigma_s = \sigma_p/N \quad \text{--(1B)}$$

Where:

μ_s = Mean of sample mean of bucket loads

μ_p = Mean of truck payloads

σ_s = Standard Deviation of the small sample mean of bucket loads, i.e. the mean of bucket loads in a truck payload.

σ_p = Standard deviation of truck payloads

N = is the number of bucket loads (small sample) in a truck payload

The above is a naive arithmetical relationship used in the research only for checking calculations and as a scale reference.

MPN 3.2 Sampling Distribution of Sample Means

Sample theory provides that, for samples (N) from a large sample (N_p), (where N_p>N), the standard deviation of sample means:

$$\sigma_s = [\sigma/\sqrt{N}][\sqrt{\{(N_p - N)/(N_p - 1)\}}] \quad \text{-------------(2)}$$

where:

σ = Standard Deviation of bucket loads in the total bucket-load population i.e. all the bucket loads in all truck payloads.

N_p = number of bucket loads in the total bucket-load sample (a large number)

The expression $[\sqrt{\{(N_p - N)/(N_p - 1)\}}]$ is a finite population correction factor (FPCF) that allows for:

- Finite populations sampled without replacement.
- Samples that are relatively large, say greater than 5% of the population.

When $N_p >>> N$ (say $N \approx < 5\%$ of N_p) or when $N_p \rightarrow \infty$, (the case when considering bucket loads in the context of the research), so $FPC \rightarrow 1$, the above equation simplifies to:

$$\sigma_s = \sigma/\sqrt{N} \quad \text{---------------(3)}$$

In examination of the distribution of the sample mean, Devore proposes the following relationships for a random sample from a distribution with mean value μ and standard deviation σ (Devore, 1999):

Expected Value of Sample Mean: $\mu_s = \mu$ --------------------(4)

Expected Value of Standard Deviation $\sigma_s = \sigma/\sqrt{N}$ ---------------(5)

Expected Value of Total (Sum) of Sample $T_s = N \cdot \mu$ ----------------(6)

Expected Value of Standard Deviation of Totals $\sigma_T = \sqrt{N} \cdot \sigma$ ----(7)

Context of Expected Value ≡ Mean in above equations (4) to (7).

In the context of the research:

μ_s = Mean of bucket loads in a small sample (truck payload).

μ = Mean of bucket loads in the total bucket-load (large) population

T_s = Mean of totals of bucket loads in a small sample, i.e., mean of truck payloads.

The above context clarification can be generally adopted for bucket cycle times and truck loading times by substituting the appropriate descriptive statistics and giving due regard to relevant differences between load and time data.

MPN 3.3 Dispersion Affect of N

In terms of statistics applicable to totals of random samples from a population:

And considering two cases: N_a and N_b

Coefficient of Variation $CV = \sigma/\mu$ ----------------------(8)

Subscripts applied to variables in Equation 8 indicate context.

$CV_a = \sigma_a/T_a$ $CV_b = \sigma_b/T_b$

$T_a = N_a \cdot \mu$ $T_b = N_b \cdot \mu$

Substituting and transposing:

$CV_a \cdot T_a/\sigma_a = 1 = CV_b \cdot T_b/\sigma_b$

$CV_a \cdot N_a \cdot \mu/\sigma_a = CV_b \cdot N_b \cdot \mu/\sigma_b$

$CV_b = CV_a \cdot N_a / N_b \cdot \sigma_b/\sigma_a$ -------------------------------------- (9)

$\sigma_a = \sqrt{N_a} \cdot \sigma \qquad \sigma_b = \sqrt{N_b} \cdot \sigma$

$\sigma_b/\sigma_4 = \sqrt{N_b}/\sqrt{N_a}$ ---(10)

Substituting in (9) with (10):

$CV_b = CV_a \cdot N_a/N_b \cdot \sqrt{N_b}/\sqrt{N_a}$

$CV_b = CV_a \cdot \sqrt{N_a}/\sqrt{N_b}$ ---(11)

From Equation (11) dispersion of the distribution of small sample means obviously decreases with the increase in the number of observations in each small sample.

Devore's Figure 5.14 below illustrates the outcome of analysis (Devore, 1999)

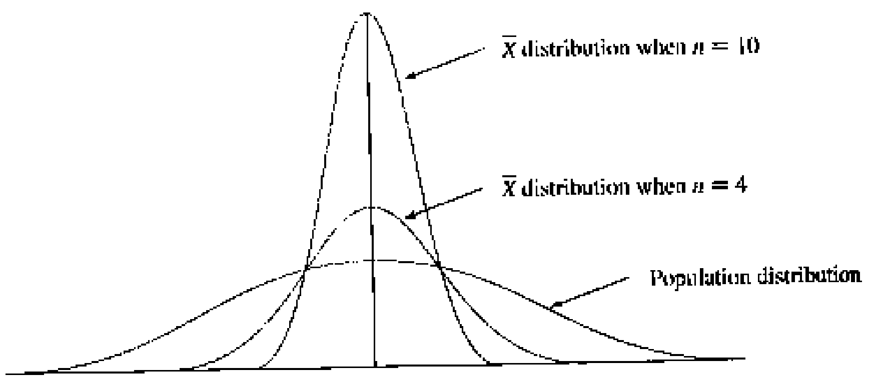


Figure 5.14 A normal population distribution and $\bar{X}$ sampling distributions

.

Consider two cases: $N_a = 4$ and $N_b = 6$ and applying equation (11):

$\sqrt{N_a} = 2$; and $\sqrt{N_b} = 2.45$

$CV_6 = CV_4 \cdot 2/2.45$

$= 0.81 CV_4$

So confirming that increasing the number in each random sample reduces the dispersion of sample means. Equation (11) above was used to calculate relative CV indices for a range of bucket passes as shown by Table 3.33 and illustrated by Figure 3.27.

Two points to be noted:

1. As N increases the distribution of sample means is asymptotically normal – an expression of the Central Limit Theorem (CLT) (Spiegel, 1961) – see notes below on CLT and implications for the research.
2. If the population of bucket loads is normal so will be any sub-population distribution; also the distribution of sample means will be normal regardless of how small the sample size may be.

The implications of equation (11) are discussed at some length in the thesis. In particular the research benefits from mathematical proof that truck payload dispersion tends to reduce in inverse ratio to the square root of the number of bucket passes and the subsequent confirmation by empirical data from actual loading and hauling operations. Similar mathematical propositions were also applied to the relevant extent; and with due regard to validity, in analysing bucket cycle times and truck loading times.

MPN 3.4 Central Limit Theorem (CLT)

The CLT has been described as: "the most important theorem of probability" (Devore, 1999). It formalizes the central tendency of sample means and is of general relevance to the research and related issues as described below. The proposition has tended to become a given in the simple application of probability to predicting behaviour of random variables observed as empirical data from a wide range of production activities including mining operations.

The following proposition is a formal statement of the CLT:

If a sample of N values is selected randomly from a population distribution of a random variable having mean μ and standard deviation σ, then, "if N is sufficiently large", sample mean μ_s has approximately a normal distribution with $\mu_s = \mu$ and sample mean standard deviation $\sigma_s = \sigma/\sqrt{N}$. Also expected value of sample totals T_o has approximately a normal distribution with $T_s = N \cdot \mu$ and $\sigma_T = \sqrt{N} \cdot \sigma$. The larger the value the value of N, the better is the approximation.

When a population distribution is normally distributed so are the means of samples of that population for every number N of observations in the sample. Even when the population is highly non-normal, averaging in the form of sample means produces a distribution that tends to the normal, a tendency that increases with N as illustrated by Figure 5.15 (Devore, 1999) below.

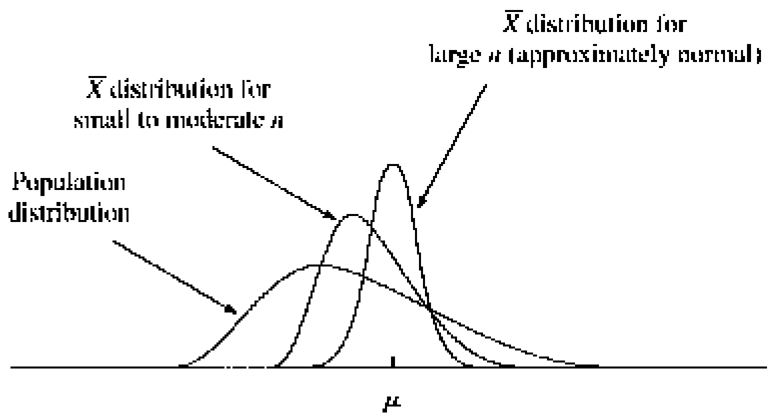


Figure 5.15 The Central Limit Theorem illustrated

The CLT confirms central of tendency of the mean of sample means allowing modelling of the sample mean distribution with a probability density function based on a standard normal distribution. In practice there is a tendency to rely on the CLT to enable prediction of probability of limitations on selected values of a random variable collected as empirical data.

Descriptive statistics calculated from empirical observations can be used as parameters to identify a normal distribution. This normal distribution can, in turn, be modelled by a probability density function that facilitates evaluation of compliance with constraints on the random variable for practical reasons determined by the specific commercial endeavour. In the research random variables examined include bucket loads that consolidate to truck payloads that are total values of small samples. A similar arrangement applies for bucket cycle time and truck loading times. Even if statistics calculated from empirical data do not indicate a symmetrical distribution and the distribution appears to be non-normal, specifically bucket cycle times, the assumption that total values of samples, specifically truck loading times, can be modelled by a normal distribution-based probability distribution function is often sufficiently accurate and generally acceptable.

This allows for simple probability procedures to test for compliance with management controls on payloads such as Caterpillar's 10:10:20 Policy.

MPN 4 Gamma Distribution Parameters

Devore's proposition (Devore, 1999) is used for calculating parameters to develop gamma distributions for qualitative matching to histograms of actual bucket cycle and truck loading times.

Expected value: $E(X) = \mu = \alpha \cdot \beta$ ----------------------------(1)

Variation: $V(X) = \sigma^2 = \alpha \cdot \beta^2$ so $\sigma = \sqrt{\alpha} \cdot \beta$ --(2)

Transposing and using $CV = \sigma / \mu$

$\alpha = 1 / CV^2$ ----------------------------------(3)

$\beta = \mu \cdot CV^2$ ----------------------------------(4)

μ = mean

σ^2 = variance

σ = standard deviation

CV = Coefficient of variation

α =. Gamma distribution scale parameter

β = Gamma distribution shape parameter

Parameter values, calculated using the above equations, and summarized in Table 3.32, were used to develope the gamma distribution frequency curves for qualitative comparison in Figures 3.18, 3.19, 3.24 and 3.25.

MPN 5 "*Mode*" – A Suitable Measure of Central Tendency for Cycle Times

Comparing measures of central tendency at both Filter Level 2 (refer to thesis text for clarification) – Figures 3.16 and 3.17, - and Filter Level 3 (refer to thesis text for clarification) – Figures 3.22 and 3.23, over the 5-pass to 7-pass range indicated that:

- At Filter Level 2, as expected for positively skewed distributions, the general relationship between central tendency measures of average range > mean > median > mode are generally in compliance with the empirical equation:

  Mode = Mean – 3 . (Mean-Median) ……. (Chou, 1969)

  Transposing: Mean – Mode = 3 . (Mean – Median)…………….(3)

  Equation (3) was applied to test consistency of measures of central tendency.

- At Filter Level 3, as expected, the effect of filtering has centralized the residual data but, generally, each measure of central tendency retains its hierarchical position. The small overlap in Figure 3.23, where the mode falls inside the median value for the 6-pass sub-sample of truck loading times, is believed to be due to the effect of hypothetical filtering smoothing data to a bi or multi-modal state.

- Modal values for continuous random distributions are a “simple and useful concept” (Chou, 1969) but of doubtful reliability, but may be positively influenced by:

  - Sample size.
  - Rounding method to whole numbers.
  - Frequency interval. (Chou, 1969)

  Most of these reliability impediments are removed “in whole or part when data is arranged in a frequency distribution” and the mode is determined from “the frequency distribution of the same data” (Chou, 1969). These are the circumstances for treatment of data samples and mode determination herein.

- Intuitively mode should not be significantly affected by filtering. Filter Level 1, Filter Level 2 and Filter Level 3, for the reasons described earlier in this section, all aimed to remove outlying values that will not affect modal values. So constant modal values were expected across filtering levels. Non-compliance with this expectancy triggered investigation and determination of the reasons.

- Table 3.28, summarises all of the measures of central tendency and dispersion at each of the levels of filtering. It is noted that:

  - For bucket cycle times, mode values are generally consistent through the filter levels and only exhibit small variability through the range of all passes and 5, 6 and 7 passes.
  - Mode values for first passes including exchange time, a relatively smaller data set, are consistent through the filter levels for all passes but the sub-samples of 5, 6 and 7 passes indicate a tendency for unreliability.
  - Truck loading times, also a smaller data set, exhibited some anomalous mode values through the range of filter levels.

- Expected values, means, of bucket cycle times, first passes including exchange time and bucket cycle times all exhibit sensitivity to filtering level, i.e., to outlying values that filtering aims to remove. This leads to the conclusion that mean values are not a reliable measure of central tendency for predicting expected values of bucket cycle times and truck loading times. The mean as an applicable measure of central tendency in all cases suffers from its sensitivity to outlying high or low values (Devore, 1999).

- In the case of bucket cycle and truck loading times, comparison of mean and mode provides a measure of the effect of time lost to events or activities that are non-intrinsic to, or symptoms of inefficiency of, loading operations.

- It is noted that, at Filter Level 3, for all loading times, the mode of 199 seconds appears anomalously high. Data for all truck loading times was tabulated as a frequency distribution in one-second and five-second intervals as shown in Table 3.30. At one-second frequency intervals two modes of 7 records manifest at 193 and 199 seconds. Two sub-modes of 6 records manifest at 168 and 178 seconds. This raises the possibility that for larger samples of data with tendency to *statistical regularity (Harr, 1977)* a single mode could manifest at 168, 178, 193, or 199 or perhaps (but unlikely) at some other value. In such circumstances there is no logical basis for selecting one modal value in preference to another. (Chou 1969,). Accumulating to five-second frequency intervals a single mode (21 records) manifests at the 191 –195 second interval. As mode is the most frequent value observed it adopts the designation of the frequency interval – in this case, say, mid value 192.5. It is concluded that a larger sample of data would more likely have promoted 193 seconds as the mode at one-second frequency intervals. This modal value was adopted as an alternative. These observations at ***hypothetical*** Filter Level 3 collectively evidence that modal values can be unstable and so unreliable for relatively small data samples.

- The inconsistent mode value for all truck loading times at Filter Level 3 outlined above, as a result of selection of a single mode in a multi-mode distribution by the Excel spreadsheet statistical MODE function, triggered further comprehensive review of modal values for bucket cycle time and truck loading time similar to the analysis for all truck loading times at Filter Level 3 shown in Table 3.30.

- Intuitively modal values will become less reliable as the data sample numbers decrease with multi-modal distributions resulting. Multiple appearances of observations in the same frequency interval of a continuous random distribution decrease towards 0 as sample size reduces. So modal values of bucket cycle times for all passes or passes less the first appear to be comparatively more reliable than any sub-group of the total data sample such as first passes including truck exchange time and truck loading time particularly when the total data sample is sub-grouped into 5, 6 and 7 passes. For example at Filter Level 3 there is a total of 1,070 bucket cycle times, 188 first pass and truck loading times, 57, 5-pass loads, 96, 6-pass loads and only 23, 7-pass loads.
- Analysis of modal values for truck loading times at Filter level 3 indicated an alternative mode of 193 seconds for all truckloads; and at 7 passes three mode values were identified at 207, 211, and 231 seconds. The computed mode of 207 seconds was retained. Review at 5 and 6 passes confirmed the computed modes for truck loading times.
- Analysis of first passes including exchange time at Filter Level 3, confirmed computed modes at all passes and 6 passes (where three modes at 42, 43 and 44 were identified), but indicated alternatives of 40 seconds at 5 passes and 44 seconds at 7 passes.
- Truck loading times at Filter Level 2 were analysed for multiple modes confirming computed modes for all passes and 5 passes and indicating an alternatives of 193 seconds at 6 passes for the computed 198 seconds. At 7 passes multiple modes of 203, 207,211, 220, 231, 244, 283 and 304 seconds manifested – a result of the small sub-sample number of observations. Widening frequency intervals to 5 seconds and 10 seconds still produced multiple modes. For the purposes of the analysis an alternative of 207 seconds at 7 passes was adopted.

- Analysis of first passes including exchange time at Filter Level 2, confirmed computed modes at all passes and 6 passes, but indicated alternatives of 40 seconds at 5 passes and 44 seconds at 7 passes.
- Generally for examination of multi-modal frequency distributions, frequency interval was increased from one to five, to ten and to twenty seconds as a basis for interpolating mode (Chou, 1969).
- Anomalous modal values in Table 3.28 were replaced with alternative values as shown in Table 3.29. It will be noted that the alternative values generally comply with the hierarchical order of measures of central tendency, i.e., mean > median > mode and correlation with the empirical ratio (mean-median) **:** (mean – mode) = 1 **:** 3.

Review of Tables 3.28 and 3.29 indicates the apparent stability of mode through the filter levels for all bucket cycle times and cycle times nett of the first pass. Modal values are generally minimum values of comparable measures of central tendency in compliance with the hierarchical order for positively skewed distributions of mean > median > mode. There is a notable exception at Filter Level 3 for all loading times where the hypothetical level of filtering has overly modified the data to create a negatively (to left) skewed distribution for all truck loading times – an artificial and impractical distribution outcome.

Figure 3.24, for bucket cycle times, qualitatively illustrates the centralizing effect of Filter Level 3. The histogram of empirical bucket cycle time data compared with normal and gamma distribution models is positively skewed (to the right), albeit modestly. Distribution of truck loading data at Filter Level 3 is qualitatively illustrated by Figure 3.25 - also compared with normal and gamma distribution models. In the case of truck loading times at Filter level 3 the distribution is modestly negatively skewed (to the left)

Figure 3.25 qualitatively indicates that the hypothetical distribution is approximately symmetrical and likely can be modelled by a normal distribution. This is verified by Kolmogorov-Smirnov (K-S) non-parametric testing (see K-S output cross referenced to Figure 3.25:- in Distribution Testing – Kolmogorov–Smirnov Testing in this Volume). This outcome is considered purely hypothetical and considered of little practical importance because of the artificially high level of filtering. It is most unlikely that a distribution of empirical data from the most efficient operation where there is insignificant extraneous time losses in loading operations would be negatively skewed.

MPN 6 Skewness of Distributions of Bucket Cycle & Truck Loading Times

The validity of mode as a measure for bucket cycle time and truck loading time has been discussed in Section 3.2.9 in the thesis text and in MPN 5 above. The natural positive skew of bucket cycle time and loading time distributions has also been discussed. Qualitatively from the several histograms included in Illustrations the positive skew is moderate to subtle for truck loading times and generally more pronounced for bucket cycle times.

Criteria for assessing skewness as high, moderate or mild were derived using Pearson's First Coefficient of Skewness (Chou, 1969).

$$\mathbf{Sk_p = (Mean – Mode) / \sigma} \ \ldots\ldots\ldots\ldots\ldots..(4)$$

$\mathbf{Sk_p}$ = Pearson's First Coefficient of Skewness

As Mode is only an approximation and, in general, is the least stable of the central tendency measures, a median-based coefficient, Pearson's Second Coefficient of Skewness, may be substituted (particularly applicable for moderately skewed distributions).

$$\mathbf{Sk_p = 3 \,.\, (Mean – Median) / \sigma} \ \ldots\ldots\ldots\ldots\ldots..(5)$$

Equation (5) above results from substituting for Mode in equation (4) with:

Mode = Mean – 3(Mean – Median) – see MPN 5 above.

For symmetric, normal, distributions:

Mean = Median = Mode so

$\mathbf{Sk_p}$ = 0

$\mathbf{Sk_p}$ range = +/- 3 theoretically. But practically **$\mathbf{Sk_p}$ range rarely exceeds +/- 1** (Chou, 1969)

It should be noted that skewness coefficients developed in Tables 3.24 and 3.25, and in other tables throughout this thesis are developed from data samples by the Microsoft Excel application. Excel uses a version of skewness measure based on the third moment of each data point from the mean divided by the standard deviation to produce a dimensionless skewness coefficient.

Included in Tables 3.24 are Pearson Second Coefficients of Skewness (PSC) based on Median values that are generally lower than skewness measures from the Excel version.

Based on Chou's practical range limits of **+/- 1 for Sk_p** as described above, the following criteria were adopted for PSC:

High Skewness >0.6

Moderate Skewness =/<0.6>0.3

Mild Skewness =/<0.3

It will be noted that **Sk_p** values at Filter level 2 from Table 3.24:

- For Truck Loading Times:

  All Passes and the 5 Pass sub-sample are indicated in the moderate range; and 6 pass and 7 pass sub-sample are indicated to be towards the lower margin of the high range.

- For Bucket Cycle Times:

  Generally all sub samples of data are in the high range.

At Filter Level 3, **Sk_p** values from Table 3.25:

- For Truck Loading Times:

  All Passes, 5 Pass and 6 Pass sub- samples are in the moderate to mild range and the 7 Pass sub-sample is at the lower margin of the high range.

- For Bucket Cycle Times:

  There is a general tendency for reduction from the high skewness range to the moderate range as passes increase.

In conclusion (refer Tables 3.24 and 3.25):

- Degree of skewness is a measure of distribution asymmetry. A tendency towards distribution symmetry will be accompanied by a reduction in the degree of skewness.

- Following on from the concept of Mode as a performance and control criteria for bucket cycle ant truck loading times, and the discussion on reliability of mode values the concept of skewness as an alternative improvement measure was considered.

- Expected reduction of skewness from Filter Level 2 to Filter Level 3 was positive for truck loading times based on all bucket cycle time data; but for sub-samples at selected pass numbers the tendency for reduction was positive but with negative anomalies.

- Similar results were noted for the Skewness measure used by Microsoft Excel.

- In view of the anomalies and the limited statistical evidence it can only be hypothesized, at this stage, that coefficients of skewness have potential to provided a measure of improvement and control.

- More evidence is required from more tests with larger data samples to firm up and elevate the hypothesis to a practical protocol.

MPN 7 Parameters for Gamma Distributions

As stated in Section 3.2.9, parameters $\boldsymbol{\alpha}$ and $\boldsymbol{\beta}$ **-** derived for modelling gamma distributions for comparison with actual frequency distributions, using the proposition $\boldsymbol{\mu = \alpha\beta}$ and $\boldsymbol{\sigma^2 = \alpha\beta^2}$ (Devore, 1999) - were both relatively high numbers. This is consistent with generation of a distribution model tending away from the asymmetric towards a symmetric, normal distribution; particularly as the level of filtering increased. This tendency has already been discussed in empirical terms above.

Table 3.32 summarises the $\boldsymbol{\alpha}$ and $\boldsymbol{\beta}$ values derived and used for gamma distribution models applied in relevant histograms shown in Figures 3.18, 3.19, 3.24 and 3.25.

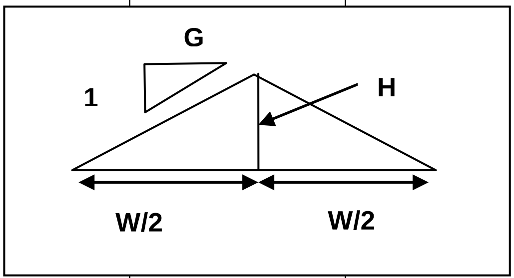
MPN 8 Payload Distribution and Individual Wheel Loads

Refer to: Section 3.3.7 Truck Payload Centre of Gravity (CG) and Related Issues.

Part A - Solution of Payload Centre of Gravity Location

PART A - DATA				
FOR: Caterpillar 793C		Design Distribution %		
		Total	Front	Rear
Gross Machine Weight	Tonnes	383.7	33.33%	66.67%
Nett Machine Weight (Tare)	Tonnes	160.8	43.50%	56.50%
Target Payload - By Difference	Tonnes	222.9	To be determined	
Truck Wheelbase	Metres	5.9		
Proportion of Payload Superimposed	%	30.00%		
PART A - SOLUTION				
		Distribution		
Item	Unit	Front Wheels	Rear Wheels	Total
Gross Machine Weight	Tonnes	127.887	255.813	383.700
Nett Machine Weight	Tonnes	69.948	90.852	160.800
Payload By Difference		57.939	164.961	222.900
Payload Distribution		0.260	0.740	1.000
Payload Centre of Gravity Relative to Rear Wheel Centre	Metres	1.534		

Part B Proportion of Payload with Variable Location

PART B - DATA & CALCULATIONS				
FOR: Caterpillar 793C	**Units**	**Struck**	**Heaped 2:1**	**Ratio**
SAE Rated Capacity				
Flat Floor	**Cubic Metres**	110	147.6	1.342
Dual Slope	**Cubic Metres**	96	129	1.344
Cross-sectional Sketch Of Superimposed Proportion of Payload				
Slope Gradient	**G:1**	**2:1**	**1.5:1**	
FOR: Caterpillar 793C				
W	**Metres**	7.2	7.2	
H	**Metres**	1.8	2.4	
Superimposed Volume:				
Pyramidal	**Index**	1.000	1.333	
Conical	**Index**	0.785	1.047	
Relative to Struck Volume x 1.34:				
Pyramidal	**Index**	1.342	1.789	
Conical	**Index**	1.054	1.405	
Superimposed Volume Proportion:		=(1.405 - 1)/1.405		
Conical 1.5:1	**Index**		**28.82%**	
Round Up & Allow for Side Rail Freeboard:			**30.00%**	
		Distribution		
		Front Wheels	**Rear Wheels**	**Total**
Payload Distribution:		0.260	0.740	
Fixed	**70%**	**40.557**	**115.473**	**156.030**
Relocatable (Superimposed)	**30%**	**17.382**	**49.488**	**66.870**
Total				**222.900**

PART C - RELATIONSHIP BETWEEN LOCATION OF SUPERIMPOSED PAYLOAD AND WHEEL LOAD ING DISTRIBUTION

Part 1 of 2 Parts of Table C		Distribution		
		Total	Front Wheels	Rear Wheels
Nett Machine Weight (Tare)			43.50%	56.50%
	Tonnes	160.8	***69.948***	***90.852***
Fixed Payload			25.99%	74.01%
	Tonnes	156.030	***40.557***	***115.473***
Relocatable Payload				
Move forward -	**Metres**	**1.00**		
Distance From Centre of Gravity	**Metres**		3.366	2.534
Distribution			42.94%	57.06%
	Tonnes	**66.870**	***28.716***	***38.154***
		383.700	***139.221***	***244.479***
			36.28%	**63.72%**
			33.33%	66.67%
			2.954%	-2.954%
GMW Per Wheel - Design	**%**		**16.67%**	**16.67%**
GMW Per Wheel - Payload Forward 1 m	**%**		**18.14%**	**15.93%**
Wheel Load Change	**%**		**8.86%**	**-4.43%**
Move forward -	**Metres**	**0.50**		
Distance From Centre of Gravity	**Metres**		3.866	2.034
Distribution			34.47%	65.53%
	Tonnes	**66.870**	***23.049***	***43.821***
		383.700	***133.554***	***250.146***
			34.81%	**65.19%**
			33.33%	66.67%
			1.477%	-1.477%
GMW Per Wheel - Design	**%**		**16.67%**	**16.67%**
GMW Per Wheel - Payload Forward 0.5 m	**%**		**17.40%**	**16.30%**
Wheel Load Change	**%**		**4.43%**	**-2.22%**

Part 2 of 2 Parts of Table C		Distribution		
		Total	**Front Wheels**	**Rear Wheels**
Move forward -	**Metres**	**1.50**		
Distance From Centre of Gravity	**Metres**		2.866	3.034
Distribution			51.42%	48.58%
	Tonnes	**66.870**	***34.383***	***32.487***
		383.700	***144.888***	***238.812***
			37.76%	**62.24%**
			33.33%	66.67%
			4.431%	-4.431%
GMW Per Wheel - Design	**%**		**16.67%**	**16.67%**
GMW Per Wheel - Payload Forward 1.5 m	**%**		**18.88%**	**15.56%**
Wheel Load Change	**%**		**13.29%**	**-6.65%**
Move forward -	**Metres**	**2.00**		
Distance From Centre of Gravity	**Metres**		2.366	3.534
Distribution			59.89%	40.11%
	Tonnes	**66.870**	***40.050***	***26.820***
		383.700	***150.555***	***233.145***
			39.24%	**60.76%**
			33.33%	66.67%
			5.908%	-5.908%
GMW Per Wheel - Design	**%**		**16.67%**	**16.67%**
GMW Per Wheel - Payload Forward 2 m	**%**		**19.62%**	**15.19%**
Wheel Load Change	**%**		**17.72%**	**-8.86%**
Move rearward -	**Metres**	**-0.50**		
Distance From Centre of Gravity	**Metres**		4.866	1.034
Distribution			17.52%	82.48%
	Tonnes	**66.870**	***11.715***	***55.155***
		383.700	***122.220***	***261.480***
			31.85%	**68.15%**
			33.33%	66.67%
			-1.477%	1.477%
GMW Per Wheel - Design	**%**		**16.67%**	**16.67%**
GMW Per Wheel - Payload Rearward 0.5 m	**%**		**15.93%**	**17.04%**
Wheel Load Change	**%**		**-4.43%**	**2.22%**

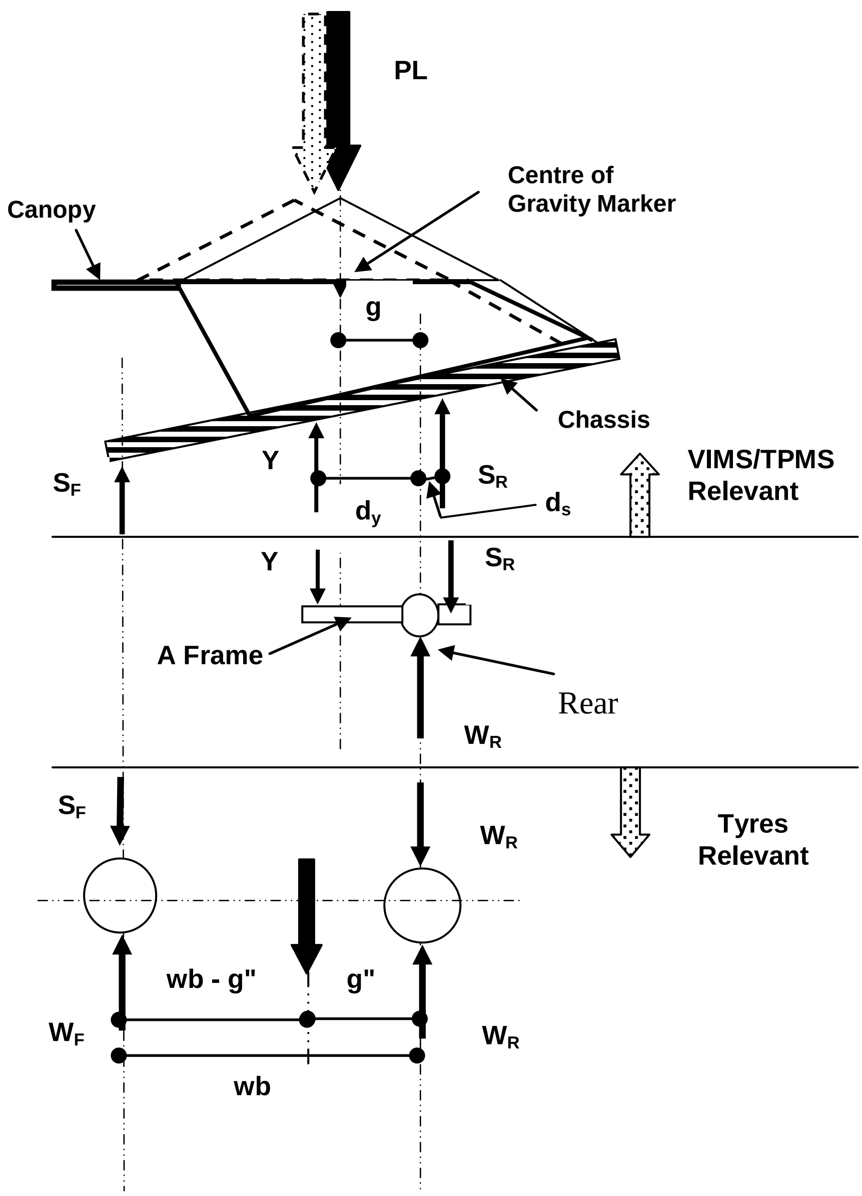
PART D - RELATIONSHIPS BETWEEN PAYLOAD TRANFER SYSTEMS

Assuming Wheel Centres In Horizontal Plane		
Unless Otherwise Stated All Load Symbols, e.g., $W_{R,}$ Are Payload Increments Only		
Basic Statics Equations:		
Overall	$W_F + W_R = PL$	**(1)**
Across Rear Wheel Group	$Y + S_R = W_R$	**(2)**
Front Wheel Load	$S_F = W_F$	**(3)**
Chassis Support System	$S_F + Y + S_R = PL$	**(4)**
Calculations:		
Moments About Y	$S_R(d_y+d_s)/d_y = W_R$	**(5)**
Moments About S_R	$Y(d_y+d_s)/d_s = W_R$	**(6)**
Combining (5) & (6)	$Y = S_R.\ d_s/d_y$	**(7)**
Substituting in (4)	$S_F + S_R(1 + d_s/d_y) = PL$	**(8)**
Ratios:		
Payload:- Ride Struts	$S_F/S_R = (g + d_s)/\{wb - (g + d_s)\}$	**(9)**
Rear Wheel Group	$Y/S_R = d_s/d_y$	**(10)**
Wheel Loads - Payload Increment	$W_F/W_R = g/(wb - g)$	**(11)**
Wheel Loads - NMW	$W'_F/W'_R = g'/(wb - g')$	**(11A)**
Wheel Loads - GMW	$W''_F/W''_R = g''/(wb - g'')$	**(11B)**
Wheel Load Distribution Ratio - Payload	$g/(wb - g) = 0.260/0.740$	**(12)**
Wheel Load Distribution Ratio - NMW	$g'/(wb - g') = 0.435/0.565$	**(12A)**
Wheel Load Distribution Ratio - GMW	$g''/(wb - g'') = 0.333/0.667$	**(12B)**

Figure 3.3.7.D

PAYLOAD TRANFER SYSTEMS

Refer to Part D Above



PART E - REAR WHEELS ON TOE OF FACE

Assuming Wheel Centres Remain Practically In Horizontal Plane

W_R is reactive load for an individual pair of rear wheels

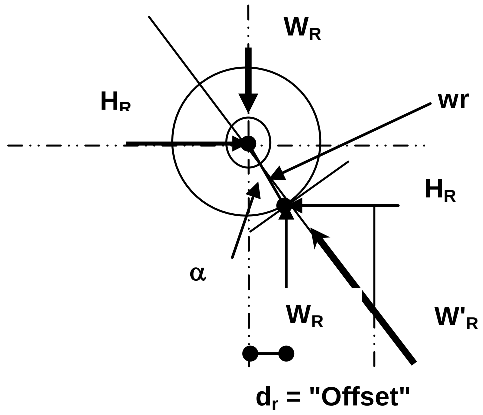
Analysis Generally Applies for Payload Increment, NMW or GMW

Static Forces

Tyre Load: $W'_R = W_R / \cos\alpha$ (13)

Tyre Load Offset: $dr = wr \cdot \sin\alpha$ (14)

It will be noted that vertical axle load W_R is not affected by the offsetting of tyre load. So for Payload Increment, NMW or GMW the vertical rear wheel load W_R will be constant. As the offset increases, The horizontal component that is a result of the braking effort will increase as does W'_R the resultant increased tyre load. Component H_R is accommodated by A frame front trunnion, 4-bar linkage or equivalent componentry.



Incipient Dynamic Forces Resulting From Brake Torque

Braking Torque $T_B = W_R \cdot d_r$

Resultant Reactive on A Frame Trunnion:

$B_T = W_R \cdot d_r/d_y$ (15)

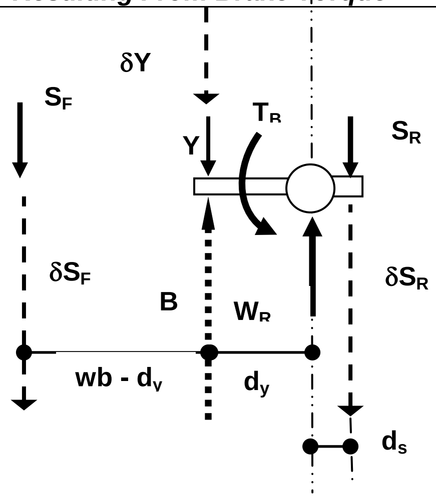
$\delta Y = B_T$ (16)

$\delta Y = \delta S_F + \delta S_R$ (17)

$\delta S_F(wb - d_y) = \delta S_R(d_y + d_s)$ (18)

$\delta SF = \delta Y[1 + (wb - d_y)/(d_y + d_s)$ (19)

To retain the truck in position brake torque must be applied to react to the moment of wheel load W_R offset by d_r. Reactive B_T is applied at the trunnion effectively acting as an additional load on the chassis shown as δY. The suspension struts react to this virtual load increment and experience incremental down thrusts δS_F and δS_R apportioned in inverse ratio to horizontal distance from the trunnion. Equations 14 to 17 above express the load relationships. By taking moments & substituting/transposing Equations 18 and 19 facilitate a solution for the two resultant incremental suspension loads δS_F and δS_R from This is a simplified analysis - no allowance has been made for geometrical reorientation of load transfer systems. See text for further discussion.



PART F LOAD DISTIBUTION ON RAMPS

Context of Analysis Is Uphill Loaded Hauling.
Other Hauling Configurations Require Separate But Similar Analysis
Considering Tyre Load Distribution In Terms of GMW Only

Basic Equations

Horizontal Road:

$W_F + W_R = GMW$ (20)

WF/WR = g/(wb-g) (21)

= 0.333/0.667 = 0.5 for design GMW

g/wb = 0.333 for design GMW

h = Vertical Distance of Centre of Gravity of GMW above wheel centres

Inclined Road:

$W'_F + W'_R = GMW$ (22)

$W'_F/W'_R = g'/(wb'-g')$ (23)

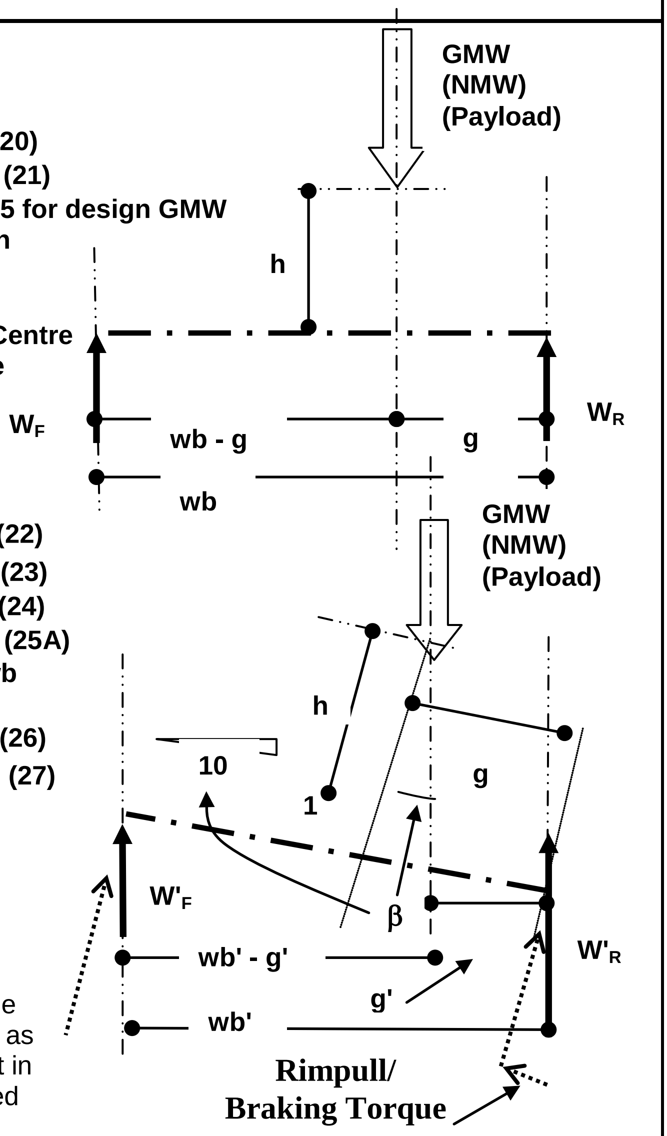
$g' = g - h \, . \, \sin\beta$ (24)

$g' = 0.333wb - h \, . \, \sin\beta$ (25A)

$g'/wb = 0.333 – h \, . \, \sin\beta/wb$ (25B)

$wb' = wb \, . \, \cos\beta$ (26)

$\beta = \mathrm{Tan}^{-1}$ Road Gradient (27)



This is a simplified, non-exhaustive analysis. No consideration has been given to the effects of drive and brake torque reactive forces that will affect the loads sensed by the suspension struts as such affect is only of academic interest in this research. See text for more detailed discussion.

PART G – GRADIENT-FORCE RELATIONSHIPS

There are several relationships relating grade of the ramp/road and forces experienced by mining trucks: Referring to the diagrams and equations in Part F above the following relationships are relevant here:

Grade = Rise/Horizontal Distance (1:10 in Part F above) (28)

Grade = Tan β ≈ Sin β for small β (29)

Rimpull = GMW . Sin β ≈ GMW . Tan β ≈ GMW . Grade (30)

For a practical maximum "effective grade" ('effective grade" interpreted below) of, say, 18% (15% +3% rolling resistance), necessarily of limited distance:

Equation (30) above: **Rimpull ≈ 0.18 . GMW**

Above is overstated by: **Tan β /Sin β = 0.18/0.1771 = 1.016**

That is +**1.6%** - conservative - error.

It should be noted that rimpull is sensitive to "Effective Grade" (including rolling resistance) **–** sometimes termed "Total Resistance" (Caterpillar PHB 35, 2004)

Ramps and roads with lesser grades will have reduced errors. For practical purposes, estimating rimpull by applying grade to GMW is reasonable.

It should be noted that ride struts rotate in space by the same angle as the grade of the ramp or road. Accordingly the ride struts will measure a component of the vertical loads normally carried, viz.,

Strut load on ramp = normal strut load. Cos β (31)

For 15% Grade: **Ride Strut Loads = Normal loads . 0.989**

That is an effective reduction of 1.1%.

Load measuring using ride strut pressures is not relevant on ramps so the above has little, if any, practical utility.

"Effective Grade≡Total Resistance": In addition to the resistance or propulsive effects of road surface grade there are other elements of resistance that can conveniently be combined with geometrical grade as "effective grade". There is an inherent resistance to rolling of pneumatic tyres on a perfectly flat non-yielding surface that depends on the internal tyre structure and tyre pressure. Tyre rolling resistance for mining trucks is in the order of 1.0% to 2.5 % of GMW depending on the number of wheels and tyre construction. With fabric-reinforced, bias-ply tyres practically history, having been essentially replaced by steel-reinforced radial-ply tyres, the current practical range is generally 1.0% to 1.5% for conventional six-wheel mining trucks (Caterpillar PHB 35. 2004).

MPN 9. Bucket Passes and Payloads

Refer to Section 3.3.9 Bucket Passes and Payloads - *Productivity and Sacrificing Passes.*

Assuming there is always a truck waiting to be loaded, bucket loads in excess of planned passes effectively delays all subsequent trucks by a cycle time for each additional pass.

On each occasion, sacrificing a bucket pass or passes saves time as follows:

$$T_{TS} = T_B \bullet n_S \bullet N_T \qquad (1)$$

T_{TS} = Trucking time saved – seconds

T_B = Bucket cycle time - seconds

n_S = Number of passes sacrificed, equivalent to number of occasions

N_T = Number of trucks affected

Extra payload proportion represented by time saved per occasion:

$$PL_X = T_B \bullet n_S \bullet N_T / T_T \qquad (1A)$$

PL_X = Extra payload(s)

T_T = Truck trip time – seconds.

Payload proportion lost by sacrificing a bucket pass:

$$PL_L = 1 \bullet F_F / M_P \qquad \text{for one pass sacrifice} \qquad (2)$$

$$PL_L = n_S \bullet F_F / M_P \qquad \text{generally} \qquad (2A)$$

PL_L = Payload proportion lost by sacrificing a bucket pass.

F_F = Fill discount factor for the sacrificed bucket pass or mean over passes sacrificed.

M_P = Mean number of passes for target payload

In terms of productivity, the test for sacrifice of a bucket pass is:

$$PL_X \geq PL_L \qquad (3)$$

The above equations 1A, 2A and relationship 3 are the basis for Table 3.57. The above analysis and Table 3.57 are limited to a single pass sacrifice. Multiple pass sacrifice is a relatively minor complication. Equations (1) and (1A) can be applied generally for sacrificed passes. Equation (2) is modified to (2A) to be generally applicable and F_F is then the mean bucket fill over the passes sacrificed. The spreadsheet containing Table 3.57 can accommodate multiple passes by an optional selection for number of passes. This can be verified by review of the copy of the file on the CD inside the back cover of the thesis.

It should be noted that the above analysis is in terms of productivity alone. Relevant cost implications have been considered and analyzed in Section 5.4.3, to this stage of analysis.

DISTRIBUTION TESTING

KOLMOGOROV – SMIRNOV NON-PARAMETRIC TESTING

P-P PLOTS

Q-Q PLOTS

Includes Table of Contents

TABLE OF CONTENTS - DISTRIBUTION TESTING

KOLMOGOROV – SMIRNOV (K-S) NON-PARAMETRIC DISTRIBUTION TESTING

KS 1 INTRODUCTION

Historical and current production and cost records; also current industry-wide experience are a source of data to be used as a basis, generally for inference and specifically, for pre-selection analysis during all levels of feasibility from preliminary to final. The value of historical and current data records continues through procurement investigation and due diligence activity to arrive at a decision to issue orders or execute supply contracts for initiating or continuing commercial mining operations.

These notes are provided as support for the general investigative process used in the course of the research and described by this thesis. Some inferences drawn and hypotheses on behaviour of some critical productivity and cost criteria based on statistical analysis are validated by observation of events, i.e. empirical data, from actual mining operations.

KS 2 DISTRIBUTIONS AND TESTS

Events recorded as empirical data from open-pit mining activities are most often continuous (e.g., bucket loads and truck payloads; bucket cycles and truck loading time, truck travel time including variable dump time) or discrete (e.g., bucket passes per truck load, truck loads per unit of time). The noted sources of data are random variables.

In the broadest sense investigation and analysis, especially in the process of establishing the services required from a load and haul fleet for an open-pit mining operation, can be viewed generically as a simulation process. “A simulation is the imitation of the operation of a real-world process or system over time. Whether done by hand or on a computer, simulation involves the generation of an artificial history of a system, and the observation of that artificial history to draw inferences concerning the operating characteristics of the real system.”(Banks J, 1996)

The behaviour of a system is studied:

- By developing a simulation model – usually in the form of assumption(s) regarding operation of the system.

- Assumption(s) are expressed in mathematical, logical, symbolic relationships between the objects of interest.
- Once developed and validated a model can be utilised to examine any "what-if" queries about the real-world system.
- Simulation models can be used both for analysis of existing systems and for design, prediction of performance of new systems. (Banks J, 1996, p3)

Utilisation of simulation models varies from the simplest application of mathematical processes to the highly complex where mathematical analysis is difficult if not possible. Techniques applied range from:

- Traditional simple methods of hand-worked estimating using deterministic methods through;
- Linear programming, multiple regression and complex; but well-developed, relatively complex, mathematical methods to;
- Imitation of the behaviour of the system using numerical, computer-based simulation.

Recent common usage of the term "simulation" has tended to confine it to the more complex numerical computer-facilitated analytical processes. But, in the broadest sense, all estimating processes infer adoption of a model that, when analysed, is predictable and provides repeatable outcomes. But all of these processes are estimating and any model chosen to represent a real system is an estimate of the form and characteristics of the real system.

All analysis from the simplest deterministic to the complex simulation requires input of relevant operating criteria and general understanding of the interrelationship between component functions that make up the system. Investigation of some of the applicable criteria and understanding of interrelationships was a primary objective of the research.

As described herein the approach was to examine data from actual operations, whether from continuous or discrete variables, to estimate a range of descriptive statistics including:

- Maximum and minimum values - from which the range and mean of the range were derived – as first indicative measures of dispersion and symmetry.
- ***Arithmetic Mean*** **–** as the principle measure of central tendency.
- Median and Mode – as secondary measures of central tendency; and with the mean as joint indicators of symmetry.
- Variance – produced but not utilised.
- ***Standard Deviation*** **–** as an important measure of dispersion.
- ***Coefficient of Variation*** derived from Standard Deviation / Mean –an important dimensionless measure of dispersion allowing comparison of actual data dispersion with other similar operating activities.
- ***Skewness and Kurtosis*** - as measures of central tendency and distribution "shape".

Descriptive statistics of principal interest in the research are shown in bold.

The listed descriptive statistics provided impressions of symmetry, degree of dispersion and skewness inferring, in many cases, a suitable model distribution. For example:

- Tendency to symmetry, low dispersion and low skewness inferred a Normal Distribution – particularly for bucket loads and truck payloads.
- Long positive tails, i.e., high positively skewness and high dispersion with mean>median>mode in the ratio of 1:2 inferred a Gamma Distribution – particularly for bucket cycle times and truck loading times especially where empirical data is unfiltered.

When continuously variable data from a real system is available, the process of selecting a model distribution to approximate the real system consists:

- Organize data in a frequency distribution with equal-width frequency intervals selected .as described in Section 3.2.8 in the thesis.
- Construct a histogram from frequency distribution data iteratively adjusting frequency intervals to smooth up the shape.

- From the histogram shape infer a comparative test distribution and overlay a continuous test distribution curve over the histogram.

The purpose of preparing a histogram is to infer a known probability distribution function (pdf) of probability mass function (pmf) (Banks J, 1996)

This qualitative identification process is severally illustrated in cases identified in List of Figures by any of Figures 3.2 through 9 and Figure 3.where ? Indicates any integer 2 to 9, .

Valuable references for background to the following discussion to follow and for definition of terms include Chou Y, 1969, p45 for explanation of terms, “relative distribution”, “relative frequency distribution” and “absolute frequency distribution”; p59 & 60 for Probability as the limiting value of Relative Frequency; p123 “relative frequency is the objective or empirical definition of probability”; p160 & 161 for basics including random variables, events and probability function; and Devore J, 1999, p60 “Limiting Relative Frequency ≡ Probability”.

KS 3 KOLMOGOROV – SMIRNOV NON-PARAMETRIC TESTS

Preamble

Where empirical data distributions were highly skewed, and for the general avoidance of doubt, applicability of selected test distributions was verified with the non-parametric Kolmogorov – Smirnov (K - S) test.

Reasons for selection of the K - S test were listed in MPN 2 in Mathematical Principle – Notes.

Essentially the K - S test is a “goodness-of-fit” test that enables repeatable testing of distributions of differing event scale. As adopted in SPSS the K – S test measures the degree of agreement between the distribution of a sample of empirically generated data and a theoretical distribution. Data from the empirical distribution is presented as a cumulative distribution function (cdf) of relative frequencies and compared with a cdf of relative frequencies (essentially a probability density function) for the test distribution. The test distribution is chosen intuitively or qualitatively on the basis of descriptive statistics and frequency histograms for the empirical data.

K-S Test Process

The process generally consists:

- Reducing empirical data to relative frequency of each individual observation and standardizing so that the sum of all empirical frequencies = 1.
- Arranging the empirical frequency observations in ascending order and compiling a cdf.
- Overlaying the cdf of the test distribution in similar relative frequency form.
- Calculate differences between the frequency distributions and determining the maximum absolute difference (maximum value ignoring sign), i.e., $|D|$.
- Multiply the maximum absolute frequency difference, $|D|$, by$\sqrt{N}$, where N is the number of empirical observations, to determine K - S "Z" - the test statistic. (Multiplying by $\sqrt{N}$ adjusts the sample measure of variation σ_S to the population value $\sigma = 1$ in the standardized form of the test distribution).
- At the calculated Z_C value a P-value = α_C, is compared with a critical value that corresponds to the selected significance (of "goodness-of-fit") level, α – generally one of 0.1, 0.05 and 0.01 – the SPSS K - S application is based on a critical α_C compared with a selected significance level of 0.05.
- α is the "rejection area" – containing all of the K – S Z values where the null hypothesis H_0 is rejected.

The K – S test adopts the null hypothesis - H_0 - that there is no significant difference between the empirical sample distribution and the test distribution.

Discussion and Interpretation

If a K - S test yields a Z_C statistic < critical Z_α and a corresponding Asymptotic Significance (AS – discussed below) ≥ 0.05, i.e., the critical value of α - the specified level of significance - then H_0 will be accepted. Alternatively a Z_C statistic > critical Z_α and a corresponding AS < 0.05, then H_0 will be rejected.

α = P-value; the area in the upper tail of a pdf and is the error probability for a Type I Error, i.e., that the null hypothesis is rejected when it is true. For more detail on "Tests of Hypotheses" and error "Types" refer to Devore J, 1999 pp310 to 314.

Figure 8.7, 1, Upper-tailed Test, illustrates the relationship between test statistic Z and rejection area α.

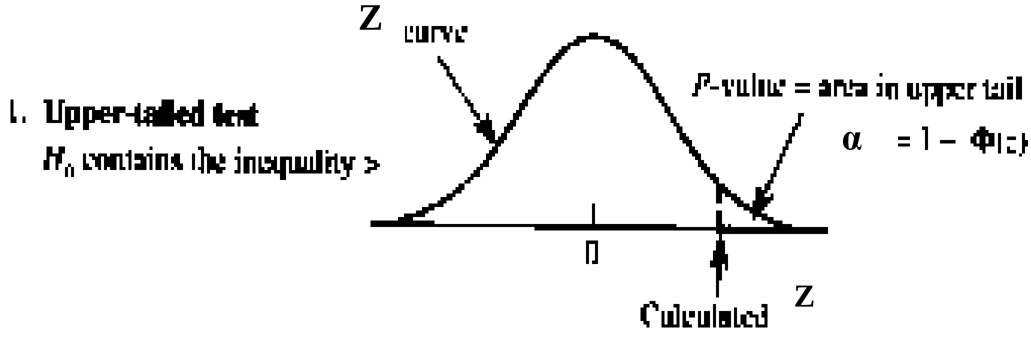


Figure 8.7, Upper-tailed Test

(Devore J, 1999, p343)

In Figure 8.7:

- Calculated Z = Z_C
- P-value = α = 1 – ΦZ
- ΦZ is the cumulative area corresponding to calculated statistic Z_α for the test distribution.
- α is as defined.

For more details refer to Jay Devore, *Probability and Statistics for Engineering and the Sciences*, 5th Edition, 1999, Duxbury, pp310 to 344, (Devore J, 1999) describing *Tests of Hypotheses Based On a Single Sample.* Particularly pp342 to 344 describes *The P-value for a Z Test* – adopted for the K – S test.

The K – S test as applied by the SPSS analytical software used in the research calculates values of the Z statistic and corresponding calculated values of the P-value α_C to be compared with the selected significance level of α = 0.05. K – S Z statistics only provide a means of discrete non-modulated acceptance or rejection of "goodness of fit". By reporting AS results the K – S test leaves it to the individual observer to compare α_C with α and so determine that the null hypothesis cannot ($\alpha_C > \alpha$); or should ($\alpha_C \leq \alpha$) be rejected. In the K – S test output α_C values enable judgement by the individual to accept the null hypothesis where $\alpha_C \leq 0.05$. Appropriate circumstances for such discretion is where the sample number N is large. Reporting AS results allows application of discretion in terms of “practical significance” assessment to temper or override “statistical significance” when N is large (Devore J, 1999, p348). This discussion item is of academic interest only in the context of the research as the results of K – S tests were generally robustly in favour of acceptance of the null hypothesis or, alternatively, to reject it.

The K – S test as applied by SPSS reports calculated values of the Z statistic and the corresponding α value termed Asymptotic Significance (AS) in the report table. Comparing the calculated α_C value with the selected significance value of 0.05 facilitates the test.

Critical Z values drawn on by SPSS appear to correspond to the values in Table A8 from Banks J, Carson JS, and Nelson BL, *Discrete Event Simulation* 2nd Edit, 1996, Prentice-Hall, p539 except that the K – S Z values are extended by the √N multiplier as described. A copy of Table A8 is shown herein.

Table KS 3.1 includes extended K – S critical Z values for the three levels of selected reject area, α, significance in Table A8. Figure KS 3.1 illustrates the trend towards limiting values as critical Z values approach limits of:

AS Critical Value α	**Limiting Critical Z Value (N > 35)**
0.10	1.22
0.05	**1.36 adopted for K – S test**
0.01	1.63

The trend lines at the three levels of significance are generally smooth from N = 10 to 25. For N > 25 the more erratic Z values are believed to be due to rounding up critical values in Banks’ Table A8 to the second decimal place. For a practical range of sample numbers critical Z values can be approximated as constants:

N Range		**Critical AS 0.10**	**0.05**	**0.01**
10 to 20		1.17	**1.30**	1.56
20 to 35		1.20	**1.34**	1.60
>35	As above	1.22	**1.36**	1.63

Small differences between the cumulative relative frequency distribution of a sample of N observations and the test distribution, in like configuration, report as small K – S Z values that in turn correspond to calculated α_C values higher than selected significance value of $\alpha = 0.05$ indicating that H_0 cannot be rejected, i.e., should be accepted. This validates the selected test distribution family.

In the K – S results table produced by SPSS a value of Asymptotic Significance (AS) is shown. This is the value of reject area α corresponding to the calculated Z for the test. So:

Calculated AS $\geq$ Selected significance value 0.05

Corresponds to:

Calculated $Z_C <$ Critical Z_α

In both cases the null hypothesis H_0 cannot be rejected. Obviously, for AS $< 0.05 \rightarrow 0$ the rejection area is diminishing. Continues under illustration on page K – S 10

Table A.8 Kolmogorov-Smirnov Critical Values

Degrees of Freedom (N)	$D_{0.10}$	$D_{0.05}$	$D_{0.01}$
1	0.950	0.975	0.995
2	0.776	0.842	0.929
3	0.642	0.708	0.828
4	0.564	0.624	0.733
5	0.510	0.565	0.669
6	0.470	0.521	0.618
7	0.438	0.486	0.577
8	0.411	0.457	0.543
9	0.388	0.432	0.514
10	0.368	0.410	0.490
11	0.352	0.391	0.468
12	0.338	0.375	0.450
13	0.325	0.361	0.433
14	0.314	0.349	0.418
15	0.304	0.338	0.404
16	0.295	0.328	0.392
17	0.286	0.318	0.381
18	0.278	0.309	0.371
19	0.272	0.301	0.363
20	0.264	0.294	0.356
25	0.24	0.27	0.32
30	0.22	0.24	0.29
35	0.21	0.23	0.27
Over 35	$\frac{1.22}{\sqrt{N}}$	$\frac{1.36}{\sqrt{N}}$	$\frac{1.63}{\sqrt{N}}$

Source: F. J. Massey, "The Kolmogorov-Smirnov Test for Goodness of Fit," *The Journal of the American Statistical Association*, Vol. 46, © 1951, p. 70. Adapted with permission of the American Statistical Association.

(Banks J, 1999, p539)

Continues from bottom of page K – S 9

Selecting an AS critical value higher than .05, say 0.10, will increase the reject area and will be a more stringent test of "goodness of fit". The likelihood of Type I errors, i.e. rejecting H_0 when it is true, will increase. Conversely selecting a AS critical value lower than 0.05, say 0.01, will reduce the discrimination of the test and tend to increase Type II errors, i.e. not rejecting (acceptance) of H_0 when it is false.

K – S Test Outcomes

A sample of K – S tests performed during analysis of bucket loads and truck payloads in Section 3.2.8 and bucket cycle times and truck loading times in Section 3.2.9 follow this discussion. Each K – S test is cross-referenced to the corresponding Table and Figure in Sections 3.2.8, 3.2.9 or other appropriate sections of the thesis text where K - S tests have been applied.

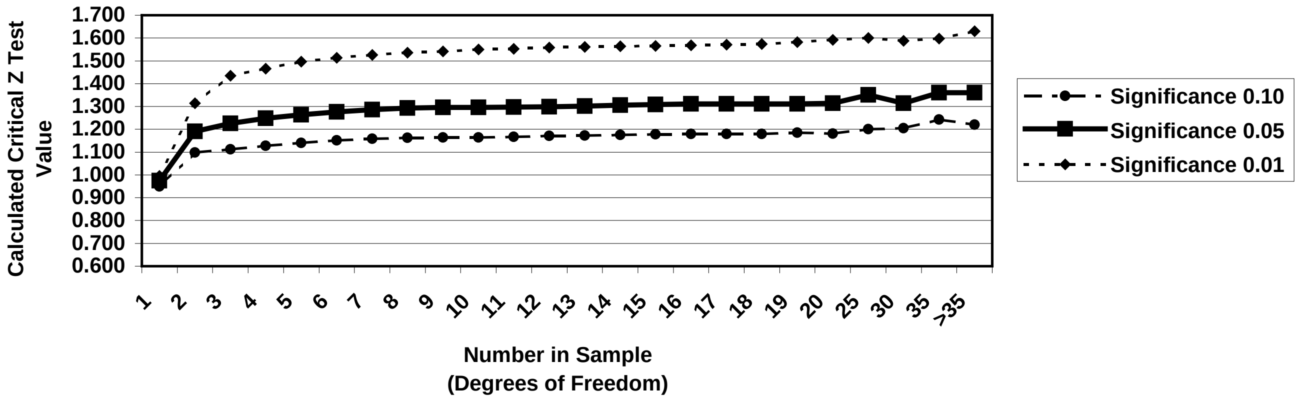
The results of the sample of K – S tests have been summarized in Table KS 6.1.

Table KS 3.1 - Kolmogorov - Smirnov Test
Critical Values of Z Statistic v. Number In Sample

Based On, and Modified from Table A8, p 539, Banks J, Carson JS, Nelson BL *Discrete-Event System Simulation,* 2nd Edit, 1996

Degrees of Freedom N (Sample Size)	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	25	30	35	>35
Specified Significance																								
0.10	0.950	1.097	1.112	1.128	1.140	1.151	1.159	1.162	1.164	1.164	1.167	1.171	1.172	1.175	1.177	1.180	1.179	1.179	1.186	1.181	1.200	1.205	1.242	**1.220**
0.05	0.975	1.191	1.226	1.248	1.263	1.276	1.286	1.293	1.296	1.297	1.297	1.299	1.302	1.306	1.309	1.312	1.311	1.311	1.312	1.315	1.350	1.315	1.361	**1.360**
0.01	0.995	1.314	1.434	1.466	1.496	1.514	1.527	1.536	1.542	1.550	1.552	1.559	1.561	1.564	1.565	1.568	1.571	1.574	1.582	1.592	1.600	1.588	1.597	**1.630**

Figure KS 3.1 - Kolmogorov - Smirnov Test - Critical Values of Z Statistic v. Number in Sample
Modified from Table A8 - Banks J, 1996, p539



KS 4 PROBABILITY PLOTS

An effective and convenient method of testing an assumed family of distributions against a distribution of empirical sample data is by comparing probability plots (Devore J, 1999, p186).

The SPSS application package provides, in addition to the K – S test procedure, a facility for comparative Probability Plots – termed P-P Plots in SPSS but simplified to Pplots herein- where pairs of relative frequency values, empirical sample and selected test distribution, are compared graphically.

Pplots produced by SPSS in conjunction with the K – S tests are constructed generally as follows:

- Empirical sample data is sorted in order from the smallest to highest values.
- Arrange the data in cumulative frequency intervals that practically consist a cumulative probability distribution – the "observed cumulative probability".
- Plot the "expected cumulative probability" for the test distribution based on similar cumulative frequency intervals against the "observed cumulative probability".
- If the Pplot generally falls on a straight line with gradient 1 (45^o) then the assumed test distribution is plausible.
- Deviations from the 45^o straight line, at extreme values, notwithstanding good correlation for central values, indicate that the test distribution is not an acceptable choice.

Convenient availability in the SPSS application of Pplots promoted inclusion with each K – S test as additional evidence of acceptance or non-acceptance of the test distribution. It should be noted that a decision based on a Pplot whether to accept or reject a hypothesized distribution model is subjective.

KS 5 QUANTILE-QUANTILE PLOTS

Histograms from empirical data and their qualitative fit to a continuous density function – a test distribution – may be difficult to compare. This is particularly so for a small number of events, N; and if the frequency intervals adopted for the frequency distribution of empirical data to draft the histogram are not optimum. Histograms of a small number of events in an even smaller number of frequency intervals are often ragged and difficult to compare.

Banks (Banks J, 1996, p365) provides details of the underlying theory for Quantile-Quantile Plots (Q-Q Plots as produced by SPSS and simplified to Qplots herein). For the purposes of application of Qplots as a supplement to K – S tests it is considered sufficient to understand that:

- An empirical cumulative distribution function can be compared with a member of a family of distributions – the test distribution – in cdf form.
- A Quantile (Q) is a statistic that can conveniently be viewed as a special frequency value.
- A cumulative Q plot of the empirical data that will be approximately a straight line can be compared with a Q plot of the test distribution.
- A Q plot of the test distribution will also be a straight line.
- Comparing the two Q plots will yield an approximately straight line with a gradient of 1 (45^o).
- Deviation, usually in a systematic manner, from the 45^o straight line indicates the assumed test distribution is inappropriate.

As with Pplots, convenient availability in the SPSS application of Qplots encouraged and facilitated inclusion with each K – S test as additional evidence of acceptance or non-acceptance of the test distribution.

It should be noted that Qplots are based on a similar comparison of empirical data with a distribution model as the K – S test. But the Qplot has some similarities with Pplots. So in some respects the inclusion of Qplots in the context of the research could be considered academic. Notwithstanding this reservation Qplots have been reported for completeness and for readers who favour the Qplot as a test procedure.

It should be noted that, as with Pplots a decision based on Qplots whether to accept or reject a hypothesized distribution model is also subjective.

KS 6 SUMMARY OF K – S TEST RESULTS, PPLOTS AND QPLOTS AND INTERPRETATION

Summary of Results

Table KS 6.1, summarises the results of K – S Tests, Pplots and Qplots with comments on individual test comparisons.

Interpretation

Bucket Loads and Truck Payloads

It is clearly shown by the results in Table KS 6.1 and in the K - S test results following this section - Figures 3.2 to 3.4 and 3.6 to 3.9 - that bucket loads and truck payloads are generally normally distributed. Truck payload control criteria (such as Caterpillar's 10:10:20 rule) in terms of statistics can be drawn from the normal probability model. This is discussed in some detail in Section 3.2.8 in the thesis.

Bucket Cycle Time and Truck Loading Time

Modelling of bucket cycle times and truck loading times is differing problem as illustrated by the results in Table KS 6.1 for Figures 3.2.9.1, 3.2.9.7, 3.2.9.8, 3.2.9.13, 3.2.9.14. As discussed in 3.2.9 in the thesis, bucket cycle times and truck loading times, as accumulated bucket cycle times, have intrinsic and non-intrinsic time components. Intrinsic bucket cycle and truck loading times appear to have central tendency and likely can be modelled by a normal distribution, albeit positively skewed. Normal probabilities could likely be applied to intrinsic times. But non-intrinsic time in bucket cycles and accumulated in truck loading times result in positively skewed empirical distributions that can likely modelled by a gamma distribution including the less positively skewed Weibull distribution.

In the process of productivity planning and equipment selection it may be a more transparent practice to develop bucket cycle times and accumulated truck loading times in terms of intrinsic time and deal with the non-intrinsic bucket cycle and truck loading time separately as discussed in 3.2.9.in the thesis.

Bucket Passes Per Truck Payload

As shown by the results in Table Ks 6.1 for Figures 3.5 and 3.12, modelling of bucket passes per truckload is a further separate complication.

Hypothetically, the range of distribution models that can be emulated by a random empirical sample of passes per truckload can, in practice, vary from a discrete single constant value (by operational protocol) to a quasi symmetrical normal distribution for a large number (N) of passes.

The results in Table KS 6.1 for Figures 3.5 and 3.12 indicate the range of possibilities, at least to some degree.

The K – S test for Figure 3.5 – for a range of passes 4 to 6, a limited range of 3 values – provides no lead to an acceptable distribution model. Pplots for Normal, (encouraging, but doubtful acceptance) through Lognormal, Gamma to Weibull (reasonable acceptance as a model) indicates improving "goodness of fit".

The K – S test for Figure 3.12 - for a range of passes 3 to 9, an extended range of 7 values – did not support acceptance of the normal model. Pplots indicated a Normal distribution as reasonably acceptable with Gamma improving and Weibull as best "goodness of fit". But the indication by Pplot that the Normal model is "reasonably acceptable" indicates the trend to central tendency of the increased range of number of passes.

The implication of passes per truckload for truck payload dispersion has been discussed at some length in Section 3.2.8 in the thesis.

In terms of truck loading time the implications of numbers of passes is discussed further and in more detail in Sections 3.2.11 and 4.1.4.

Table KS 6.1 – Summary & Comments – Kolmogorov – Smirnov Test, Pplots & Qplots

Distribution Type Legend: BL = Bucket Loads TPL = Truck Payloads BP/TL = Bucket Passes/Truck Payload

BCT = Bucket Cycle Times TLT = Truck Loading Time

Poisson or other terms in italics = Discrete Distribution

Reference Figure	Type of Distrib.	Num. Records	K - S Results		K – S Test Distribution & Comments	Pplot Comments	Qplot Comments
			AS/α	**Z**			
3.2	**BL**	**35**	0.915	0.557	Normal: Robust acceptance	Confirms K-S. Some non-systematic, in consequential departures	Confirms Pplot
3.2A	**BL**	**43**	0.925	0.548	Normal: Robust acceptance	Ditto above	Confirms Pplot
3.3	**BL**	**237**	0.441	0.866	Normal: Cannot be rejected	Confirms K-S. Some departures but acceptable	Confirms Pplot
3.4	**TPL**	**73**	0.815	0.635	Normal: Robust acceptance	Ditto PP Plot for Fig. 3.2	Confirms Pplot
3.5	**BP/TL**	**73**	0.000 0.000 0.000	3.115 2.974 5.055	Normal: Cannot accept *Poisson:* Cannot accept Exponential: Cannot accept	Normal PP - Contrary to KS - Possibly acceptable model Lognormal – Improved fit c.f. Normal PP Gamma – further improved fit Weibull - Best fit - acceptable model (Small sample numbers makes P and Q plots ragged)	Confirms Pplot Confirms Pplot (not shown) Ditto Lognormal Confirms Pplot
3.6	**BL**	**59**	0.442	0.865	Normal: Cannot be rejected	Confirms K-S. Some departures but acceptable. Positive (right) skew evident.	Confirms Pplot
3.7	**BL**	**272**	0.403	0.892	Normal: Cannot be rejected	Confirms K-S. Filtering-induced small negative (left) skew shown clearly	Confirms Pplot

Reference Figure	Type of Distrib.	Num. Records	K - S Results		K – S Test Distribution & Comments	Pplot Comments	Qplot Comments
			AS/α	**Z**			
3.8	**BL**	**140**	0.908	0.564	Normal: Robust acceptance	Confirms K-S. Close fit to normal model. Reflects low range/dispersion of selected, 4-pass only, data	Generally confirms Pplot. Some minor raggedness in upper values confirms Histogram Fig 3.8
3.9	**BL**	**115**	0.458	0.855	Normal: Cannot be rejected	Confirms K-S. Reasonable fit. Some general raggedness confirms Histogram 3.9	Confirms Pplot
3.12	**BP/TL**	**428**	0.000	5.231	Normal: Cannot accept	Contrary to KS - Reasonably acceptable model. Refer back to Section 3.2.8.5. Increased range of passes reports as increased central tendency Gamma – good fit Weibull – closest fit Positive skew more subtle than for Fig 3.5	Confirms Pplot – hint of positive skew Not shown Confirms Weibull Pplot
3.18	**BCT**	**1725**	0.000 0.000	6.382 5.395	Normal: Cannot accept *Poisson*: Cannot accept	Normal - confirms KS – indicates positive (right) skew Gamma - improved fit and indicates heavy positive tail	Confirms Pplot Confirms Pplot

Reference Figure	Type of Distrib.	Num. Records	K - S Results		K – S Test Distribution & Comments	Pplot Comments	Qplot Comments
			AS/α	**Z**			
3.19	**TLT**	**368**	0.023	1.496	Normal: Cannot accept	Normal confirms KS – indicates substantial positive skew c.f. Figure 3.18 Gamma – indicated best fit Weibull – indicates positive skew and long positive (right) tail	Confirms Pplot Confirms Pplot Weibull Qplot not shown
3.24	**BCT**	**882**	0.000 0.000	4.159 2.729	Normal: Cannot accept *Poisson:* Cannot accept	Normal confirms KS – as expected positive skew c.f. Figure 3.18 Gamma – indicates good fit	Confirms Pplot Confirms Pplot
3.25	**TLT**	**188**	0.423	0.878	Normal: Cannot be rejected	Confirms K-S. Filtering-induced small negative (left) skew shown clearly	Confirms Pplot

CROSS-REFERENCE LIST OF FIGURE NUMBERS

The following Figures illustrate application of K-S Testing, P-P and Q-Q Plots. This part of the research used a different numbering system to that finally adopted for the thesis. The K-S and related testing was executed using SPSS analytical software.

All test results and plots are cross-referenced

The following table - List of "Figure Numbers" cross-references the numbering systems.

List of Figure Numbers

SPSS Figure Number	Thesis Figure Number	Thesis Table Number
3.2.8.1	3.2	3.7
3.2.8.2.	3.2A	3.10
3.2.8.3	3.3	3.12
3.2.8.4	3.4	3.12
3.2.8.5	3.5	3.12
3.2.8.6	3.6	3.14
3.2.8.7	3.7	3.15
3.2.8.8	3.8	3.16
3.2.8.9	3.9	3.17
3.2.9.1	3.12	3.20
3.2.9.7	3.18	3.24
3.2.9.8	3.19	3.24
3.2.9.13	3.24	3.25
3.2.9.14	3.25	3.25
Table 3.3.9.1		3.53

FIGURE 3.2 (3.2.8.1) AND TABLE 3.7

K - S NPar Test Results

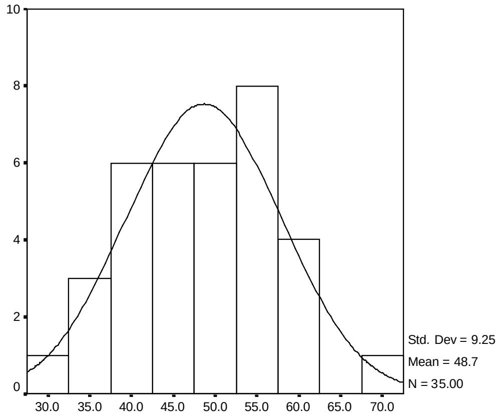
One-Sample Kolmogorov-Smirnov Test

		Figure 3.2.8.1
N		35
Normal Parameters[a,b]	Mean	48.6543
	Std. Deviation	9.24692
Most Extreme Differences	Absolute	.094
	Positive	.070
	Negative	-.094
Kolmogorov-Smirnov Z		.557
Asymp. Sig. (2-tailed)		.915

a. Test distribution is Normal.

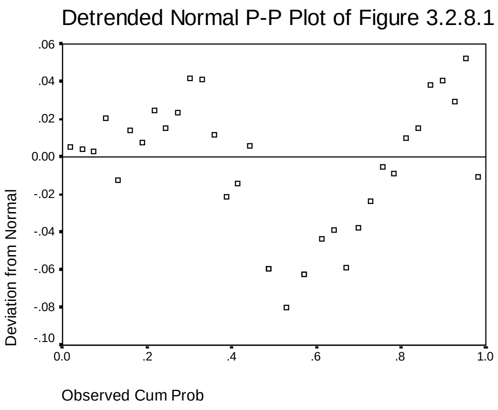
b. Calculated from data.

Histogram



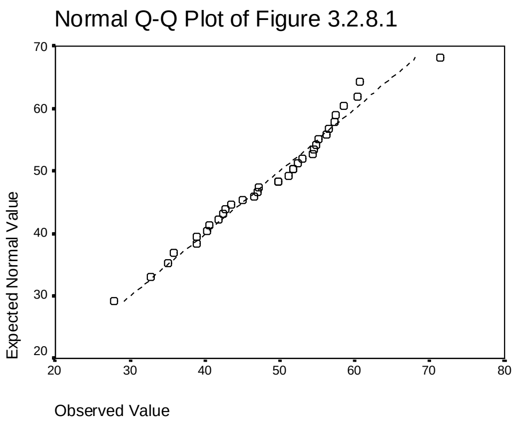
Pplot

```
MODEL:  MOD_2.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.2 ...
Normal distribution parameters estimated: location = 48.654286 and
scale = 9.2469182
```



Pplot

```
MODEL:  MOD_1.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.2 ...
Normal distribution parameters estimated: location = 48.654286 and
scale = 9.2469182
```



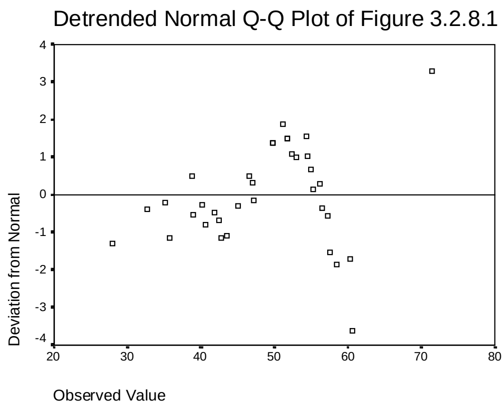


FIGURE 3.2A (3.2.8.2) AND TABLE 3.10

K - S NPar Test Results

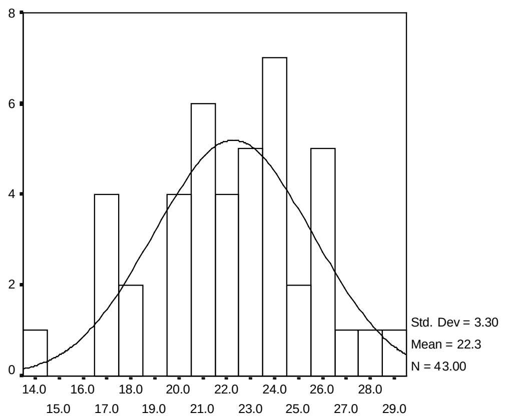
One-Sample Kolmogorov-Smirnov Test

		Figure 3.2.8.2
N		43
Normal Parameters[a,b]	Mean	22.2651
	Std. Deviation	3.29552
Most Extreme Differences	Absolute	.084
	Positive	.057
	Negative	-.084
Kolmogorov-Smirnov Z		.548
Asymp. Sig. (2-tailed)		.925

a. Test distribution is Normal.

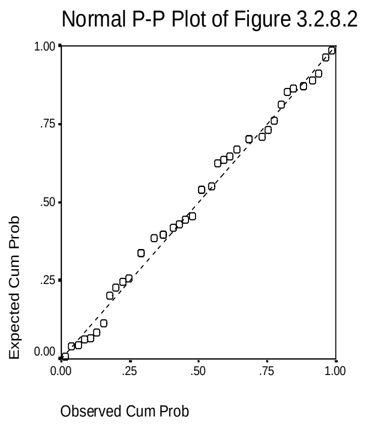
b. Calculated from data.

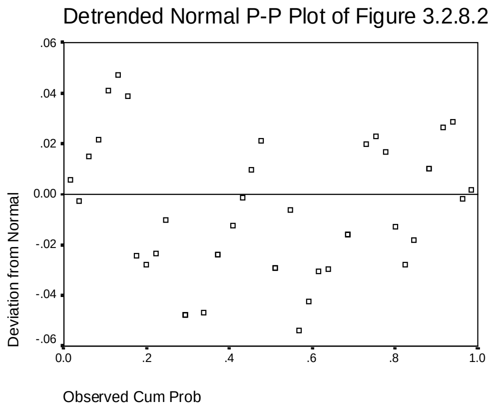
Histogram

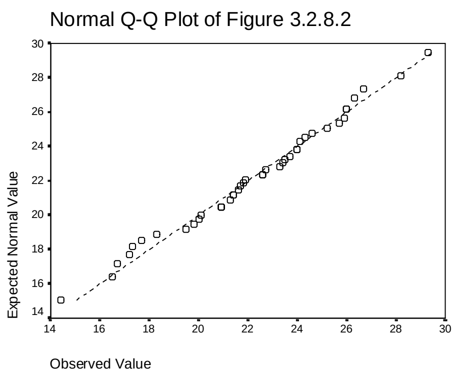


Pplot

```
MODEL:  MOD_1.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.2A ...
Normal distribution parameters estimated: location = 22.265116 and
scale = 3.2955153
```


Normal P-P Plot of Figure 3.2.8.2
Expected Cum Prob
Observed Cum Prob
0.00
.25
.50
.75
1.00


Detrended Normal P-P Plot of Figure 3.2.8.2
Deviation from Normal
Observed Cum Prob
.06
.04
.02
0.00
-.02
-.04
-.06
0.0
.2
.4
.6
.8
1.0


Normal Q-Q Plot of Figure 3.2.8.2
Expected Normal Value
Observed Value
14
16
18
20
22
24
26
28
30

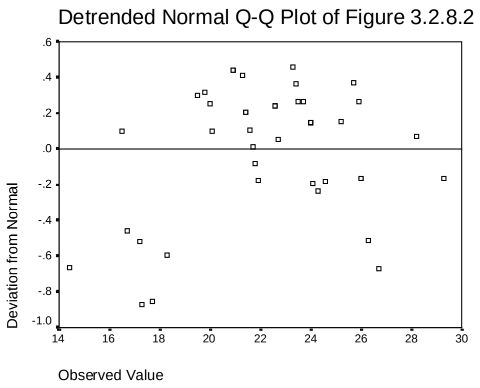

Detrended Normal Q-Q Plot of Figure 3.2.8.2
Deviation from Normal
Observed Value
.6
.4
.2
.0
-.2
-.4
-.6
-.8
-1.0
14
16
18
20
22
24
26
28
30

FIGURE 3.3 (3.2.8.3) AND TABLE 3.12

K - S NPar Test Results

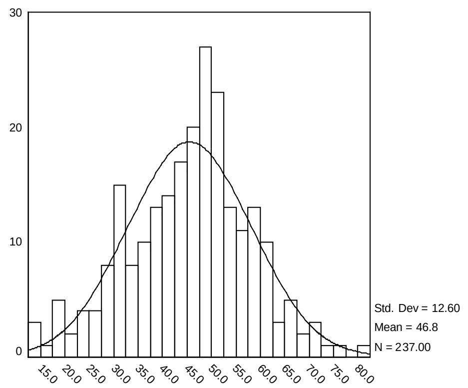
One-Sample Kolmogorov-Smirnov Test

		Figure 3.2.8.3
N		237
Normal Parameters[a,b]	Mean	46.7949
	Std. Deviation	12.60374
Most Extreme Differences	Absolute	.056
	Positive	.031
	Negative	-.056
Kolmogorov-Smirnov Z		.866
Asymp. Sig. (2-tailed)		.441

a. Test distribution is Normal.

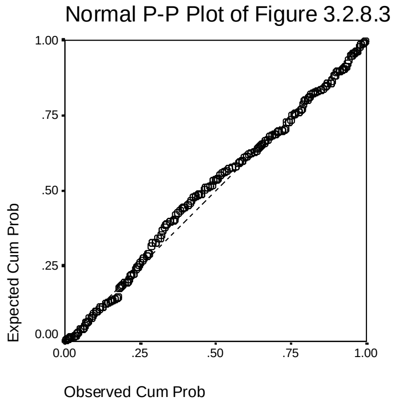
b. Calculated from data.

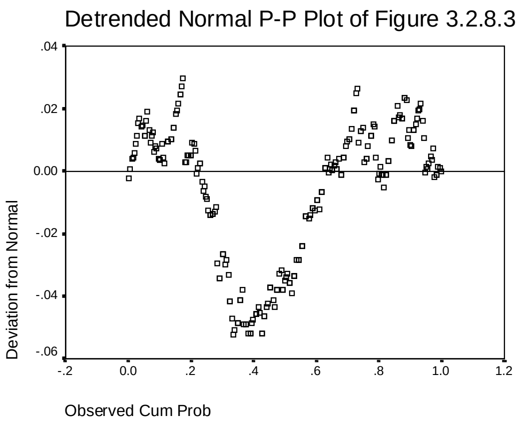
Histogram



Pplot

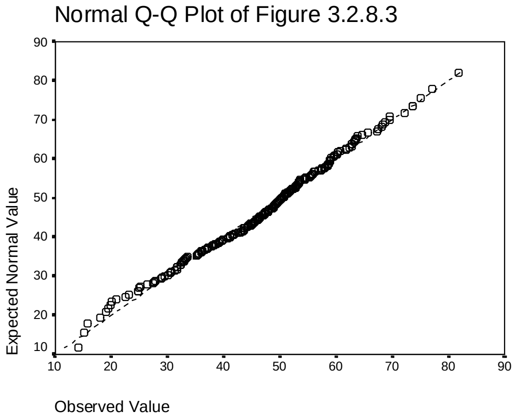
```
MODEL:  MOD_1.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.3 ...
Normal distribution parameters estimated: location = 46.794937 and
scale = 12.603741
```





Qplot

```
MODEL:  MOD_1.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.3 ...
Normal distribution parameters estimated: location = 46.794937 and
scale = 12.603741
```



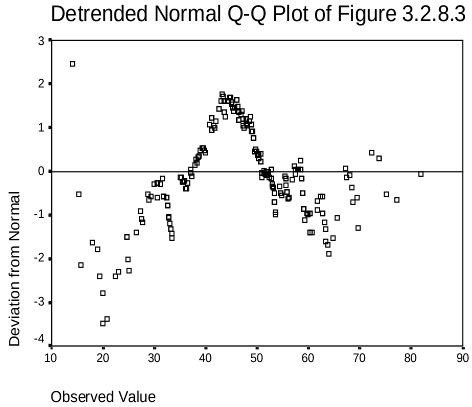


FIGURE 3.4 (3.2.8.4) AND TABLE 3.12

K - S NPar Test Results

One-Sample Kolmogorov-Smirnov Test

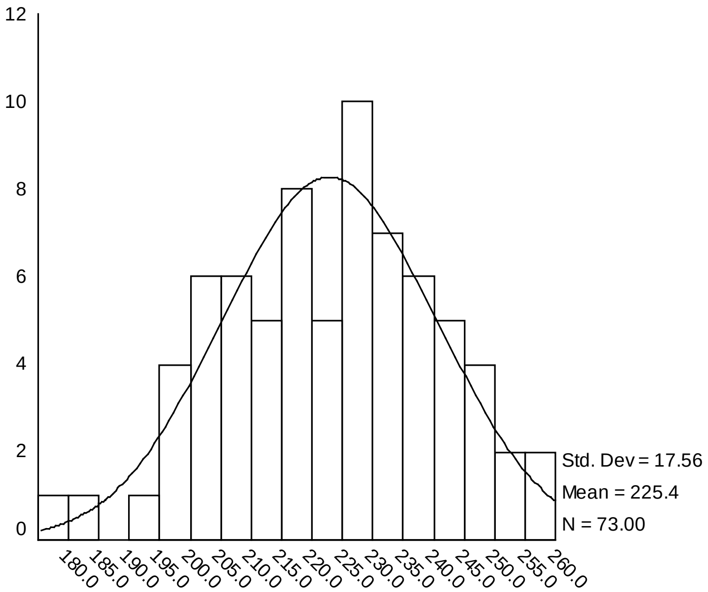
One-Sample Kolmogorov-Smirnov Test

		Figure 3.2.8.4
N		73
Normal Parameters[a,b]	Mean	225.3781
	Std. Deviation	17.56181
Most Extreme Differences	Absolute	.074
	Positive	.074
	Negative	-.061
Kolmogorov-Smirnov Z		.635
Asymp. Sig. (2-tailed)		.815

a. Test distribution is Normal.

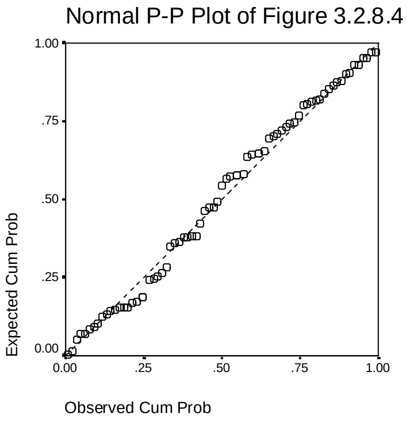
b. Calculated from data.

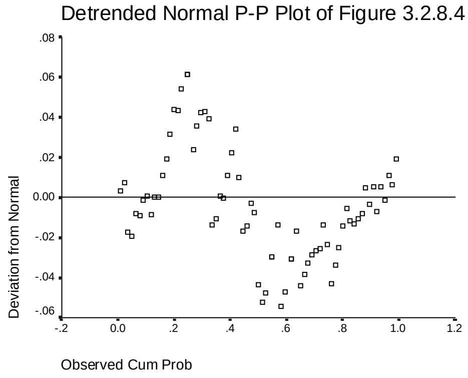
Histogram



Pplot

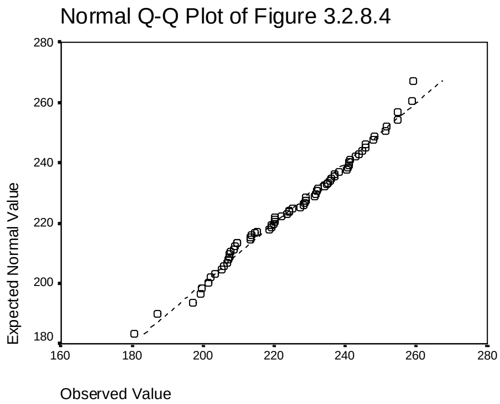
```
MODEL:  MOD_2.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.4 ...
Normal distribution parameters estimated: location = 225.37808 and
scale = 17.561813
```





Qplot

```
MODEL:  MOD_3.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.4 ...
Normal distribution parameters estimated: location = 225.37808 and
scale = 17.561813
```



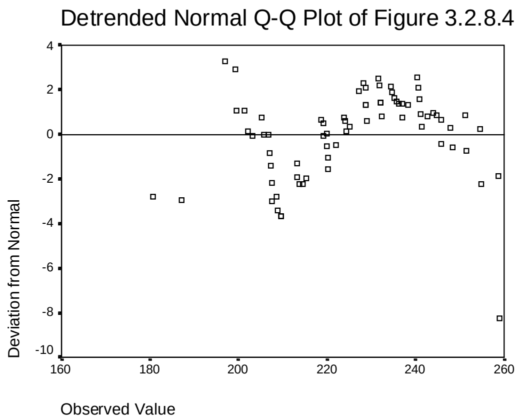


FIGURE 3.5 (3.2.8.5) AND TABLE 3.12

K - S NPar Test Results

One-Sample Kolmogorov-Smirnov Test

		Figure 3.2.8.5
N		73
Normal Parameters[a,b]	Mean	4.4658
	Std. Deviation	.57932
Most Extreme Differences	Absolute	.365
	Positive	.365
	Negative	-.246
Kolmogorov-Smirnov Z		3.115
Asymp. Sig. (2-tailed)		.000

a. Test distribution is Normal.

b. Calculated from data.

One-Sample Kolmogorov-Smirnov Test 2

		Figure 3.2.8.5
N		73
Poisson Parameter[a,b]	Mean	4.4658
Most Extreme Differences	Absolute	.348
	Positive	.250
	Negative	-.348
Kolmogorov-Smirnov Z		2.974
Asymp. Sig. (2-tailed)		.000

a. Test distribution is Poisson.

b. Calculated from data.

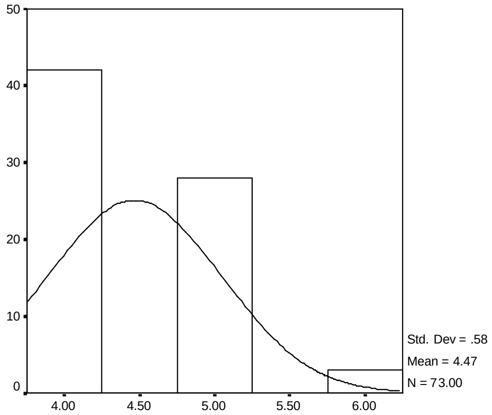
One-Sample Kolmogorov-Smirnov Test 3

		Figure 3.2.8.5
N		73
Exponential parameter.[a,b]	Mean	4.4658
Most Extreme Differences	Absolute	.592
	Positive	.285
	Negative	-.592
Kolmogorov-Smirnov Z		5.055
Asymp. Sig. (2-tailed)		.000

a. Test Distribution is Exponential.

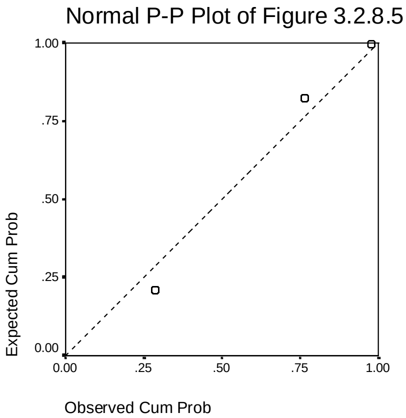
b. Calculated from data.

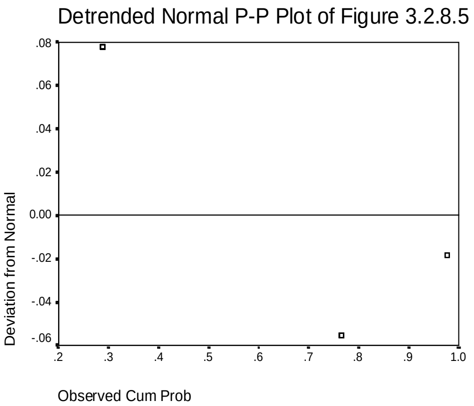
Histogram



Pplot

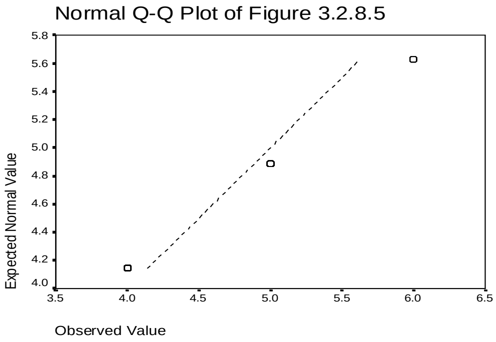
MODEL: MOD_3.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.5 ...
Normal distribution parameters estimated: location = 4.4657534 and scale = .57932412

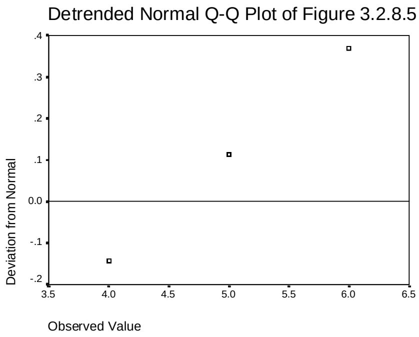




Qplot

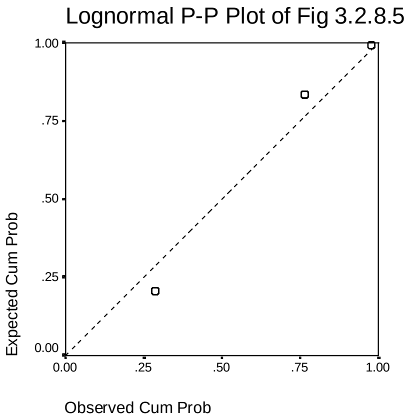
MODEL: MOD_1.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Table 3.2.8.5 ...
Normal distribution parameters estimated: location = 4.4657534 and scale = .57932412

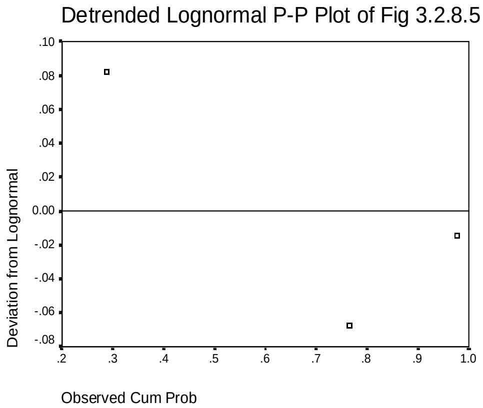




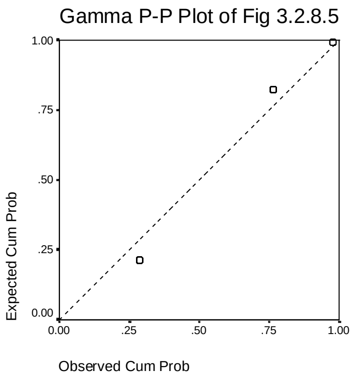
Additional Pplots & Qplots

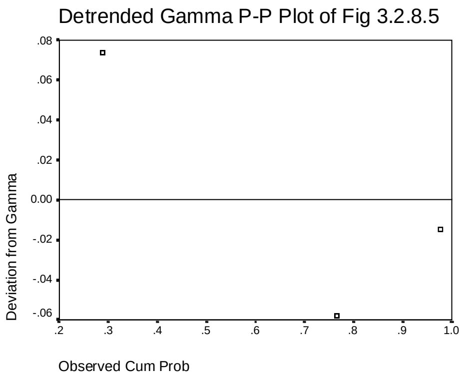
```
MODEL:  MOD_1.
Distribution tested: Lognormal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.5 ...
Lognormal distribution parameters estimated: scale = 4.4306514 and
shape = .12495321
```





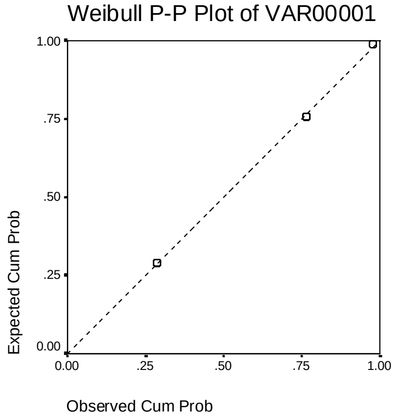

```
MODEL:  MOD_3.
Distribution tested: Gamma
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.5 ...
Gamma distribution parameters estimated: shape = 59.421862 and scale
= 13.306122
```

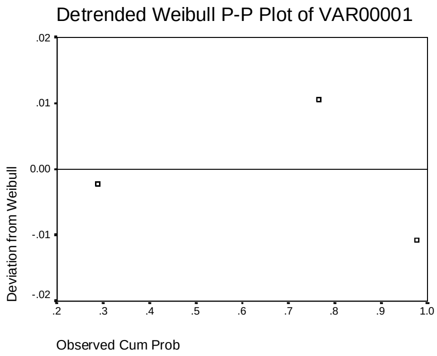





```
MODEL:  MOD_4.
Distribution tested: Weibull
```

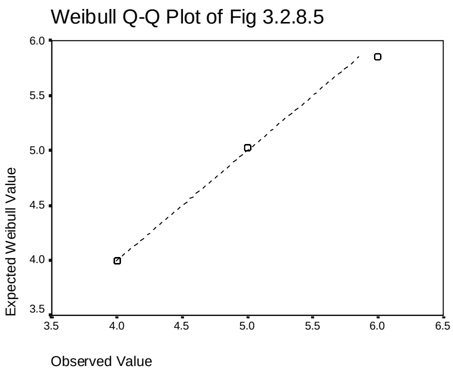
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Fig 3.5 ...
Weibull distribution parameters estimated: scale = 4.7362239 and shape = 6.3264985





Qplot

MODEL: MOD_8.
Distribution tested: Weibull
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.5 ...
Weibull distribution parameters estimated: scale = 4.7362239 and shape = 6.3264985



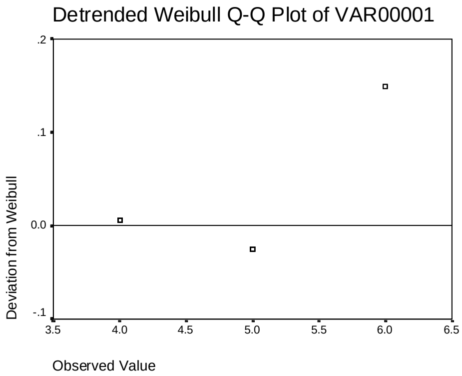


FIGURE 3.6 (3.2.8.6) AND TABLE 3.14

K - S NPar Test Results

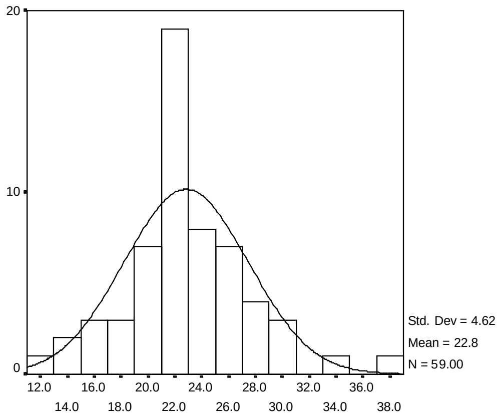
One-Sample Kolmogorov-Smirnov Test

		Figure 3.2.8.6
N		59
Normal Parameters[a,b]	Mean	22.8390
	Std. Deviation	4.61728
Most Extreme Differences	Absolute	.113
	Positive	.113
	Negative	-.090
Kolmogorov-Smirnov Z		.865
Asymp. Sig. (2-tailed)		.442

a. Test distribution is Normal.

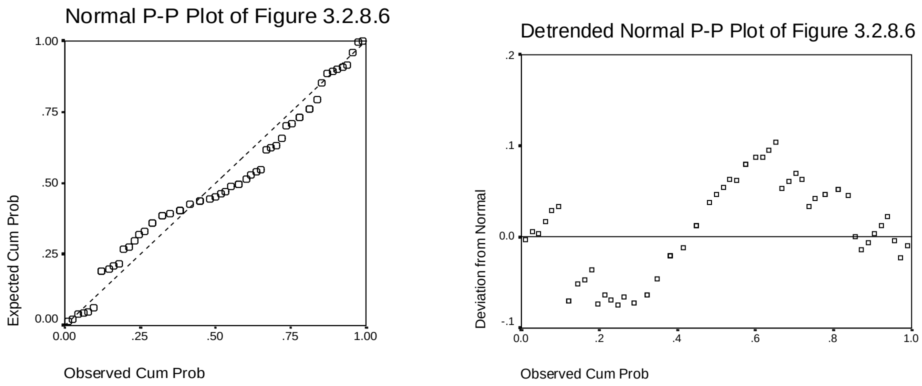
b. Calculated from data.

Histogram



Pplot

```
MODEL:  MOD_1.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.6 ...
Normal distribution parameters estimated: location = 22.838983 and
scale = 4.6172845
```



Pplot

```
MODEL:  MOD_2.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.2.8.6 ...
Normal distribution parameters estimated: location = 22.838983 and
scale = 4.6172845
```

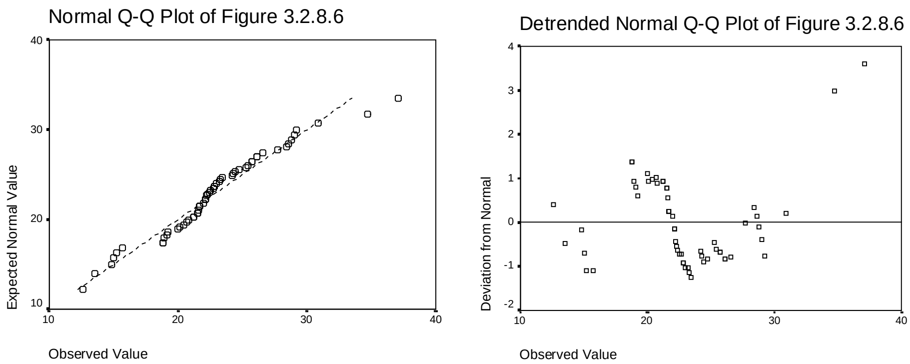


FIGURE 3.7 (3.2.8.7) AND TABLE 3.15

K - S NPar Test Results

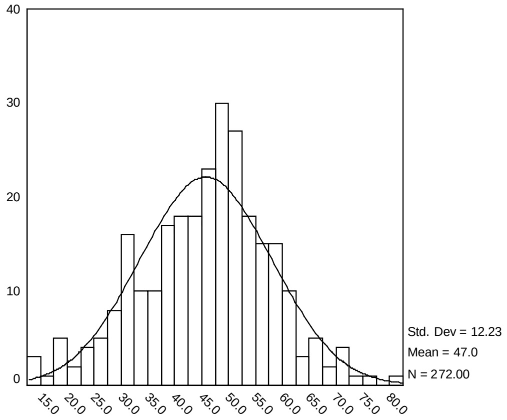
One-Sample Kolmogorov-Smirnov Test

		Figure 3.2.8.7
N		272
Normal Parameters[a,b]	Mean	47.0342
	Std. Deviation	12.22517
Most Extreme Differences	Absolute	.054
	Positive	.033
	Negative	-.054
Kolmogorov-Smirnov Z		.892
Asymp. Sig. (2-tailed)		.403

a. Test distribution is Normal.

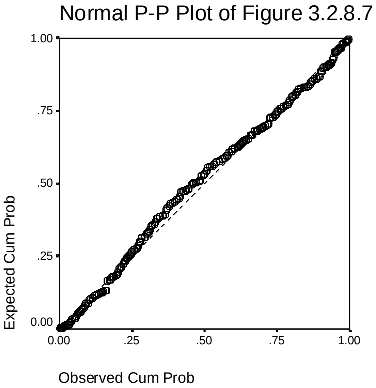
b. Calculated from data.

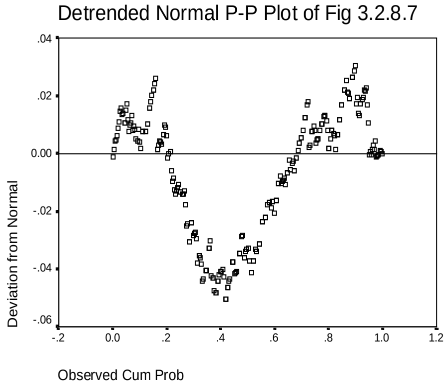
Histogram



Pplot

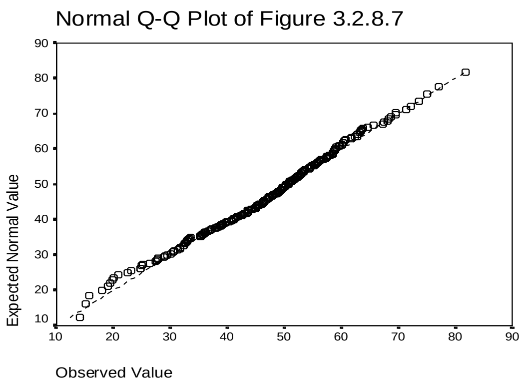
```
MODEL:  MOD_1.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.7 ...
Normal distribution parameters estimated: location = 47.034191 and
scale = 12.225168
```





Qplot

```
MODEL:  MOD_2.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.7 ...
Normal distribution parameters estimated: location = 47.034191 and
scale = 12.225168
```



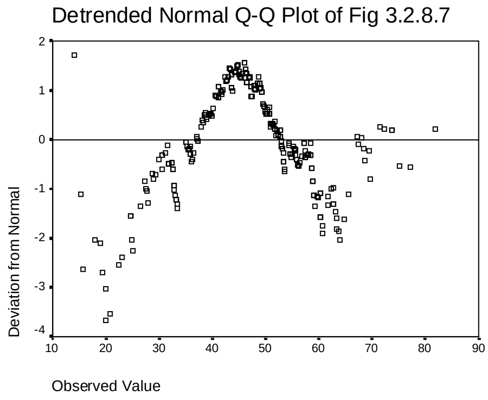


FIGURE 3.8 (3.2.8.8) AND TABLE 3.16

K - S NPar Test Results

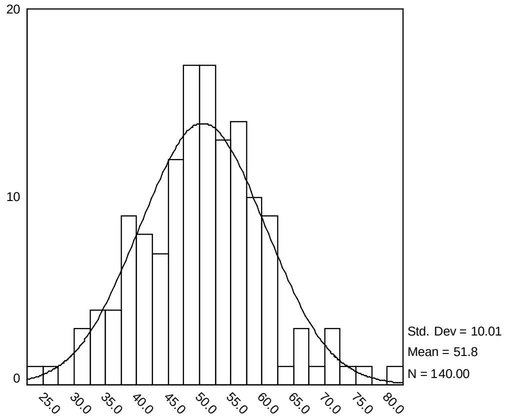
One-Sample Kolmogorov-Smirnov Test

		Figure 3.2.8.8
N		140
Normal Parameters[a,b]	Mean	51.7879
	Std. Deviation	10.01472
Most Extreme Differences	Absolute	.048
	Positive	.048
	Negative	-.042
Kolmogorov-Smirnov Z		.564
Asymp. Sig. (2-tailed)		.908

a. Test distribution is Normal.

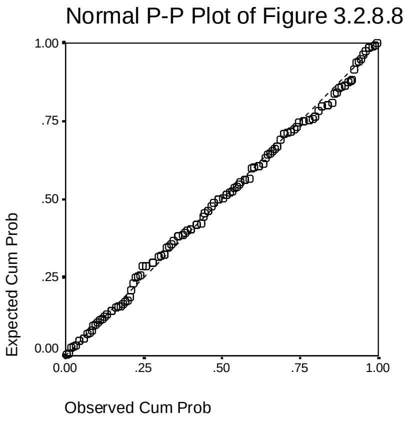
b. Calculated from data.

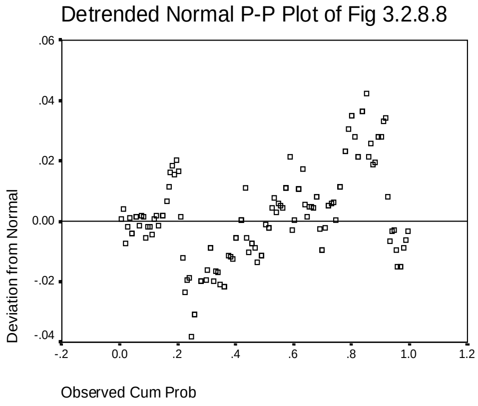
Histogram



Pplot

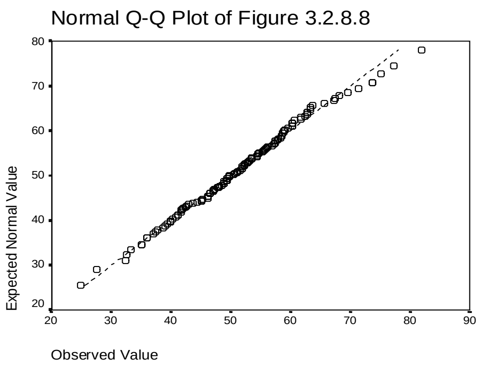
```
MODEL:  MOD_1.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.8 ...
Normal distribution parameters estimated: location = 51.787857 and
scale = 10.014719
```





Qplot

```
MODEL:  MOD_2.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.8 ...
Normal distribution parameters estimated: location = 51.787857 and
scale = 10.014719
```



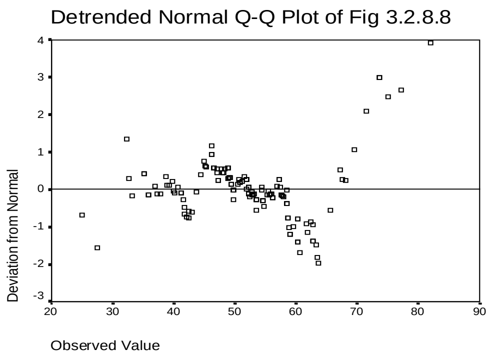


FIGURE 3.9 (3.2.8.9) AND TABLE 3.17

K - S NPar Test Results

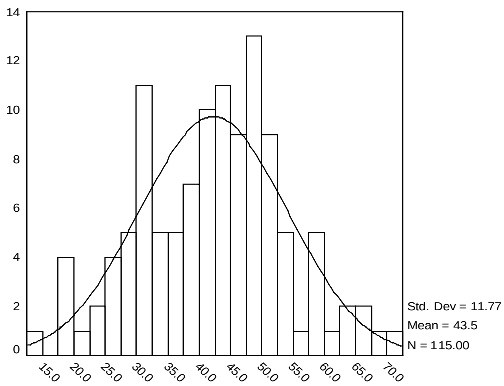
One-Sample Kolmogorov-Smirnov Test

		Figure 3.2.8.9
N		115
Normal Parameters[a,b]	Mean	43.5426
	Std. Deviation	11.76885
Most Extreme Differences	Absolute	.080
	Positive	.054
	Negative	-.080
Kolmogorov-Smirnov Z		.855
Asymp. Sig. (2-tailed)		.458

a. Test distribution is Normal.

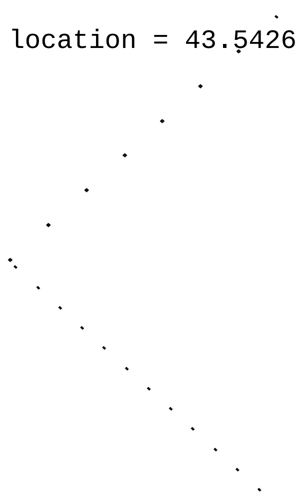
b. Calculated from data.

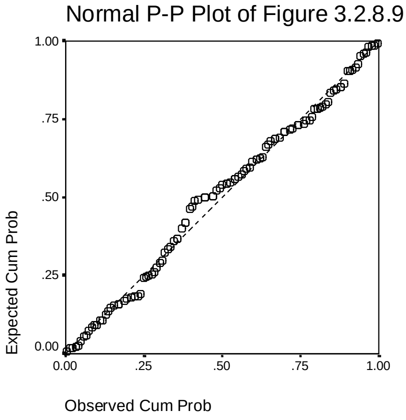
Histogram

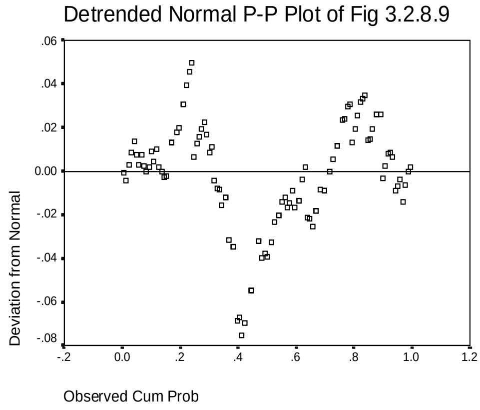


Pplot

```
MODEL:  MOD_1.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.9 ...
Normal distribution parameters estimated: location = 43.542609 and
scale = 11.768849
```

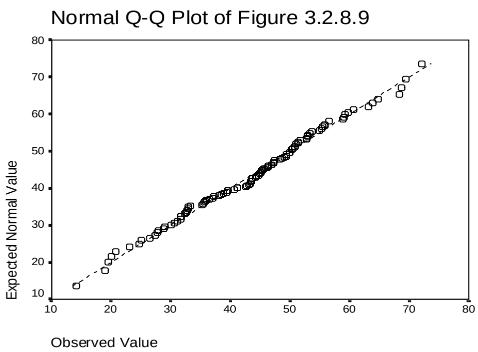






Qplot

```
MODEL:  MOD_2.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.9...
Normal distribution parameters estimated: location = 43.542609 and
scale = 11.768849
```



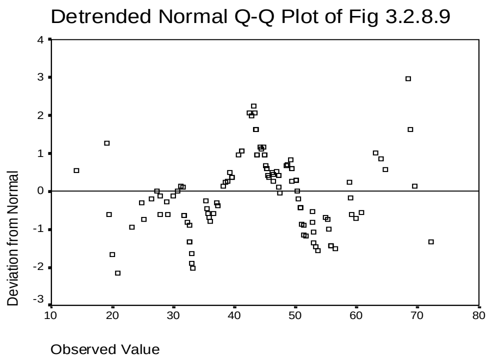


FIGURE 3.12 (3.2.9.1) AND TABLE 3.20

K - S NPar Test Results

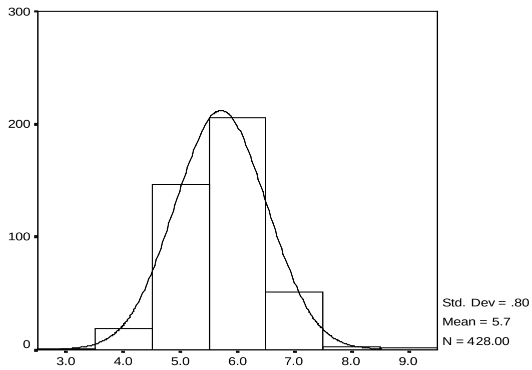
One-Sample Kolmogorov-Smirnov Test

		Figure 3.2.9.1
N		428
Normal Parameters[a,b]	Mean	5.7103
	Std. Deviation	.80408
Most Extreme Differences	Absolute	.253
	Positive	.228
	Negative	-.253
Kolmogorov-Smirnov Z		5.231
Asymp. Sig. (2-tailed)		.000

a. Test distribution is Normal.

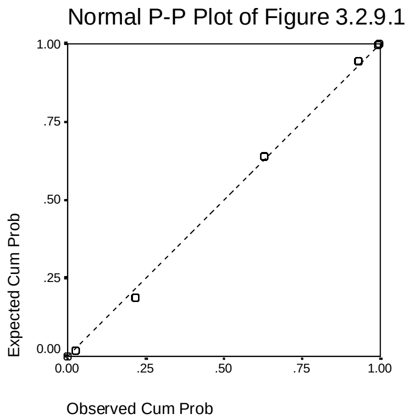
b. Calculated from data.

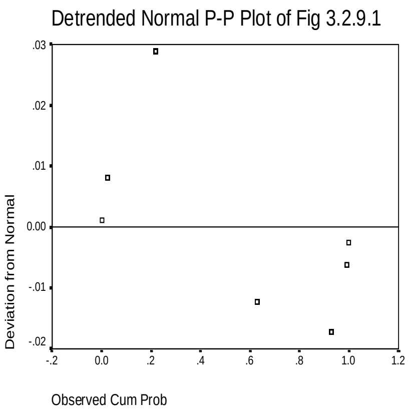
Histogram



Pplot

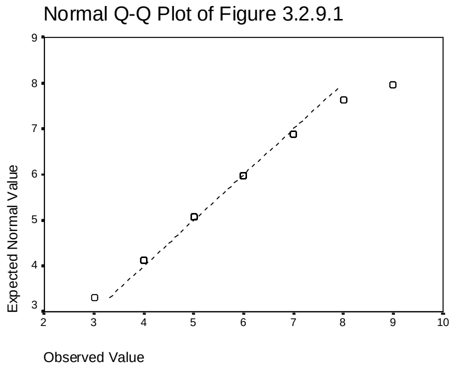
```
MODEL:  MOD_13.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.12 ...
Normal distribution parameters estimated: location = 5.7102804 and
scale = .80408029
```

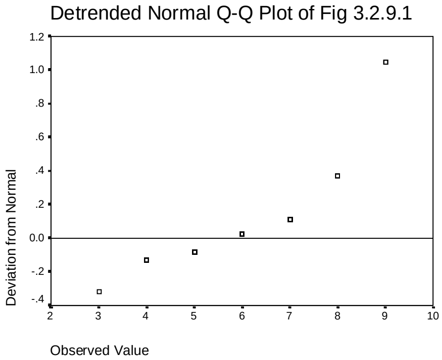




Qplot

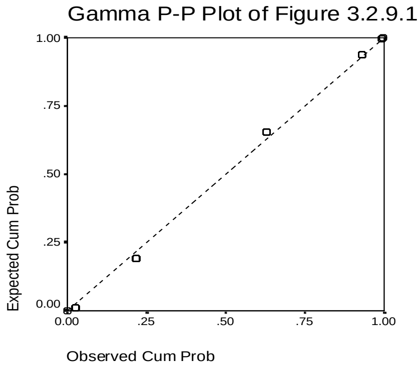
```
MODEL:  MOD_14.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.12 ...
Normal distribution parameters estimated: location = 5.7102804 and
scale = .80408029
```

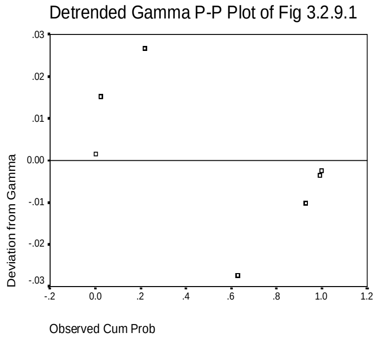




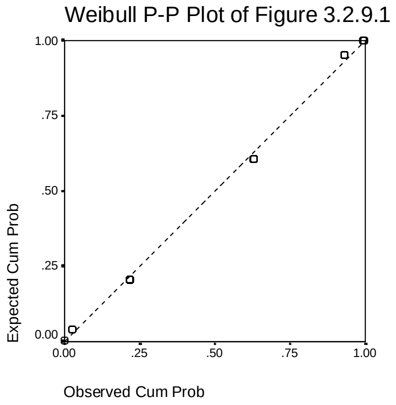
Additional Pplots

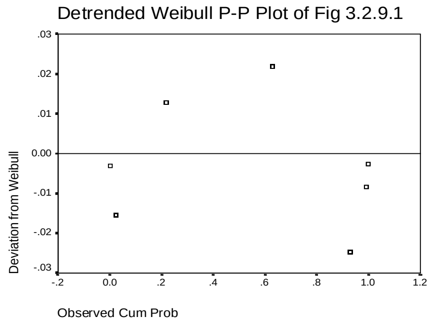
```
MODEL:  MOD_15.
Distribution tested: Gamma
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.12 ...
Gamma distribution parameters estimated: shape = 50.433142 and scale
= 8.8319905
```





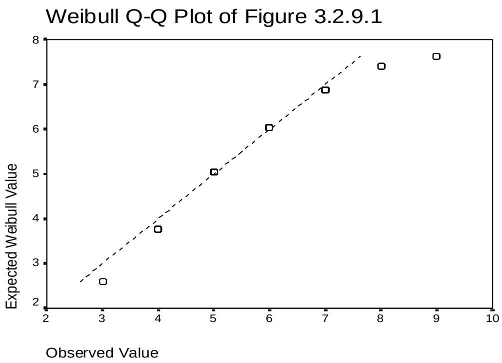
```
MODEL:  MOD_16.
Distribution tested: Weibull
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.12 ...
Weibull distribution parameters estimated: scale = 6.05427 and shape
= 7.7076501
```





Additional Qplot

```
MODEL:  MOD_17.
Distribution tested: Weibull
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.12 ...
Weibull distribution parameters estimated: scale = 6.05427 and shape
= 7.7076501
```



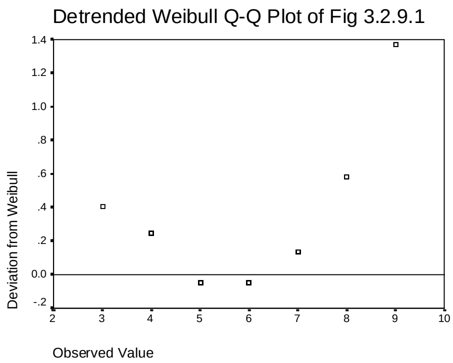


FIGURE 3.18 (3.2.9.7) AND TABLE 3.24

K - S NPar Test Results

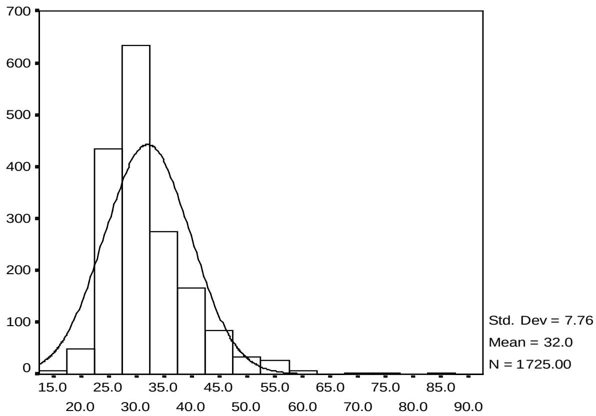
One-Sample Kolmogorov-Smirnov Test

		Figure 3.2.9.7
N		1725
Normal Parameters[a,b]	Mean	32.0400
	Std. Deviation	7.75817
Most Extreme Differences	Absolute	.154
	Positive	.154
	Negative	-.101
Kolmogorov-Smirnov Z		6.382
Asymp. Sig. (2-tailed)		.000

a. Test distribution is Normal.

b. Calculated from data.

Histogram



Additional K - S NPar Tests

One-Sample Kolmogorov-Smirnov Test

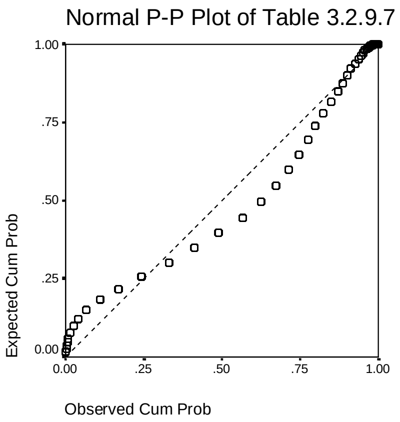
		Figure 3.2.9.7
N		1725
Poisson Parameter[a,b]	Mean	32.0400
Most Extreme Differences	Absolute	.130
	Positive	.130
	Negative	-.056
Kolmogorov-Smirnov Z		5.395
Asymp. Sig. (2-tailed)		.000

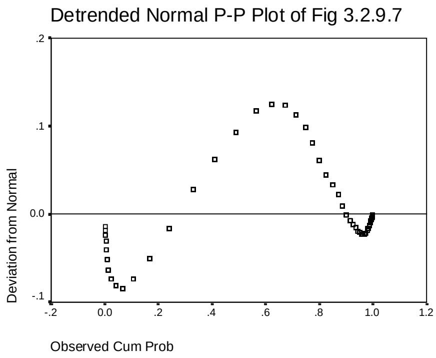
a. Test distribution is Poisson.

b. Calculated from data.

Pplot

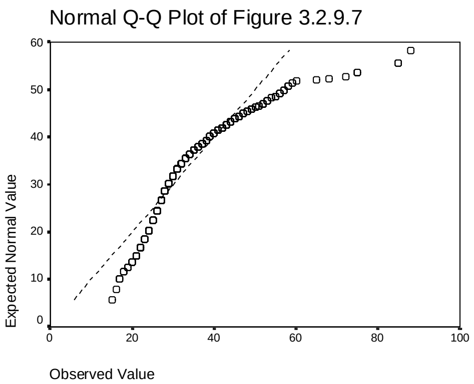
```
MODEL:  MOD_1.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.18 ...
Normal distribution parameters estimated: location = 32.04 and scale
= 7.7581721
```

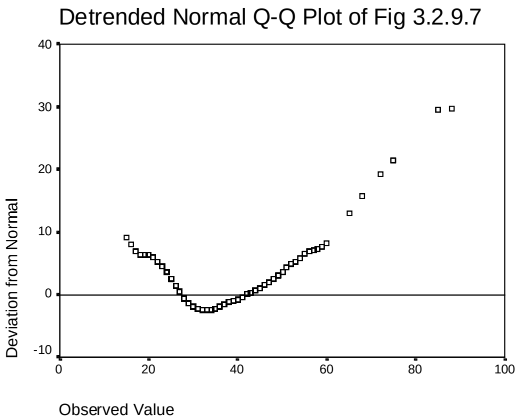




Qplot

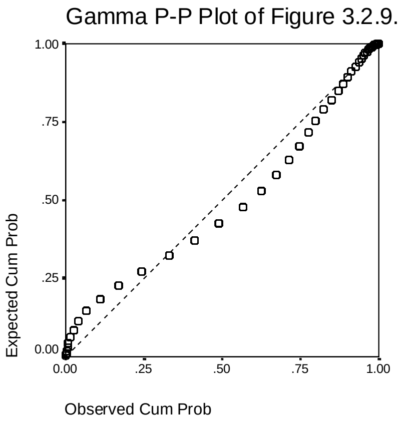
```
MODEL:  MOD_2.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Table 3.25 ...
Normal distribution parameters estimated: location = 32.04 and scale
= 7.7581721
```

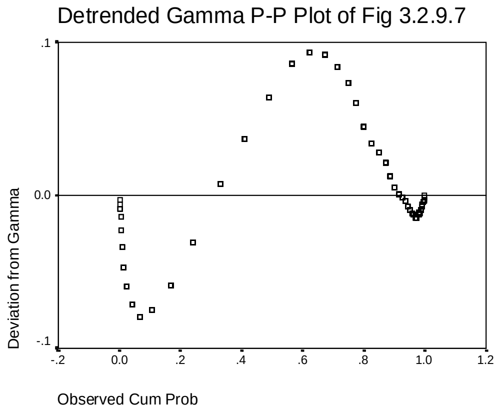




Gamma Pplot

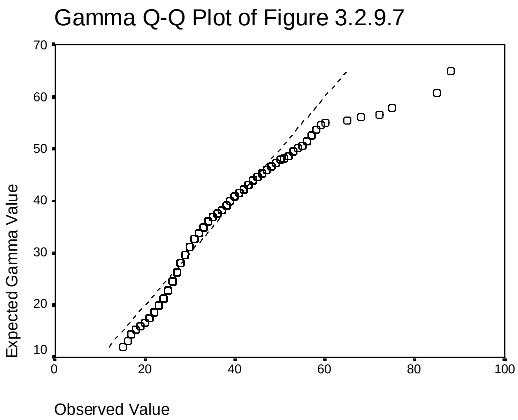
```
MODEL:  MOD_1.
Distribution tested: Gamma
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.18 ...
Gamma distribution parameters estimated: shape = 17.055568 and scale
= .53232111
```





Qplot

```
MODEL:  MOD_3.
Distribution tested: Gamma
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.18 ...
Gamma distribution parameters estimated: shape = 17.055568 and scale
= .53232111
```



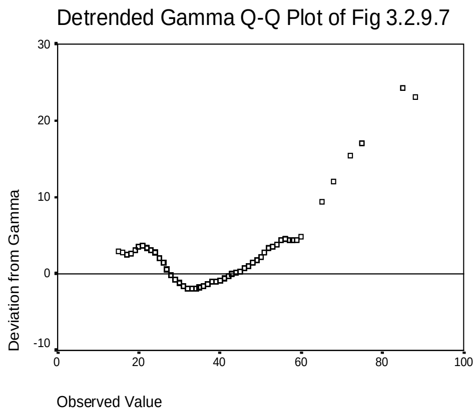


FIGURE 3.19 (3.2.9.8) AND TABLE 3.24

K - S NPar Test Results

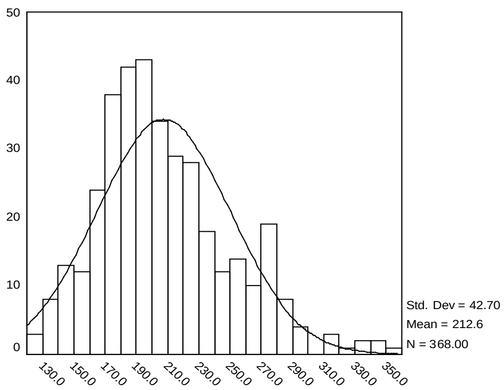
One-Sample Kolmogorov-Smirnov Test

		Figure 3.2.9.8
N		368
Normal Parameters[a,b]	Mean	212.5897
	Std. Deviation	42.70471
Most Extreme Differences	Absolute	.078
	Positive	.078
	Negative	-.040
Kolmogorov-Smirnov Z		1.496
Asymp. Sig. (2-tailed)		.023

a. Test distribution is Normal.

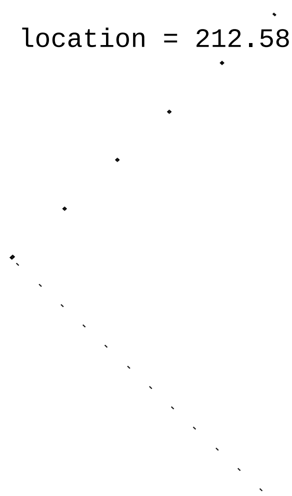
b. Calculated from data.

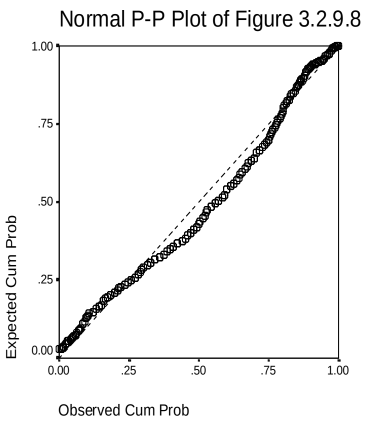
Histogram

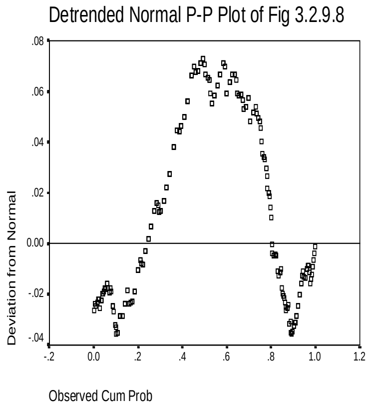


Pplot

MODEL: MOD_2.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.19 ...
Normal distribution parameters estimated: location = 212.58967 and scale = 42.704709

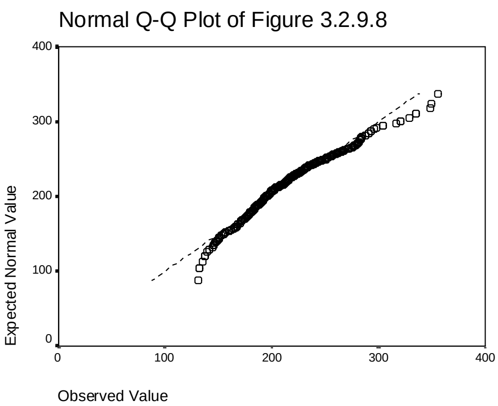


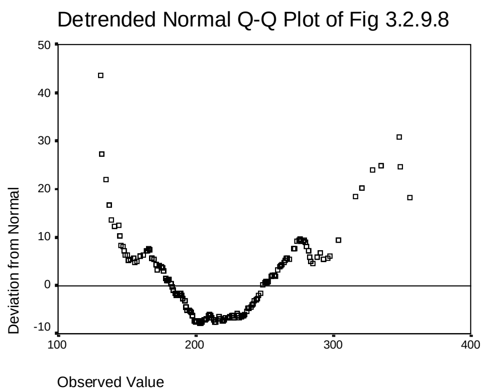




Q Plot

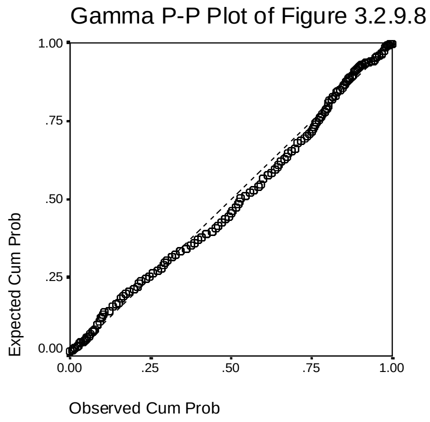
```
MODEL:  MOD_3.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.19 ...
Normal distribution parameters estimated: location = 212.58967 and
scale = 42.704709
```

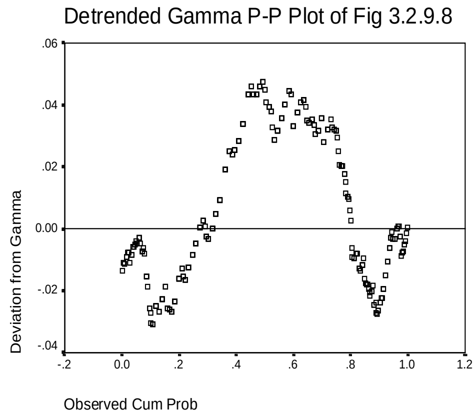




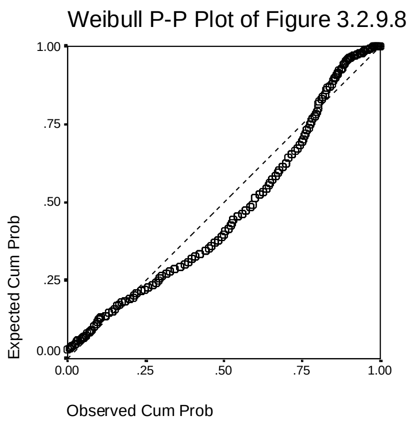
Additional Pplots

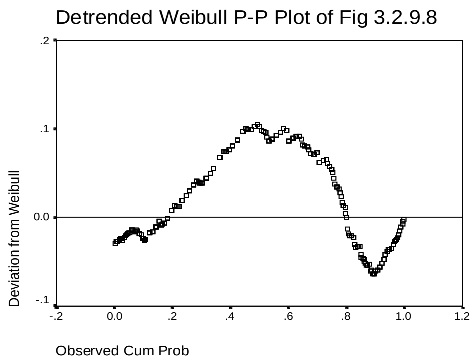
```
MODEL:  MOD_1.
Distribution tested: Gamma
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.19 ...
Gamma distribution parameters estimated: shape = 24.781797 and scale
= .11657103
```





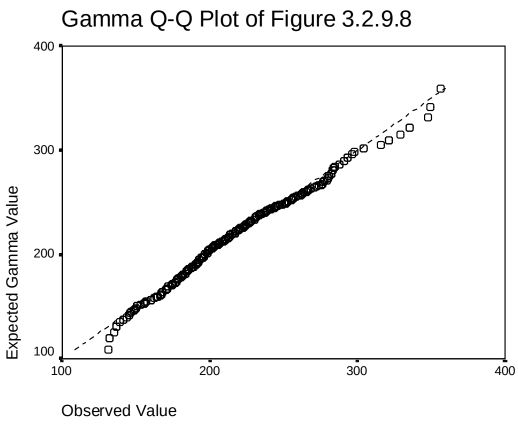

```
MODEL:  MOD_2.
Distribution tested: Weibull
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.19 ...
Weibull distribution parameters estimated: scale = 228.73259 and
shape = 6.2037825
```





Additional Qplot

```
MODEL:  MOD_3.
Distribution tested: Gamma
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.19 ...
Gamma distribution parameters estimated: shape = 24.781797 and scale
= .11657103
```


Gamma Q-Q Plot of Figure 3.2.9.8
Expected Gamma Value
Observed Value
400
300
200
100
100
200
300
400

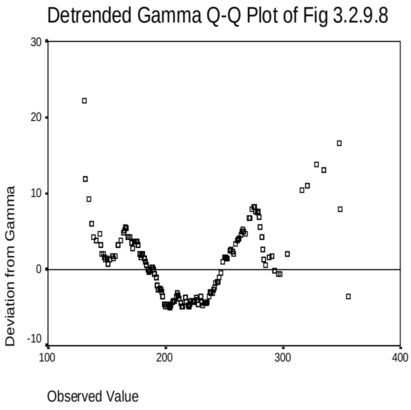

Detrended Gamma Q-Q Plot of Fig 3.2.9.8
Deviation from Gamma
Observed Value
30
20
10
0
-10
100
200
300
400

FIGURE 3.24 (3.2.9.13) AND TABLE 3.25

K - S NPar Test Results

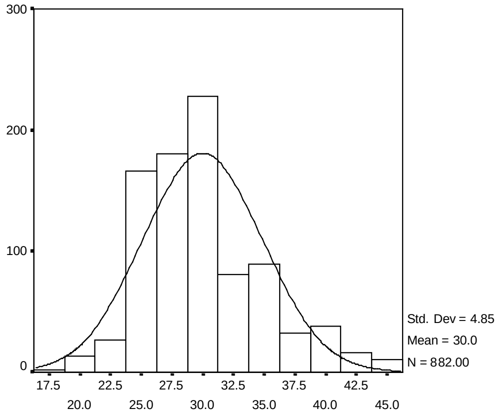
One-Sample Kolmogorov-Smirnov Test 1

		Figure 3.2.9.13
N		882
Normal Parameters[a,b]	Mean	29.9796
	Std. Deviation	4.85425
Most Extreme Differences	Absolute	.140
	Positive	.140
	Negative	-.071
Kolmogorov-Smirnov Z		4.159
Asymp. Sig. (2-tailed)		.000

a. Test distribution is Normal.

b. Calculated from data.

Histogram



Additional K –S Npar Tests

One-Sample Kolmogorov-Smirnov Test 2

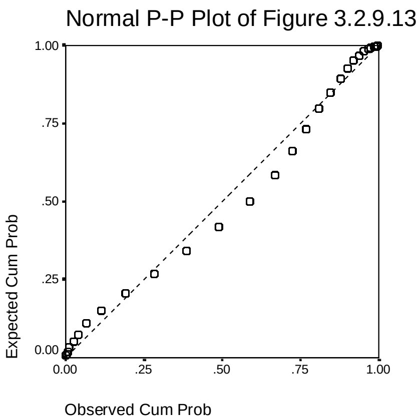
		Figure 3.2.9.13
N		882
Poisson Parameter[a,b]	Mean	29.9796
Most Extreme Differences	Absolute	.092
	Positive	.092
	Negative	-.076
Kolmogorov-Smirnov Z		2.729
Asymp. Sig. (2-tailed)		.000

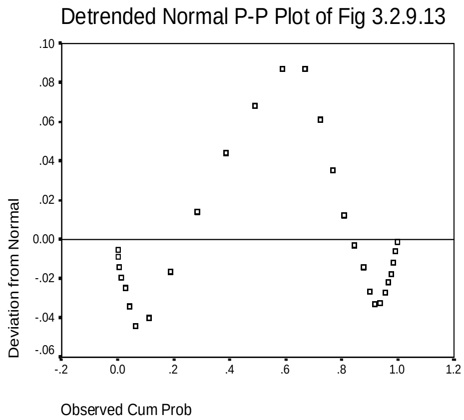
a. Test distribution is Poisson.

b. Calculated from data.

Pplot

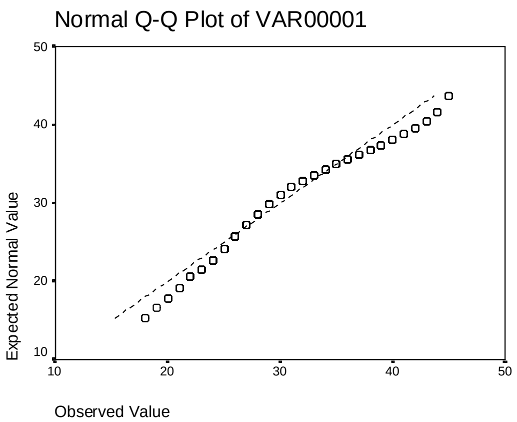
```
MODEL:  MOD_1.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.24 ...
Normal distribution parameters estimated: location = 29.979592 and
scale = 4.8542471
```

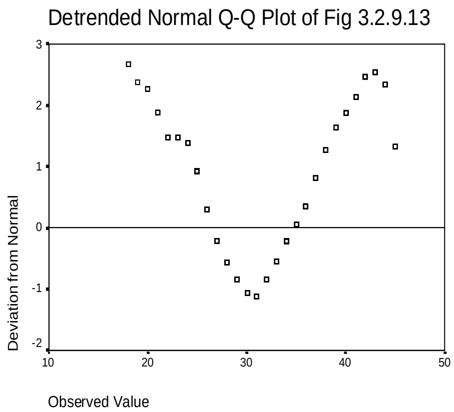




QPlot

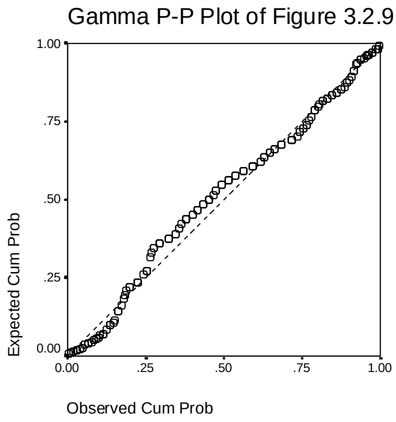
```
MODEL:  MOD_2.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.24 ...
Normal distribution parameters estimated: location = 29.979592 and
scale = 4.8542471
```

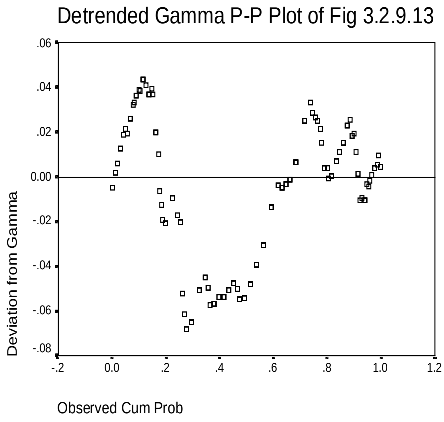




Gamma Pplot

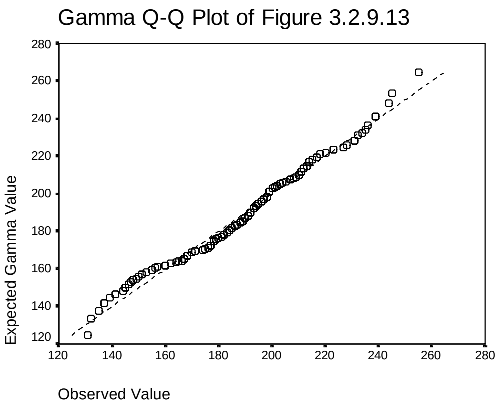
```
MODEL:  MOD_1.
Distribution tested: Gamma
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.24 ...
Gamma distribution parameters estimated: shape = 53.066151 and scale
= .28350203
```





QPlot

```
MODEL:  MOD_2.
Distribution tested: Gamma
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.24 ...
Gamma distribution parameters estimated: shape = 53.066151 and scale
= .28350203
```



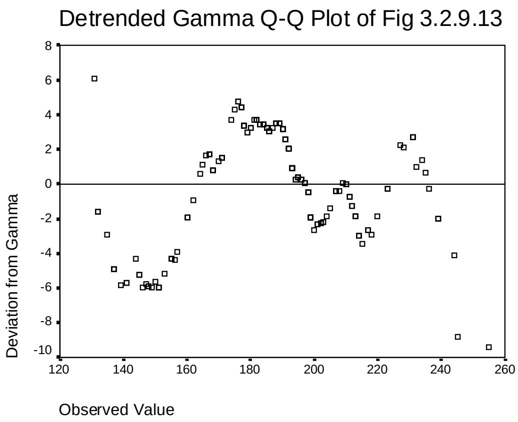


FIGURE 3.25 (3.2.9.14) AND TABLE 3.25

K - S NPar Test Results

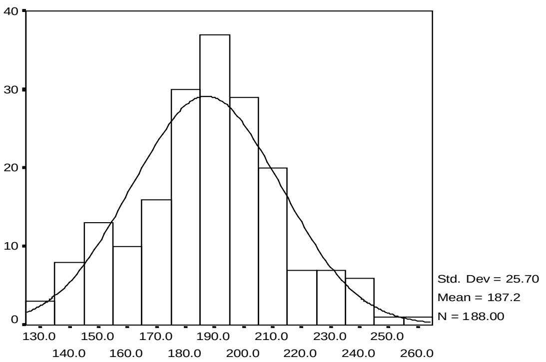
One-Sample Kolmogorov-Smirnov Test

		Figure 3.2.9.14
N		188
Normal Parameters[a,b]	Mean	187.1809
	Std. Deviation	25.69524
Most Extreme Differences	Absolute	.064
	Positive	.057
	Negative	-.064
Kolmogorov-Smirnov Z		.878
Asymp. Sig. (2-tailed)		.423

a. Test distribution is Normal.

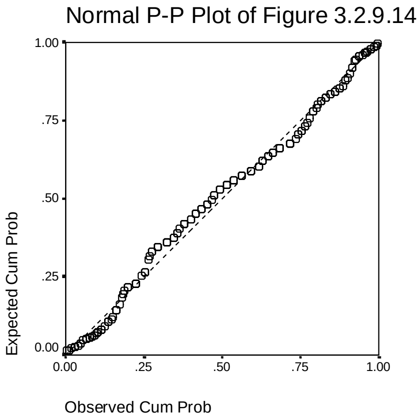
b. Calculated from data.

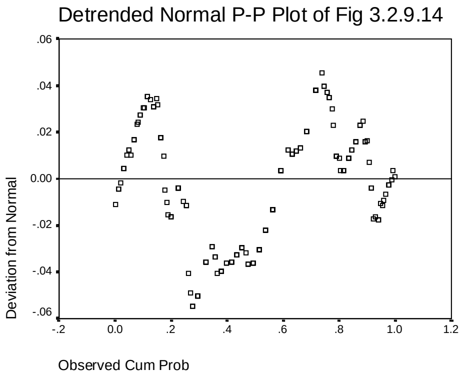
Histogram



Pplot

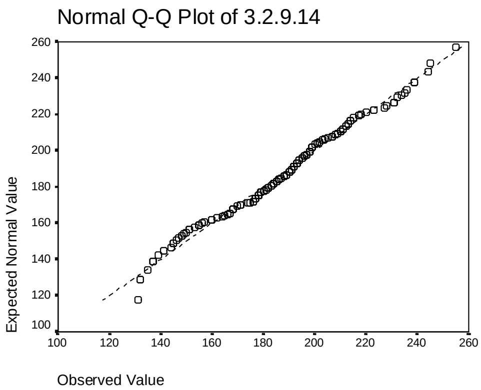
```
MODEL:  MOD_1.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Figure 3.25 ...
Normal distribution parameters estimated: location = 187.18085 and
scale = 25.695237
```





Q Plot

```
MODEL:  MOD_2.
Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
For variable Table 3.2.9.14 ...
Normal distribution parameters estimated: location = 187.18085 and
scale = 25.695237
```



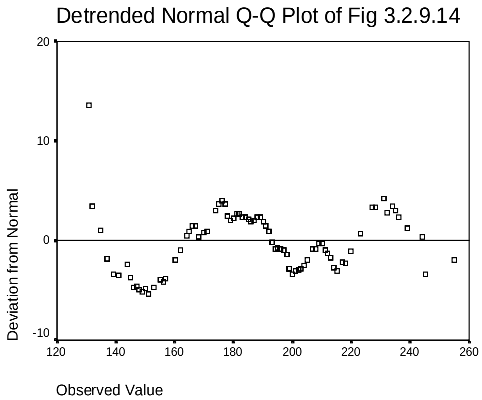


Table 3.53 (Table 3.3.9.1)

K - S NPar Test Results

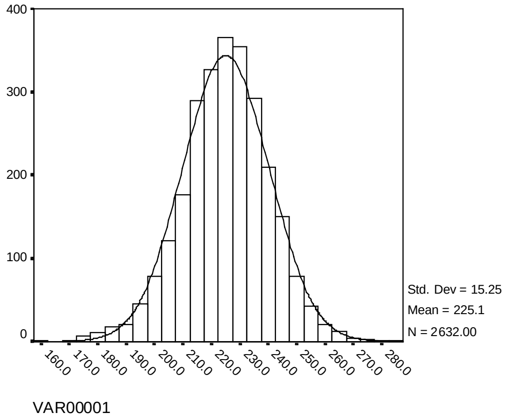
One-Sample Kolmogorov-Smirnov Test

		Table 3.3.9.1
N		2632
Normal Parameters[a,b]	Mean	225.0938
	Std. Deviation	15.24664
Most Extreme Differences	Absolute	.023
	Positive	.013
	Negative	-.023
Kolmogorov-Smirnov Z		1.201
Asymp. Sig. (2-tailed)		.112

a. Test distribution is Normal.

b. Calculated from data.

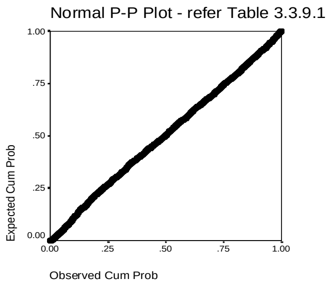
Graph

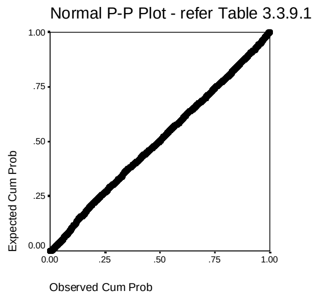


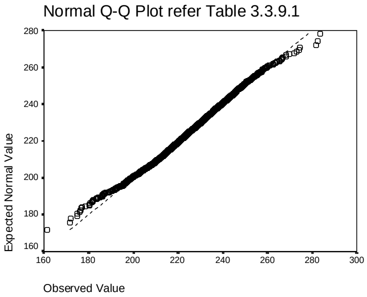
PPlot

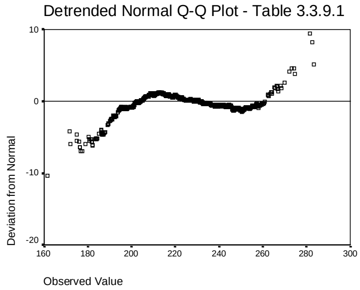
```
MODEL:  MOD_1.
```

Distribution tested: Normal
Proportion estimation formula used: Blom's
Rank assigned to ties: Mean
_For variable VAR00001 ...Table 3.53 refers
Normal distribution parameters estimated: location = 225.09377 and scale = 15.246638






Normal Q-Q Plot refer Table 3.3.9.1
Expected Normal Value
Observed Value
160
180
200
220
240
260
280
300


Detrended Normal Q-Q Plot - Table 3.3.9.1
Deviation from Normal
Observed Value
10
0
-10
-20
160
180
200
220
240
260
280
300